K92 MINING INC.

Independent Technical Report, Kainantu Gold Mine, Updated Integrated Development Plan,

Kainantu Project, Papua New Guinea

Definitive Feasibility Study and Preliminary Economic Analysis

National Instrument 43-101 Technical Report

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***Notice to Reader:** This amended and updated technical report replaces the technical report titled, "Independent Technical Report, Kainantu Gold Mine, Updated Integrated Development Plan, Kainantu Project, Papua New Guinea" filed on November 28, 2024. This report reflects formatting changes in Table 1-2 on page 8, Table 14-36 on page 263, and other minor formatting improvements. Please note the numerical data remains unchanged.

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1. SUMMARY

1.1 Introduction

This report is an Independent Technical Report dated 1 January 2024 of the Updated Integrated Development Plan (IDP) for K92 Mining Inc.'s Kainantu Gold Mine Project (the 'Kainantu Project') in Papua New Guinea. The Updated IDP includes the Kainantu Stage 3 Expansion Definitive Feasibility Study (DFS) Case and the alternative Kainantu Stage 4 Expansion Preliminary Economic Analysis (PEA) Case.

The Updated IDP was prepared by K92 Mining Ltd ('K92ML'); Entech Pty Ltd of Perth, Australia, ('Entech'); ATC Williams Pty Ltd of Brisbane, Australia ('ATC Williams'); WSP Engineering Ltd of Ontario, Canada ('WSP'); Metallurgical Management Services Pty Ltd of Perth, Australia ('MMS'); GR Engineering Services Ltd of Brisbane, Australia ('GRES); EMM Consulting of Brisbane, Australia ('EMM'); and H & S Consultants Pty Ltd of Sydney, Australia ('H&SC').

The Kainantu property covers a total area of 836.8 km^2 and is in the Eastern Highlands Province of Papua New Guinea, approximately 180 km west-northwest of Lae. Figure 1-1 shows the Kainantu Project Location and Tenements.

The Property lies within an area of mostly rugged topography, with transecting rivers forming lower lying areas. Elevations range from 400 m to 3,130 m above sea level. The processing plant infrastructure and the majority of the surface support infrastructure is located in flat lying topography, within the Markham Valley. Vegetation is mostly primary rainforest with areas of shifting cultivation in valley floors.

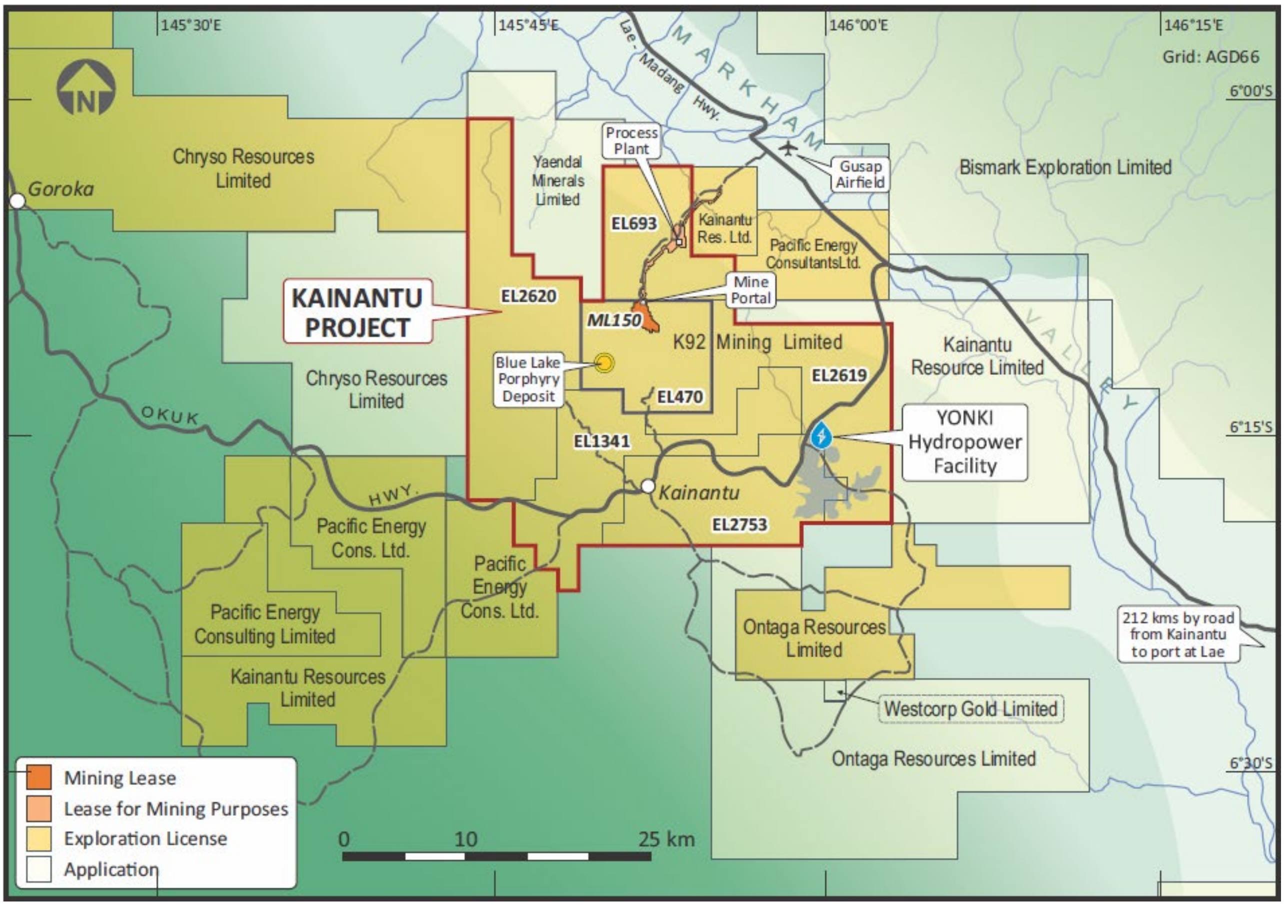


Figure 1-1 Kainantu Project Location and Tenements

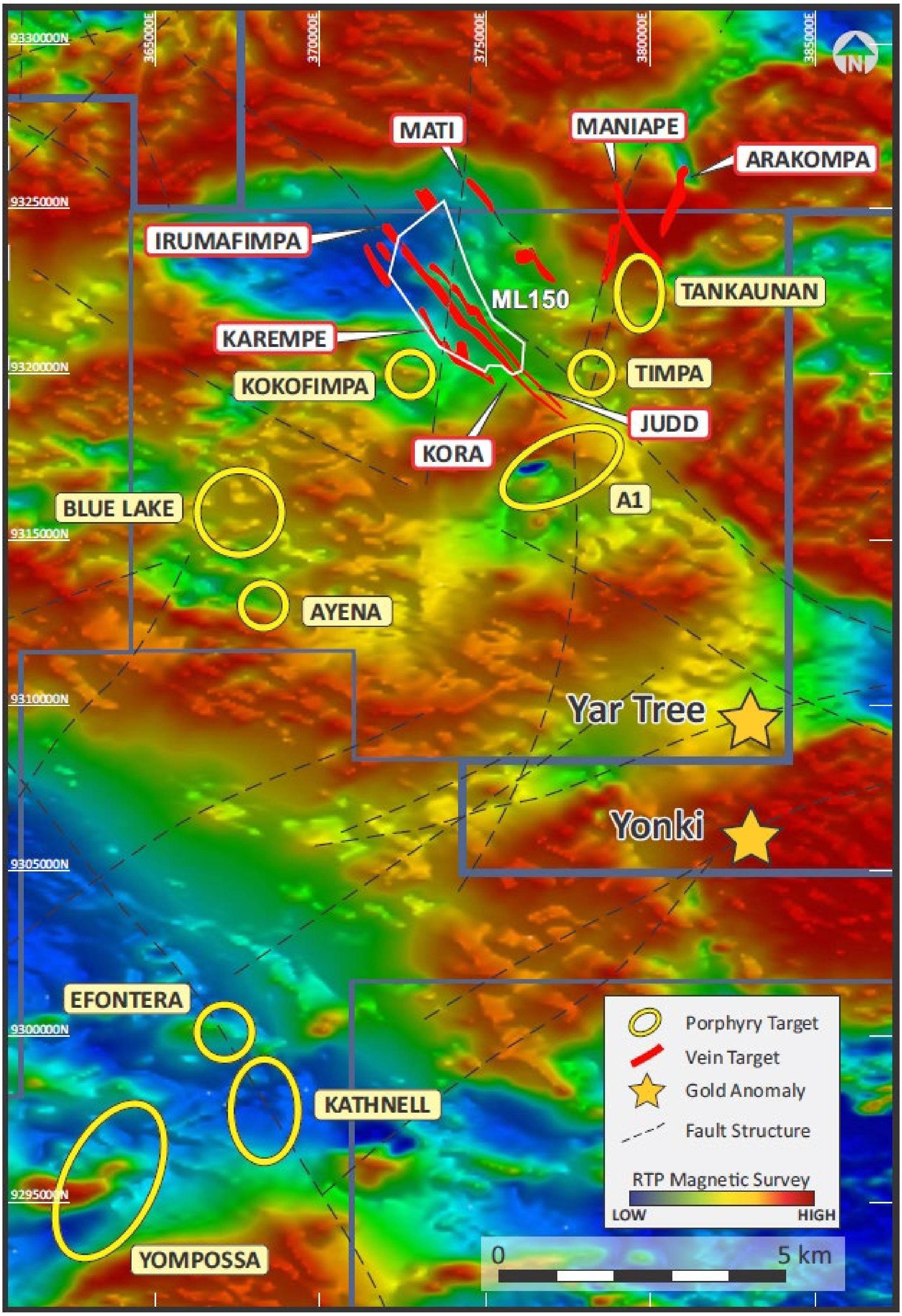
1.2 Geological Setting and Mineralization

The Kainantu region is in the north-eastern flank of the northwest trending Papuan Mobile Belt which is a major foreland thrust belt. The regional structural package of the Kainantu district is bounded in the northeast by the northwest trending Ramu-Markham Fault, a major suture zone that marks the northern margin of the Australian craton, and in the southeast by the Aure Deformation Zone. The belt is characterized by several north-northeast trending fault zones that commonly host major ore deposits.

The dominant host rock of the Kora - Judd - Irumafimpa vein systems (Kora was previously termed Kora Consolidated and made up of Kora, Kora North and Eutompi in earlier documents) is the highly sheared and deformed Bena Bena Formation, composed of low grade metamorphosed phyllites and amphibolites, intruded by Elandora type dacitic intrusives, marginal to the vein system.

Mineralization on the property includes gold, silver and copper in epithermal gold-telluride veins (Irumafimpa), gold, copper and silver sulphide veins of Intrusion Related Gold Copper ('IRGC') affinity (Kora and Judd) and also, less explored porphyry copper-gold systems (Blue Lake), and alluvial gold. The Kora vein systems have been demonstrated from K92ML's drilling and surface mapping results to be a continuous mineralized structure. This mineralized structure occurs in the centre of a large, mineralized complex approximately 5 km x 5 km, in which drilling has identified several individual zones of IRGC and porphyry style mineralization.

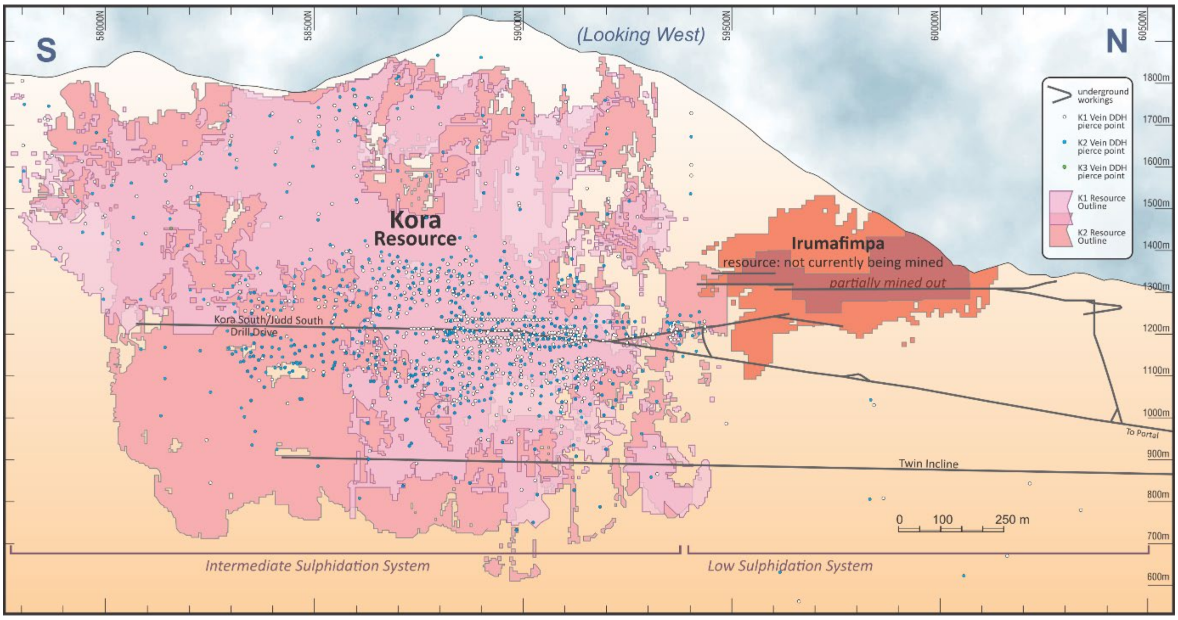
The current Mineral Resources for Kora and Irumafimpa occupy a broad northwest trending mineralized zone more than 2.5 km long and approximately 60 to 80 m wide with down dip continuity of over 1,000 m. The Kora mineralized zone comprises a series of individual veins and stringer vein systems named from west to east, as K2, Kora Link and K1. The parallel Judd vein system, which is located 90 to 150 m east of Kora, also comprises multiple veins, from J1W and J1 in the west to J3 in the east, although the J1W and J3 veins are yet to be thoroughly defined by drilling. The total width across the Judd vein system from J1W to J3 is between 60 to 80m. All of the vein lodes are composed of quartz-sulphide veins that vary in width throughout the vein packages from <1 m pinch and swell structures at Irumafimpa to veins up to 10 m wide at Kora. Figure 1-2 below shows the main vein systems and porphyry targets identified to date at the Kainantu project.



Source: K92ML (2024)

Figure 1-2 Kainantu Property Geology and Known Vein and Porphyry Deposits and Prospects

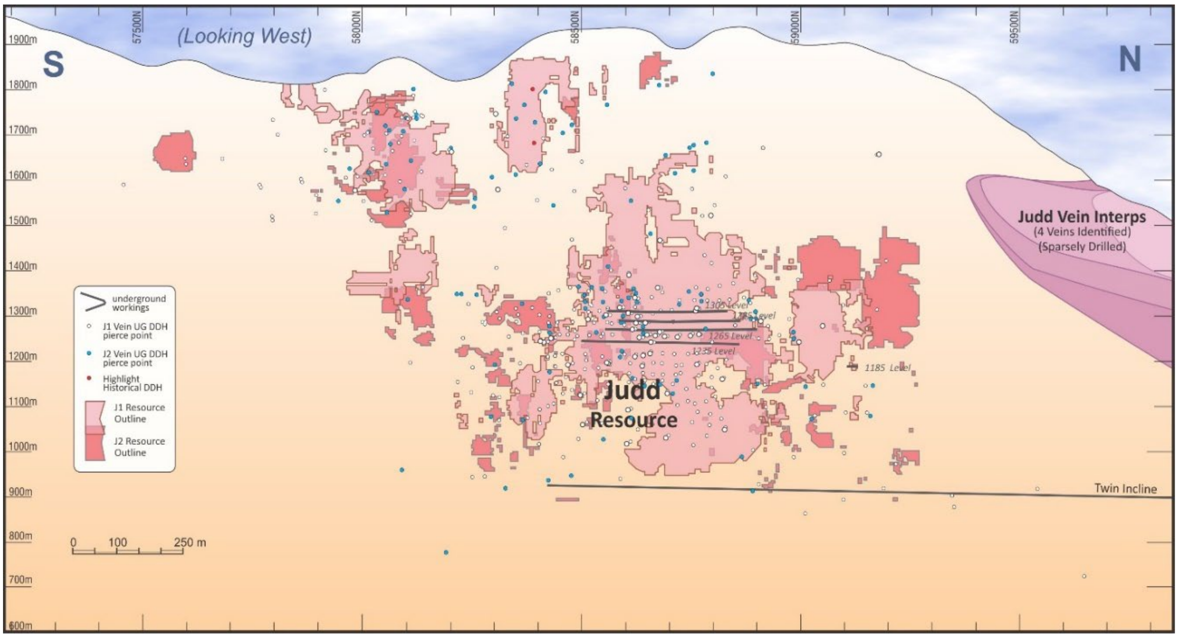
At Kora and to the northwest along strike at Irumafimpa, two stages of mineralization have been recognized in the same northwest-trending sub-vertical structure. There is an early copper sulphide dominant stage of mostly chalcopyrite overprinted by a later, quartz, pyrite dominant stage of mineralization including high grade gold associated with tellurides. The tellurides at Kora are minor and do not hamper processing, the gold and copper mineralization is significant and generates a significant revenue for the mine. The Kora deposit currently comprises two parallel, steeply west dipping, north south striking quartz-sulphide vein systems, K1 and K2. An additional structure, the Kora Link, has also been defined and provides a possible link between the two main vein systems. Drilling has confirmed that the overall system has a vertical extent greater than 1,000 m, (Figure 1-3).



Source: K92ML (2024)

Figure 1-3 Kora and Irumafimpa Longitudinal Section

The Judd vein system has been defined by drilling and the mineralization style is similar to Kora. J1 and J2 are displayed in Figure 1-4, note the Judd Vein interpretations to the north in the long section are the geologist interpretation based on sparse data.



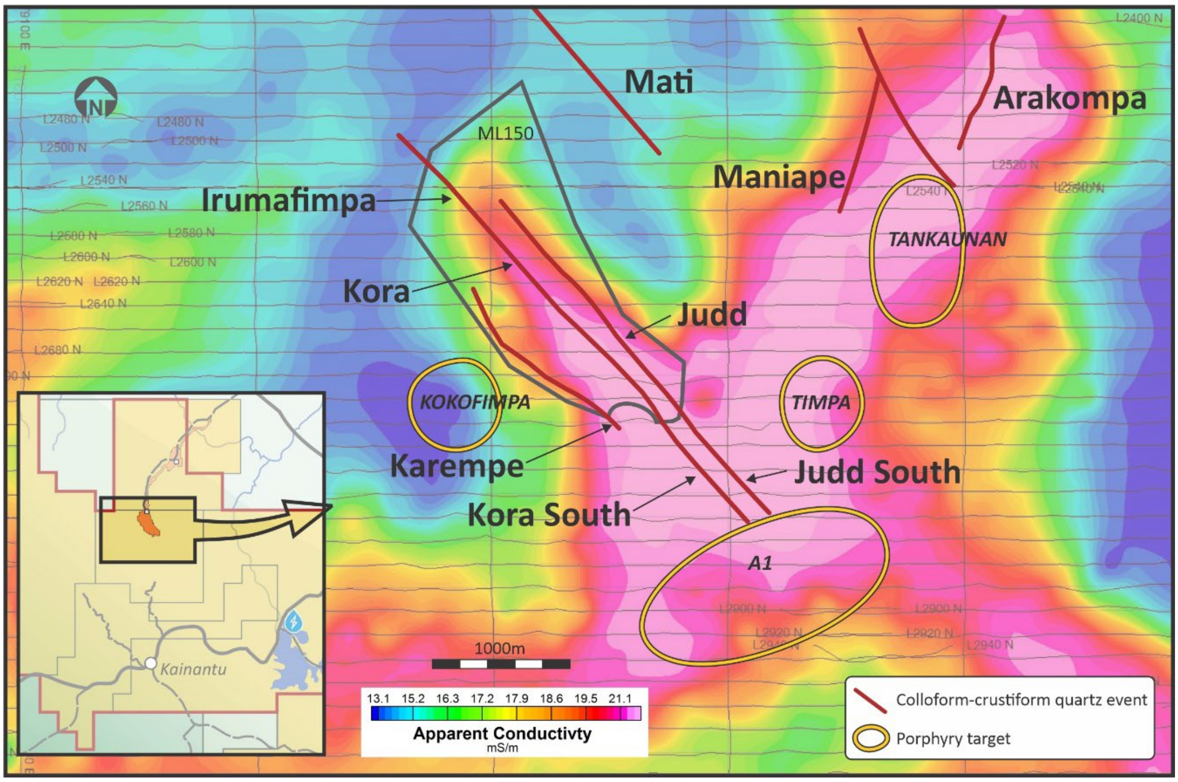
Source: K92ML (2024)

Figure 1-4 Judd Longitudinal Section

1.3 Exploration

The Kainantu Project is recognised as an important mineral district, owing to the presence of multiple economic vein deposits, as well as additional veins and porphyry prospects, at various stages of exploration. K92ML has a very substantial land package of exploration tenement totalling 836 km^2.

An airborne geophysical (Magnetotellurics or MobileMT) survey was completed in 2021 over the entire area of K92ML's tenements shown in Figure 1-5. Numerous conductive targets were identified; and, where previously drill tested, conform closely with known deposits and prospects, both vein and porphyry occurrences.



Source: K92ML (2021)

Figure 1-5 Judd, Kora and Irumafimpa Vein Systems with Mobile MT Contours

1.4 Minerals Processing and Metallurgical Testing

1.4.1 Process Plant

The 2021 testwork evaluated samples from K92ML plant operations, taken during separate campaigns on K1, K2, and Judd ores. Ore for these campaigns was supplied from a combination of mining and development activities. Additional testwork was performed on a series of variability samples from exploration drilling. This report gives a brief review of the testwork results, focussing on important aspects of the treatment response of K92ML ores.

Reference should be made to the 2022 IDP document for comprehensive details and interpretation of the testwork results. This document is available from https://k92mining.com/documents/technical-reports/.

For the 2022 IDP, plant performance data from the period January 2021 to March 2022 was used in conjunction with the testwork results to predict future plant performance. This current report considers plant performance over the period from January 2018 to June 2024 for the performance prediction. This longer interval covers changes and additions to the processing plant flowsheet during the Stage 2 (400k tpa) and 2A (500k tpa) expansions and their effect on performance and throughput capability.

1.4.2 Paste Plant

For the paste system design a range of processes, rheology and geomechanical tests were completed on Kainantu tailings. This testwork campaign comprised:

- Characterization testing, including:
  - Tailings mineralogy, particle size distribution, and specific gravity;
  - Process water chemical analysis - Water Quality Analysis (WQA) testing;
  - Proctor and Transportable Moisture Limit Testing;
  - Scanning Electron Microscope (SEM) testing;
  - Filter cake bin storage and mass flow characterization;
  - Filter cake bulk density.
- Filtration testing, including:
  - Cloth disc vacuum filtration testing;
  - Pressure Filtration testing using a bench-scale unit with different filter cloths and chamber thicknesses.
- Rheology testing including:
  - Bench scale yield vane rheometer testing;
  - Bench scale viscosity bob and cup rheometer testing;
  - Laboratory scale flow loop testing;
  - Water retention testing at different solid consistencies on uncemented paste material;
- Cemented strength testing include only unconfined compressive strength testing.

Paste mix designs included in this work considered the influence of binder type, binder quantity, solid consistencies, water sources, and curing times.

Results from this testwork were used for the purpose of selecting the optimal paste system arrangement as well as supporting the design of all relevant components.

1.5 Mineral Resource Estimates

The new Mineral Resource Estimate (MRE) for the Kora and Judd deposits are reported in Table 1-1 for a 3 g/t gold equivalent cut off respectively. The 3 g/t AuEq cut off was supplied by K92ML and is based on mine design studies incorporating previous mining. The constraints for the K1 and Kora Link MRE are for an uncut gold grade for block centroids inside the K1 and Kora Link mineral wireframes with mined depletion removed. The constraints for the K2 MRE are a cut gold grade (to 1,000 g/t) for block centroids inside the K2 mineral wireframe with mined depletion removed. The MRE for the Judd lodes are reported using block centroids inside the relevant lode wireframes but with a cut gold grade to the data of 400 g/t.

The Mineral Resources reported in this section have been classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves prepared by the CIM Standing Committee on Reserve Definitions.

The effective date of the MRE for the Kora and Judd lodes is the 12th of September 2023, which was the date that the latest database was received by H&SC.

The updated MRE for the Kora and Judd Lode systems are based on samples from diamond drillholes (surface and underground) and face sampling of development drives.

Mineral wireframes for the different lodes were interpreted from a combination of drillhole geology and assay grades. The narrow sub-vertical vein systems had a nominal minimum mining width of 5.2 m. 1 m sample composites were extracted from the drillhole database using the wireframes and subsequently modelled using Ordinary Kriging. Gold, copper and silver were modelled along with density. Minimal top cuts to the gold data were required. Classification of the Mineral Resources is primarily based on the drillhole/data point distribution in consideration of other factors e.g. the geological model, QAQC data, variography, sample recovery and density data. The resulting resource models reconciled well with production.

K92ML requested reporting of a gold equivalent g/t (AuEq) to include copper and silver credits using the formula:

$$AuEq = Au\ g/t + (Cu\ \%*1.6481) + (Ag\ g/t*0.0114)$$

Assumptions include:

- Gold prices of USD $1,700/oz: Silver USD $22.5/oz; Copper USD $4.0/lb.
- Recoveries relative to gold of 93% for copper and 80% for silver.

A gold equivalent grade of 3 g/t was used to report the Mineral Resources as block centroids from within the relevant mineral wireframes.

Table 1-1 Mineral Resources for the Kora & Judd Deposits (Effective Date 12 September 2023)

	Tonnes	Gold		Silver		Copper		AuEq	
	Mt	g/t	Mozs	g/t	Mozs	%	kt	g/t	Mozs
Kora									
Measured	3.7	8.74	1.0	20.5	2.5	1.21	45.0	10.96	1.3
Indicated	3.1	6.99	0.7	21.9	2.2	1.31	41.3	9.40	1.0
Total M&I	6.9	7.94	1.8	21.1	4.7	1.25	86.2	10.24	2.3
Inferred	14.3	5.60	2.6	28.7	13.2	1.62	231.2	8.60	3.9
Judd									
Measured	0.4	9.05	0.12	19.0	0.25	0.80	3.2	10.58	0.14
Indicated	0.8	6.37	0.17	15.6	0.42	0.73	6.2	7.76	0.21
Total M&I	1.2	7.24	0.29	16.7	0.67	0.75	9.4	8.68	0.35
Inferred	2.3	6.27	0.45	15.8	1.15	0.76	17.2	7.72	0.56
Kora and Judd									
Measured	4.1	8.77	1.2	20.4	2.7	1.17	48.2	10.92	1.5
Indicated	4.0	6.86	0.9	20.6	2.6	1.19	47.4	9.05	1.2
Total M&I	8.1	7.83	2.0	20.5	5.3	1.18	95.6	10.00	2.6
Inferred	16.5	5.69	3.0	27.0	14.3	1.50	248.3	8.48	4.5

(minor rounding errors)

The key to the confidence of the resource estimates is the apparent good reconciliation of the block model with the mill feed in an area of very high gold grades. This would strongly support the methodologies used for the resource modelling, in particular the geological interpretation, the composite interval, the variography, the top cutting practices, the search parameters and the relatively small block size.

1.6 Mineral Reserve Estimates

The Mineral Reserve estimate outlined in the DFS was prepared by Daniel Donald FAusIMM MSME of Entech, in accordance with the classification criteria set out in the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves. Daniel Donald is an independent consultant of the Company and is a Qualified Person defined by NI 43-101.

The total Mineral Reserve for the Kainantu Project is shown in Table 1-2. The Mineral Reserve estimate is based on the Global Kora and Judd Mineral Resource estimate (12 September 2023 effective date - refer to Table 1-3), net of post-resource mining depletion from 12 September 2023 to 31 December 2023, of 183,768 tonnes at 8.1 g/t Au, 0.9% Cu and 15 g/t Ag.

Table 1-2 Kainantu Mineral Reserve Statement (Effective Date 1 January 2024)

	Tonnes	Gold		Silver		Copper		AuEq	
	Mt	g/t	Moz	g/t	Moz	%	kt	g/t	Moz
Kora									
Proven	2.95	7.41	0.70	19.48	1.85	1.06	31.31	9.39	0.89
Probable	2.52	5.65	0.46	19.23	1.56	1.03	25.89	7.58	0.61
Total P&P	**5.47**	**6.60**	**1.16**	**19.37**	**3.41**	**1.05**	**57.20**	**8.55**	**1.50**
Judd									
Proven	0.24	8.26	0.06	16.53	0.13	0.56	1.33	9.40	0.07
Probable	0.47	6.48	0.10	13.15	0.20	0.52	2.45	7.50	0.11
Total P&P	**0.71**	**7.08**	**0.16**	**14.28**	**0.32**	**0.54**	**3.79**	**8.14**	**0.18**
Kora and Judd									
Proven	3.19	7.47	0.77	19.27	1.98	1.02	32.64	9.39	0.96
Probable	2.99	5.78	0.56	18.28	1.75	0.95	28.34	7.57	0.73
Total P&P	**6.18**	**6.66**	**1.32**	**18.79**	**3.73**	**0.99**	**60.98**	**8.51**	**1.69**

This estimate is based on underground mine design work undertaken by Entech Pty Ltd. The estimate includes modifications to account for un-mineable material, dilution, and inferred metal within the mining shapes (any contained inferred material was set to waste grade). The following are the key assumptions for the reserve estimate:

- The long-term metal prices used for calculating the financial analysis are USD $1,900/oz gold, USD $4.50/lb Copper, USD $25/oz Silver.
- Gold Equivalents are calculated as AuEq = Au g/t + Cu % *1.62404 + Ag g/t*0.01316, based on commodity pricing. Metal payabilities and recoveries are not incorporated into this formula.
- A minimum mining width of 3.0 m has been applied for stoping, inclusive of a 1.0 m dilution skin at contained Mineral Resource grade.
- In addition to the 1.0 m dilution skin, dilution of 5% has been added for Avoca mined stopes and 2.5% for long hole stoping with paste fill. Where a stope is within 5.0 m proximity of the HW or FW of the fault gouge, an additional 1.0m of dilution was added at a grade averaging 1.42 g/t AuEq. This results in a total average dilution of 27.8%.
- Mining recoveries of 90% have been applied to Avoca mined stopes, and 95% for long hole stoping with paste fill.
- A cut-off grade of 3.5 g/t AuEq was used to define stoping blocks. Stope shapes with uneconomic development were excluded. The cut-off grade takes into account site operating costs, G&A costs, sustaining capital costs and relevant processing and revenue inputs.
- Measured Mineral Resources were used to report Proven Mineral Reserves.
- Indicated Mineral Resources were used to report Probable Mineral Reserves. No Measured Mineral Resources were used to report Probable Mineral Reserves.
- Tonnage and grade estimates include dilution and recovery allowance.
- The Mineral Reserves reported are not added to Mineral Resources.

The mine plan assumes mining of Mineral Reserve material only and was shown to be economically viable with a reasonable degree of margin to buffer against unfavourable input movements. It should, however, be noted that sufficient negative movements in the assumed gold prices and/or exchange rates, metallurgical recoveries and/or costs could potentially render portions or the whole of the Mineral Reserves mine plan uneconomic.

1.7 Mining Methods

1.7.1 Introduction

The underground mine has been operated by K92ML since October 2016 and has a current processing throughput of 600,000 tpa as per the Stage 2A Expansion. The 2024 DFS targets an expansion to a peak processing rate of 1,200,000 tpa with a 7-year mine life. The DFS assumes a start date of the life of mine schedule of 1 January 2024.

1.7.2 Geotechnical

Overall stability of the Kora stoping panels is largely controlled by the proximity of the K1 hangingwall and K2 footwall to the Fault Gouge Zone (FGZ). Rock mass conditions in terms of Q rating ranged significantly from 'Extremely Poor' up to 'Extremely Good'. These worst and best cases are considered rarer occurrences being spatially limited and linked to lithology contacts and/or fault / shear zones, with the ground deteriorating with increased proximity to the FGZ. On average both the Kora and Judd orebodies can be classed as good ground.

The FGZ is often situated between the K1 and K2 veins. Where it is proximal or intersecting, there is the possibility of slabbing, sliding and unravelling failure types accruing along structure planes and the FGZ. To control this, restricting stope spans, avoidance of undercutting of the FGZ or within a critical standoff distance of the FGZ, along with drill and blast practices is required to minimize stoping performance issues. This is the current practice and is expected to be improved upon with pastefill.

1.7.3 Mining Method Selection

The mining method selection process resulted in Avoca and modified Avoca being selected for mining prior to commissioning of the paste fill plant in Q3 2025, and longhole stoping with pastefill for the remainder of the mine life. An economic comparison was done in 2022 on extending the Avoca and modified Avoca mining methods for the full life of mine schedule instead of transitioning to longhole stoping with paste fill and confirmed that longhole stoping with pastefill provided superior project economics.

1.7.4 Mine Design and Physicals

On completion of the trade-off studies in 2022, Entech were provided with the 2022 DFS geological model. Stope optimisations were completed to compare to the trade-off studies and validate the selected Avoca, and LHS with paste fill mining methods. The stope optimisation results aligned with the findings of the trade-off studies and K92ML confirmed selection of the Avoca/Modified Avoca and LHS with paste fill mining methods for design and scheduling of the DFS.

A new DFS geological model completed in September 2023 was provided for this (2024) update of the DFS mine plan - it includes Measured and Indicated Mineral Resources.

A provisional cut-off grade was estimated based on costs derived from both the 2022 DFS and PEA studies along with updated inputs provided by K92. This information was then used in the trade-off study to generate the lower bound of 2.5 g/t AuEq for the DFS stope optimisations. To assess the appropriate final cut-off grade to use in design and scheduling, it was agreed to produce stope optimisations on a range of cut-off grades to conduct further analysis and compare results.

K92ML selected the 3.5 g/t AuEq stope optimisations to carry forward for the final mine design and DFS LOM scheduling. This selection was supported by the NPV analysis, cut-off grade estimates, and the trade-off study results as well as aligning with K92's objectives for inventory size and grade.

The mine is planned to produce at a rate of 1,200,000 t per annum (1.2M tpa). The mine life is 7 years with a 3-year production ramp up (inclusive of 2024), sustained production of 1.2M tpa for 3 years, and final year production of 0.3M tpa.

1.7.5 Ventilation

The PEA and DFS plan to ventilate K92ML underground sees fresh air entering the mine through four portals in the side of the mountain hosting the Kora deposit, with air exhausting out a large single exhaust air drive that is currently being developed. The original Puma portal will be repurposed as an intake, which is currently the main exhaust for the existing workings.

Production haulage is confined to two horizontal side-mountain drifts, with development haulage using the Kora and internal inclines. Fresh air enters all portals and converges at the mid-point, where production is currently occurring along the Kora Incline. From here a spiral incline continues to the top of the deposit. Exhausting air links all production zones to the newly developed 6.0mW x 6.0mH Puma return air drive that daylights near the old workings in the remnant area.

Using the peak flow rate from the PEA LOM model of 673 m^3/s to ensure possible future requirements can be met, Entech recommends that multiple primary fans in parallel are used to meet these duties. This means that the wide range in the performance envelope can be met when fan speed controls alone cannot satisfy, i.e. switching fans on and off between ventilation milestones. The peak flow rate for the DFS model is 499 m^3/s.

1.7.6 Mining Fleet and Personnel

Mining Fleet

Assumptions for mine equipment requirements have been derived by K92ML based on actual site productivities and first principles' calculations. The listed equipment types have been reviewed by Entech and are appropriate for the mine design and schedule developed by Entech.

This represents the equipment necessary to perform the following duties:

- Excavate the lateral and decline development in both ore and waste.
- Install all ground support including rock bolting and surface support.
- Maintain the underground road surfaces.
- Drill, charge and bog (including remote loading) all stoping ore material.
- Drill slot rises for production stoping.
- Install all underground services for development and production.

Personnel

The Personnel requirements have been calculated based on the equipment numbers for underground production. The proposed Personnel requirements are based on a mix of Papua New Guinea national employees and expatriate employees.

The typical roster for the national employees is 20 days on site and 10 days off, with a roster of 4 weeks on and 4 weeks off for expatriate employees.

The total number of employed personnel in the underground operation at peak production reaches approximately 561, with all positions detailed in Table 16-20, Table 16-21, Table 16-22, and Table 16-23.

1.8 Recovery Methods

The plant has been designed on the basis of treating underground ore from the Kora and Judd deposits. The process plant design and technology is well proven in the industry and treats ore through a conventional crushing and grinding circuit incorporating both gravity recovery and flash flotation, followed by conventional flotation to produce saleable gold-silver-copper concentrate and a dorè product. The flowsheet makes use of metallurgical unit processes already used in the existing Kainantu processing plant treating these ore types.

The key project design criteria for the plant are:

- Throughput of 1.2M tpa of ore with a grind size of 80% passing (P_{80}) 106 µm.
- Crushing plant utilization of 68.5% (6,000 hr/y).

- Grinding and flotation plant utilization of 91.3% (8,000 hr/y) supported by crushed ore storage, standby equipment in critical areas and back-up power from an on-site diesel power station.

The treatment facility incorporates the following unit process operations:

- Single stage crushing to produce a crushed product size of 80% passing (P_{80}) of 90 mm.
- Direct mill feed via a crushed ore surge bin with an emergency dead stockpile arrangement and reclaim feeder facilities.
- Primary semi-autogenous (SAG) milling followed by secondary ball milling in closed circuit with hydro-cyclones to produce a grinding circuit product size of 80% passing (P_{80}) of 106 µm.
- A dedicated flash flotation cell on the cyclone underflow to recover gold, silver and copper bearing minerals into a high-grade stream that will report to final concentrate.
- Gravity gold recovery from a portion of the cyclone underflow followed by gravity separation using wet shaking tables and gold room operations to produce dorè bars.
- Gold-silver-copper flotation utilising roughing, scavenging, cleaning, cleaner scavenging and recleaning stages to produce a saleable high-grade gold-silver-copper concentrate.
- Concentrate thickening, filtration and loading of concentrates into 20 ft sea-containers for dispatch from the mine site.
- Tailings thickening and disposal to a tailings storage facility (TSF) for deposition or to the tailings pressure filter plant for the paste fill system.
- Reagent mixing, storage and distribution.
- Water and air services.

1.9 Project Infrastructure

1.9.1 Power and Water

The supply of electric power to the project will be via grid connection to the Papua New Guinea Power Limited (PPL) network, 22 kV overhead line (OHL) from the Gusap Substation located 24 km away.

A diesel back-up power station forms part of the site-wide power reticulation for the project.

Generators that are located on the existing power system at the 800-portal area, 500k tpa Process Plant, and camp will operate independently as satellite systems until the main back-up power plant becomes fully operational.

K92ML recently constructed a 4.8 MW power station which became fully operational beginning of 2024. The main purpose of 4.8 MW power station is to be back-up power supplier for the UG facility in area 800 via existing 11 kV OHL, prior to the completion of the back-up power station.

Currently the area 800 Portal Switchroom, 12 MVA and 1.5 MVA associated transformers are scheduled to be on site by end of 2024, which will require conversion of the existing 11 kV OHL to 22 kV. At that point 4.8 MW power Station transformers will be adjusted to step-up the voltage to 22 kV.

Following are the summary of the existing and future power stations for the Kainantu Mine:

- Existing satellite gensets: These units are spread across the site in 3 locations: existing process plant, camp, and 800 portal. These units act as back up to the current power supply to the site. (22 kV OHL from Gusap substation).
- 4.8 MW power station: This was built and commissioned recently, using some of the existing gensets from the site in addition with new units. The main purpose of this power station is to act as back up for 800 portal and UG loads; as it was expected that UG load will increase prior to installation of the main back up power 20 MW power station. Soon this 4.8 MW power station will be re-purpose to be connected to 22 kV network.

- New 8.8/20 MW power station: this power station will act as back up for the entire Kainantu Mine when it becomes operational. All the other power stations and satellite units will become redundant. The new power station will be built in two (2) stages; each having 7 x 1.63 MW generators. This will result in total installed power of 14 x 1.63 MW unit after building the second stage.

Raw water for the process plant will be sourced from mine dewatering water from the 800 Portal. The water from the 800 Portal will make its way to the plant site via an existing HDPE pipe and report to sediment pond one and then overflow to sediment pond two. Raw water is then pumped from sediment pond two into the raw water tank and used as required.

Surface run-off from the plant site will be collected in a storm water pond, this is then recycled to be used again in the process plant.

1.9.2 Paste Fill Plant Project Infrastructure

To improve underground geotechnical stability, facilitate increased ore extraction, and to dispose of approximately 74% of the Kainantu tailings underground, cemented paste backfill manufactured from thickened and pressure filtered concentrator tailings is used as the mine backfilling method. The system is designed to use approximately 74% of the instantaneous available tailings, with a nominal paste production rate of 108 m^3/h and a design rate of 148 m^3/h to facilitate sprint production when required. The nominal production rate is expected to satisfy the peak underground backfill demand of 46,298 m^3/month, at a utilization rate of 59%. The utilization rate is typical for paste backfill systems and includes downtime for reticulation system changes, filter cake stockpile buildup, and regular maintenance.

The average stope size (~2,400 m^3) is expected to be filled in a single pour campaign taking into account stope bulkheads and exclusion zones, which will not require downtime associated with fill cure periods and which will accelerate stope turn-around times. Larger sized stopes may require a series of pour campaigns to complete but these are expected to be relatively few compared to the amount of average sized stopes.

As part of the stope extraction sequence, paste is to be exposed both vertically and horizontally. Due to the narrow stope widths, relatively low paste strengths (typically less than 550 kPa) are typically required. A paste cure period of 28 days is used to achieve this relatively low strength requirement and indicates a binder content of 5% should be used. Binder content may be required to vary up to 9% for certain mining areas and pumping distances. Optimization studies show the optimal binder product to be predominantly ground granulated blast furnace slag (slag) with GP cement addition, mixed at a mass ratio of 90:10 slag:cement. Being a waste product, the predominant use of slag (in the binding agent) presents a favourable solution from an environmental as well as cost perspective.

The operating philosophy of the Paste Fill Infrastructure was modified from the original feasibility design which pumped thickened slurry from the mill area to a processing facility located at the mine portal. The basic engineering design shifted to trucking pressure filtered tailings from the mill area leveraging the back haul cycle of the ore haul fleet to a staging area near the village located near the mine. The staging area colloquially known as the "bus stop" area will act as a tailings stockpile to decouple the operation of the mill from the capability to produce paste backfill. Decoupling the backfill system from mill and filtration operations will allow operations to improve plant availability/operability. The bus stop will serve as a staging area for both filter cake and binder storage to service the backfill plant operation.

From the bus stop a dedicated fleet of tailings haul trucks will transport the filter cake to the underground paste plant. In the underground plant the tailings will be mixed with water and binder slurry to form the backfill. The mixed backfill will be pumped to the stope using twin piston diaphragm pumps with one operating and one on standby.

1.9.3 Tailings Storage Facility

The Tailings Storage Facility (TSF) is a critical component of the project infrastructure, designed to contain the waste material from the Kainantu Mine Process Plant. As part of the Kora Expansion Project Definitive Feasibility Study (DFS), processing throughput is projected to increase from 600,000 tpa to 1,200,000 tpa, necessitating an enhanced tailings management system. The increase in throughput between 2024 and 2030 will also significantly increase the total volume of tailings produced.

To accommodate the proposed Kora Expansion Project, the existing TSF will be expanded, and an underground Paste Backfill System is proposed to be implemented which reduced surface tailings storage requirements.

Initially, 100% of the tailings will be directed to the TSF. By end-2025, with the commencement of the Paste Backfill Plant, over 70% of the tailings will be redirected to fill underground voids, with the remaining continuing to the existing TSF.

The existing TSF is a cross-valley tailings impoundment with a downstream embankment, standing 33 meters high. Initially constructed in 2005 using compacted earthfill materials, tailings were sub-aerially deposited up to approximately 502 mRL between 2006 and 2008. From 2008 to 2015, the facility was placed under care and maintenance, during which no tailings were added to the TSF. Mining operations resumed in 2016, and tailings have been continuously deposited in the TSF since then. In May 2021, the TSF embankment was raised by downstream methods to 515 mRL (Stage 1A/1B) using rockfill materials. A shear key trench, up to 5 meters deep, was excavated under the downstream rockfill buttress to enhance embankment stability and improve seepage collection. The embankment has an overall downstream slope ranging from 2.5H:1V to 3H:1V. The construction of Stage 1C, with the embankment raised to 517 mRL, was completed in October 2024.

Given the site is located in a tropical rainforest climate with high annual rainfall (1,750 mm to 2,000 mm) and very high seismic activity, robust measures are necessary to ensure stability and safety of the TSF. A probabilistic seismic hazard analysis highlights the importance of these measures. Additionally, a dam break analysis has been conducted to evaluate the potential impacts on nearby populations, particularly focusing on the Kumian mine camp, which is identified as a high-risk area. Recommendations include the construction of a flood protection levee around the camp to mitigate the impact of a dam break.

Previous geotechnical investigations had identified uncertainties in the foundation conditions, such as the presence of potentially liquefiable soils. This has led to a recent drilling campaign to obtain additional information to understand and address these foundation conditions before proceeding with the TSF expansion to an embankment crest level of RL 530 m (Stage 3). It should be noted that this investigation does not affect the detailed design of Stage 2. The Stage 2 design has been finalized and is in the final stages of MRA approval.

Geochemical testing of the tailings indicates that the tailings material is non-plastic and contains pyrite (making it potentially acid-forming), while the waste rock used in embankment construction is low in sulfur and non-acid forming. This differentiation in material properties is crucial for planning the handling and disposal of these materials.

While the construction of a new TSF was considered, it was determined that expanding the existing TSF, coupled with the implementation of the Paste Backfill system, remains the preferred option. This decision is supported by the known site conditions, the relative simplicity of operation, and a more favourable balance of capital and operating costs.

This approach not only aligns with environmental management practices but also leverages existing infrastructure to meet increased production demands efficiently and sustainably.

1.10 Environmental Studies, Permitting and Social or Community Impact

1.10.1 Legal Requirements

Papua New Guinea National Government regulatory bodies such as the Mineral Resources Authority (MRA) and the Conservation and Environment Protection Authority (CEPA) have granted various approvals, permits and licenses required by K92ML to operate the Kainantu Gold Mine. The primary legislation for these approvals is the Mining Act 1992 and the Environment Act 2000.

K92ML holds Environment Permit EP-L3(34) issued under the Environment Act 2000. In December 2023, the Company received a revised environmental permit from CEPA for the Stage 3 Expansion. The receipt of the permit followed a standard, government-sanctioned environmental impact assessment process. The revised permit contained minor amendments to the existing environmental authorizations. All environmental authorizations are now consolidated under the revised EP-L3(34).

As per environmental regulations, a revised Environmental Management Plan is currently being developed for submission to CEPA to enable meeting the requirements of the revised permit. Following receipt of the permit, CEPA advised K92ML that permitting activities for a new tailings storage facility, which as per the mine plan will be required in 2027, should commence in mid-2024.

K92ML's existing mining lease (ML150), contains approximately two-thirds of the known Kora resource, 100% of the known Judd resource, and was issued under the Mining Act 1992. The Kora deposit extends from ML150 to the south, into K92ML's exploration license, (EL 470). K92ML will need to apply for another mining license to include the extension of Kora to the south prior to mining. The Company intends to submit a mining lease application for these purposes in the second half of 2024.

1.10.2 Environment and Community Setting

The project area is part of the upper Ramu River catchment. The surroundings have a dendritic drainage system composed of smaller creeks and streams draining from the surrounding mountains and hills in the area into four catchments. Two of these catchments are directly impacted by the project infrastructure. The local topography and geology mean the hydrogeological conditions are complex. Local groundwater systems, localized recharge areas and inter-connectivity with regional aquifer systems determine recharge of groundwater in the project area.

Water quality monitoring locations for the Kainantu Gold Mine are situated in three catchment areas - Baupa catchment, Maniape Catchment and at Ramu River where the access road crosses. Water quality monitoring has been undertaken or commissioned by K92ML and previous operators. Results have been reported in annual reports from 2004 to present and this monitoring is ongoing.

The vegetation surrounding the mine area is mainly of irregular hill forest. The forests in this area have irregular canopy and secondary species are common. The area has low intensity shifting agriculture surrounded by tall secondary forest. Several terrestrial flora and fauna species in the greater region (i.e. within the bordering Ramu floodplains) are IUCN red listed threatened species, some of which are endemic to the surrounding area.

The Government of PNG has designated four "impacted communities" in the project area, namely the villages of Bilimoia (comprised of three sub-villages), Unantu, Pomasi, and Sakimaniap (Watarais/Marawasa). There are other informal settlements (i.e. hamlets) within the vicinity of the mine that have been established as mining operations have progressed.

The K92ML External Affairs and Sustainable Development department conducts ongoing social baseline and mapping surveys to collect demographic and socioeconomic indicators related to local communities. Based on these surveys, the total population of the "impacted communities" is 8,718.

To support its social license to operate, K92ML has a range of community and social development initiatives in place relating to the supply of services, infrastructure and maintenance for local communities, water supply, health programs, education assistance, livelihood programs and business development. This is in addition to prioritizing members of impacted communities for direct employment at the mine.

The Company continues to work towards signing a revised Memorandum of Agreement (MOA). The MOA provides a framework for the relationship between the Company, the Community, and Government and sets out commitments from the various parties. In July 2020, the Company had a formal MOA meeting involving local landowners, the State, and the Provincial Government. Attending the meeting were representatives from local clans, the PNG Mining Minister, the Managing Director of the Mineral Resources Authority of PNG, and the Provincial Governor. In principle, the parties agreed on a revised MOA, which requires approval from the National Executive Council of the PNG Government. Follow up consultations were held in mid-2023. The original MOA framework will remain in place as mining operations and associated expansions continue, until a new MOA is formally approved, which is a common occurrence for natural resource projects in the country. Delays are primarily attributed to the Land Titles Commission (LTC) completing an investigation into landholding within the existing operations' leased boundaries.

1.10.3 Assessed Environment and Community impacts

Based on the proposed developments as part of the Kora Expansion Project the key potential environmental impacts are likely to be:

- Acid and metalliferous drainage from mine waste.
- Adverse changes to surface water quality and hydrology.
- Adverse changes to groundwater quality and flow.
- Reduction in downstream ecological values.

The scale, duration and intensity of these impacts and management and mitigation measures to address them were the focus of investigations during the EIS process.

The proposed expansion will increase the mine's workforce and employment for the local communities. There will be continued community development enhancements through provision of livelihoods training, educational assistance for children, training opportunities, provision of easier access to clean and safe drinking water, assistance in identifying genuine landowners, and creation of business opportunities. In using underground mining methods, the surface disruption to landowners is minimized; however, an extended or new mining license to the south of ML150, to cover the extension of the Kora deposit into EL470 will require the necessary permitting, according to PNG Mining Law, prior to mining. Part of the process involves a compensation agreement with the landowner group concerned.

While Landowner identification and social mapping would be required to extend the mining lease, it is anticipated that there would be no change to the recognized Landowners. It is noted that the outstanding LTC determination of landholdings within the existing project will also apply to the extension of the ML. At present, some compensation payments, including distribution of a portion of royalties, are being accrued pending the determination from the TLC.

Construction activities will produce dust and noise which may impact on nearby receptors. In the context of the operating mine, these emissions are expected to be negligible compared to dust and noise emissions from existing mining and processing activities.

Scope 1 and Scope 2 greenhouse gas (GHG) emissions associated with the Kora Expansion Project are expected to increase on an absolute basis. However, GHG emissions are expected to decline significantly based on an emissions per tonne milled and an emissions per gold-equivalent ounces produced basis.

In June 2023, K92ML set an emissions reduction target of a 25% reduction in GHG emissions by 2030 against a business-as-usual forecast, which is defined as a reduction in GHG emissions against an established baseline forecast assuming no intervention measures have been taken to reduce carbon emissions. A key component of the Company's strategy to achieve this target is enhancing the mine's access to grid electricity, and in doing so, reducing its reliance on diesel-powered electricity generation. Hydroelectricity comprises a significant amount of the local grid electricity, which is a cleaner alternative to diesel-powered generators.

1.10.4 Management Plan

The existing operations run under the CEPA-approved Environmental Management and Monitoring Plan (EMMP). The purpose of the EMMP is to satisfy conditions under the existing Environment Permit EP-L3(34) and to provide a policy framework, K92ML management commitments, and monitoring and improvement actions necessary to prevent, mitigate or manage environmentally degrading conditions resulting from the mining operations. Routine environmental monitoring as part of the EMP will continue to provide relevant environmental data to assess and manage operational impacts. As per the requirements of the revised environmental permit issued to K92ML in December 2023, the EMMP will need to be updated to account for minor permit amendments.

The Company implements ongoing community consultation and engagement initiatives tailored to the local operating environment in PNG, given the unique sociocultural context within the country. The Company maintains a strong presence in local communities, including through Village Liaison Officers (VLOs), who are present in local communities on a daily basis to enable frequent, two-way communication with local stakeholders. The Company also maintains a regional external affairs office in Kainantu, the district capital of the Kainantu District, Eastern Highlands Province.

The Company's community engagement efforts are led by its External Affairs and Sustainable Development team, which is composed of approximately 40 community relations practitioners. The Company maintains a grievance mechanism to obtain feedback and resolve complaints from local communities.

1.10.5 Closure Plan

The Kainantu Gold Mine has a conceptual mine closure plan, and it will need to be updated to include closure and rehabilitation aspects that will arise from the proposed expansion project. The key aspects that require inclusion in the updated mine closure plan that relate to the expansion project are:

- Additional infrastructure, including the new processing plant, paste plant, and ancillary infrastructure.
- Maintaining TSF integrity. (Initial work pertaining to an update of the TSF closure plan has commenced in Q4 2024.)

1.10.6 Forward Works Plan

The forward works plan for environment and community aspects includes updating the operational EMMP in support of the revised environmental permit that was issued in December 2023. Initial permitting work for a new tailings storage facility that will be required to commence construction in 2027 as part of the Stage 4 Expansion will commence in 2025.

1.11 Capital and Operating Costs

1.11.1 Capital Cost

The capital estimate for the DFS is summarised in Table 1-3 and Table 1-4.

Table 1-3 Capital Estimate Summary (USD, 1Q24, +15%/-5%)

Main Area	USD M
Mine	$0.0
Process Plant	$90.3
Paste Plant	$42.5
Power	$19.1
Camp	$4.2
River Crossing	$14.4
Warehouse/Maintenance Facilities	$3.8
Owners Costs	$20.1
Subtotal	**$194.4**

The total LOM capital cost for the DFS is estimated at USD $540.5M, including sustaining capital costs of USD $346.1M, as shown in Table 1-4.

Table 1-4 Sustaining Capital Estimate Summary (USD, 1Q24, +15%/-5%)

Main Area	USD M
Mining	$265.3
Processing Plant	$5.4
Surface Fleet	$39.2
Site Services	$19.3
TSF	$7.9
Closure	$9.0
Grand Total	**$346.1**

1.11.2 Operating Cost

The mining operating cost estimate for the DFS has been compiled by Entech. For the new process plant, the operating cost estimate has been compiled from a variety of sources, including K92ML advice, metallurgical test work, first principle calculations, vendor quotations and the GRES database. In terms of the infrastructure, the paste backfill system operating cost estimate has been compiled by GRES. The TSF operating cost estimate has been compiled by ATC Williams. The general and administration (G&A) costs have been compiled by K92ML.

The operating cost estimates for the DFS are summarised in Table 1-5.

Table 1-5 Operating Costs Summary (USD, 1Q24, +15%/-10%)

Area	Source	USD $ / Value
Mining (average over LOM)	Entech	$68.1 / t ore
Processing Plant - Existing - Fixed	GRES	$5.4M / year
Processing Plant - Existing - Variable	GRES	$12.3 / t ore
Processing Plant - New - Fixed	GRES	$12.5M / year
Processing Plant - New - Variable	GRES	$7.7 / t ore
Processing Plant - New - Total (average over LOM)	GRES	$19.4 / t ore
General & Admin - Total (average over LOM)	K92ML	$37.1 / t ore
Paste Plant - Fixed	GRES	$2.1M / year
Paste Plant - Variable	GRES	$23.8 / m^3 paste
Paste Plant - Total (average over LOM)	GRES	$29.6 / m^3 paste
TSF (average over LOM)	ATC Williams	$0.64 / t ore
Transport & Insurance (average over LOM)	K92	$9.9 / t ore
Total (average over LOM)		**USD $145.4 / t ore**

1.12 Economic Analysis

An economic analysis has been carried out for the project using a cash flow model. The model is constructed using annual cash flows by considering annual mined and processed tonnages and grades, process recoveries, metal prices, operating costs and refining charges, royalties, and capital expenditures (both initial and sustaining).

PNG tax regulations are applied to assess the tax liabilities. All amounts in this section are presented in USD.

The cash flow model commences with operation in 2024, with processing through the existing plant. Discounting has been applied monthly from the first year of operation.

The model reflects the base case and technical assumptions as described in the foregoing sections of this report.

1.12.1 Model Inputs and Assumptions

The key model inputs used in the economic analysis for the DFS are summarised in Table 1-6.

Table 1-6 Key Model Inputs

Model Inputs	Source	Unit / Value
Base Currency		USD
Base Date		1st Quarter 2024
Ore Processed over LOM	Entech	6.18 Mt
Metal Prices		
Gold	K92ML	USD1,900 / oz (fixed)
Copper	K92ML	USD4.50 / lb (fixed)
Silver	K92ML	USD25 / oz (fixed)
Recoveries		
Gold (total)	MMS	92.6%
Gold (gravity to doré)	MMS	15.0%
Copper	MMS	94.0%
Silver	MMS	78.0%
Concentrate copper grade	MMS	18.7%
Processing Plant Capacity (dry tonnes of ore)		
Existing Plant	K92ML	600,000 tpa
New Plant	GRES	1,200,000 tpa
Royalties (deducted from net revenue)	K92ML	2.0%
Levies (deducted from gross revenue)	K92ML	0.5%
Tax Rate	K92ML	30%
Depreciation		Not considered
NPV Discount Rate	K92ML	5%

1.12.2 Financial Model Results

The financial model indicates that the DFS project has a post-tax Net Present Value (NPV) of USD $680.5M at a discount rate of 5%.

Figure 1-6 shows the post-tax annual and cumulative cash flow for the project over the LOM.

Cash flow starts negative in 2024 and 2025 due to the expenditure on the Stage 3 expansion project.

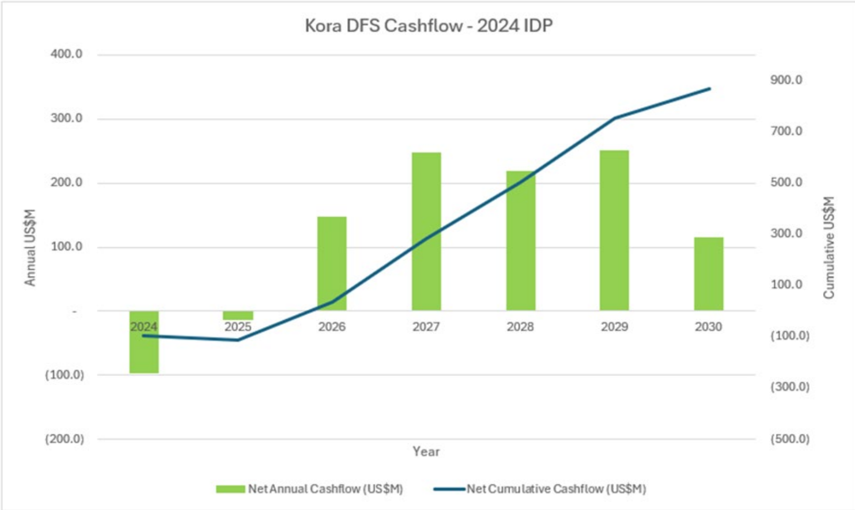


Figure 1-6 Cumulative Cash Flow

A summary of the cashflow analyses for the DFS is shown in Table 1-7.

Table 1-7 Simplified Financial Model at USD $1,900/oz Au, USD $4.50/lb Cu, USD $25.00/oz Ag

Year	2024	2025	2026	2027	2028	2029	2030
Mill Throughput (k tpa)	473	800	1,001	1,200	1,200	1,200	302
Gold Grade (g/t)	7.59	6.54	6.43	6.86	5.26	6.48	11.66
Copper Grade (%)	0.60%	0.55%	0.62%	1.11%	1.48%	1.10%	1.07%
Silver Grade (g/t)	11.48	12.04	14.37	22.44	25.07	18.50	24.39
AuEq Grade (g/t)	**8.72**	**7.59**	**7.63**	**8.95**	**8.00**	**8.52**	**13.72**
Gold Produced (000s oz)	107	154	190	245	190	231	106
Silver Produced (000s oz)	136	241	361	675	754	557	185
Copper Produced (M lbs)	5.9	9.1	12.7	27.5	37.1	27.4	6.8
AuEq Produced (000s oz)	**122**	**179**	**225**	**319**	**287**	**303**	**125**
Net Revenue (USD M)(1)	$212	$310	$405	$575	$507	$547	$225
Total OPEX (USD M)(1)	$96	$130	$159	$179	$172	$172	$57
Growth Capital (USD M)	$107	$87	-	-	-	-	-
Sustaining Capital (USD M)	$82	$72	$47	$56	$38	$30	$13
Closure Costs (USD M)	-	-	-	-	-	-	$9
After-tax CF (USD M)	($98)	($14)	$147	$247	$219	$252	$116
Cumulative After-tax CF (USD M)	($98)	($112)	$36	$283	$502	$754	$869

1. *Net revenue in summary model excludes the impact of royalty payments and transport costs. These are included within operating costs.*

1.13 Kora 2024 Preliminary Economic Assessment Case

1.13.1 PEA Overview

The alternative PEA Case conceptualizes a multi-expansion plan to an ultimate plant run-rate of 1.8M tpa, representing a 200% increase from the upgraded Stage 2A Expansion run-rate of 600,000 tpa design throughput (previously 500,000 tpa). The PEA Case involves the construction of a standalone 1.2M tpa process plant adjacent to the 600,000 tpa Stage 2A Expansion process plant. In mid-2025, the Stage 2A Expansion process plant is idled as the 1.2M tpa Stage 3 process plant ramps up, with commissioning of the Stage 3 Expansion process plant commencing in late Q2 2025. Upon achieving the Stage 3 run-rate throughput in 2027, the Stage 2A Expansion plant is recommissioned in H2 2027, ramping up to run-rate throughput of 600,000 tpa by year end, for a combined processing run-rate of 1.8M tpa at the beginning of 2028.

To support the higher throughput rate, the underground mining fleet is significantly increased to support expanded mining operations, opening up multiple mining fronts concurrently: Kora Upper, Lower and Central Zones within the Kora deposit, and the Judd deposit. Site infrastructure is also expanded, including power, camp facilities and the pressure filter plant. Several capital items, such as the paste fill system, are configured during the construction of Stage 3 to be amenable to the larger ultimate Stage 4 Expansion run-rate.

The PEA uses the conclusions of the Company's Mineral Resource estimate for Kora and Judd (effective date of 12 September, 2023) and does not incorporate post resource drilling results. The effective date of the PEA life of mine plan is 1 January, 2024; therefore, Kora is net of post-resource mining depletion from 12 September, 2023 to 31 December, 2023 of 183,768 tonnes at 8.05 g/t Au, 0.92 % Cu and 15.37 g/t Ag.

The PEA is preliminary in nature and 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, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-8 PEA Highlights

US Dollars unless otherwise stated	**Life of Mine (starting January 2024)**	**Stage 4 Run-Rate[(1)] (2028 - 2034)**
Production		
Mine life (years)	14 years	
Total mill feed (000s tonnes)	20,396	12,600
Average mill throughput (tonnes per annum)	1,457k tpa	1,800k tpa
Total Metal Production		
AuEq (000s ounces)	4,977	2,895
Gold (000s ounces)	3,620	2,011
Copper (M lbs)	508	332
Silver (000s ounces)	11,799	7,399
Peak Annual Production		
Year	2034	
AuEq (000s ounces per annum)	485	
Average Annual Metal Production		
AuEq (000s ounces per annum)	356	414
Gold (000s ounces per annum)	259	287
Copper (M lbs per annum)	36	47
Silver (000s ounces per annum)	843	1,057
Average Grade		
AuEq grade (g/t)	8.2 g/t	
Gold grade (g/t)	6.0 g/t	
Copper grade (%)	1.2%	
Silver grade (g/t)	23 g/t	
Average Recovery		
Gold Recovery (%)	93%	

US Dollars unless otherwise stated	Life of Mine (starting January 2024)	Stage 4 Run-Rate[1] (2028 - 2034)
Copper Recovery (%)	94%	
Silver Recovery (%)	78%	
Costs		
Mining Cost ($/t ore mined)	$62.60	$58.58
Processing Cost ($/t processed)	$18.19	$17.65
General & Administrative Cost ($/t processed)	$25.45	$23.33
Paste Plant Cost ($/t processed)	$9.16	$9.89
TSF ($/t processed)	$0.48	$0.42
Transport and Insurance ($/t processed)	$10.24	$10.44
Total operating cost per tonne processed ($/t)	**$126.12**	**$120.59**
Royalties ($/t processed)	$10.75	$10.10
Sustaining capital per tonne of processed ($/t)	$44.15	$35.37
Total cost per tonne processed ($/t)	**$181.02**	**$166.06**
Growth capital expenditure ($M)	$201	
Sustaining capital expenditure ($M)	$900	
Total capital expenditure with closure costs ($M)	**$1,122**	
Cash cost per ounce AuEq ($/oz)[2]	$633	$644
All-in sustaining cost per ounce AuEq ($/oz)[3]	$822	$805
Cash cost per ounce gold ($/oz)[2]	$174	$108
All-in sustaining cost per ounce gold ($/oz)[3]	$432	$338
Base Case Economic Analysis at USD 1,900/oz Gold, USD 4.50/lb Copper and USD 25.00/oz Silver		
After-tax NPV0%	$3.5 billion	
After-tax NPV5%[4]	$2.3 billion	
Economic Analysis at USD $2,500/oz Gold, USD 4.50/lb Copper and USD 25.00/oz Silver		
After-tax NPV0%	$5.0 billion	
After-tax NPV5%[4]	$3.3 billion	

1. *Run-rate calculated based on 2028-2034.*
2. *Cash costs are net of by-product credits and are inclusive of mining costs, processing costs, site G&A, treatment and refining charges, and royalties.*
3. *AISC includes cash costs plus sustaining capital costs and accretion.*
4. *Net present value is calculated utilizing monthly discounting.*

1.13.2 Kora 2024 PEA Mining Methods

The mine plan considered in the PEA is designed as an incline access operation with a series of ore passes for efficient material movement between sublevels and the twin incline for material transport to surface. Initially, the mine plan employs a long hole open stoping mining method utilizing Avoca and modified Avoca, with waste rockfill. This mining method has been successfully employed at the mine since Q1 2020 at Kora and Q4 2021 at Judd. The mining method will then transition to long hole open stoping with paste fill in Q3 2025, upon the construction of the paste fill plant. Cemented paste fill provides improved geotechnical conditions yielding lower stope dilution, higher mining recovery factors, and greater operational flexibility through the ability to mine above and below paste fill, plus a reduction in surface tailings storage requirements.

1.13.3 Kora 2024 PEA Infrastructure

This section evaluates the tailings storage solutions required for the Kora Stage 4 Expansion under the PEA scenario. The scenario forecasts an increase in processing throughput from 600,000 to 1,800,000 tpa. This expansion necessitates enlarging the existing TSF and constructing a new TSF to accommodate the additional tailings.

Initial assessments have highlighted several challenges for the development of a new TSF, including:

- Obtaining regulatory approvals and understanding the complex geological and foundation conditions characteristic of PNG.
- The availability of large volumes of waste rock from the current operating mine, add further complexity to the construction of a new TSF.

The construction of a new TSF is estimated to require a minimum of three years, accounting for site investigation, engineering design, procurement, construction, and securing necessary regulatory approvals.

A high-level desktop study has identified the construction of a conventional tailings storage facility with an 18-meter-high downstream embankment using rockfill material as the most viable option. This facility is expected to accommodate the increased volume of tailings after the existing TSF reaches its full capacity by mid-2029. The decision to utilize a rockfill dam is supported by the availability of waste rockfill material, which will become available as the underground mine transitions to the stoping method using paste backfill. This method is favoured for its reliability, feasibility under the outlined constraints, and efficiency in material handling.

The construction of a conventional rockfill dam is deemed the preferred solution for the new TSF, based on the high-level assessments conducted so far which identified its proven reliability and its compatibility with the anticipated changes in mining operations and material handling philosophies.

The detailed evaluation and strategic planning outlined in this section aims to ensure that the tailings storage facilities can support the projected increase in ore processing rates under the PEA for the Kora Expansion scenario. This planning addresses the environmental, logistical, and technical challenges associated with the PNG region.

1.13.4 Kora 2024 PEA Paste Fill System

To satisfy the PEA scenario for peak backfill requirements, ~60k m^3/mo is required. Utilising this backfill system would have an 'instantaneous' production rate of 148 m^3/h (based on a 1.8 Mtpa mine production rate), which is expected to satisfy the mining requirements with a utilization of 60%. While this is a slightly higher utilization, compared with the 54% targeted for the DFS, with the proposed system, this was considered achievable.

PEA mining scenario and stope sequence are understood to remain the same as that described for the detailed DFS and as such the stope filling strategy and fill strength requirements are also expected to remain unchanged.

Relative to the flowsheet described for the DFS, the only change required to accommodate the increased production rate, is the addition of plates to the pressure filters to facilitate higher cake production rates. No changes were required to the storage / mixing tanks, overland pipelines, paste mixer, or the binder addition system.

The PEA pastefill plant has a similar system with no increase in personnel expected to be required, relative to the DFS case. This results in a slight reduction in operating unit cost ($/m^3 placed) for the PEA relative to the DFS.

1.13.5 Kora 2024 PEA Economic Analysis

Entech prepared a conceptual cashflow and discounted cashflow derived from the life-of-mine schedule. Tax calculations for the after-tax cashflow and discounted cashflow were prepared by K92ML.

The PEA financial model indicates that the project has a post-tax Net Present Value (NPV) of USD $2,295M at a discount rate of 5%.

The PEA is preliminary in nature and includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves, and there is no certainty that the PEA will be realised. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Figure 1-7 shows the post-tax annual and cumulative cash flow for the project over the LOM.

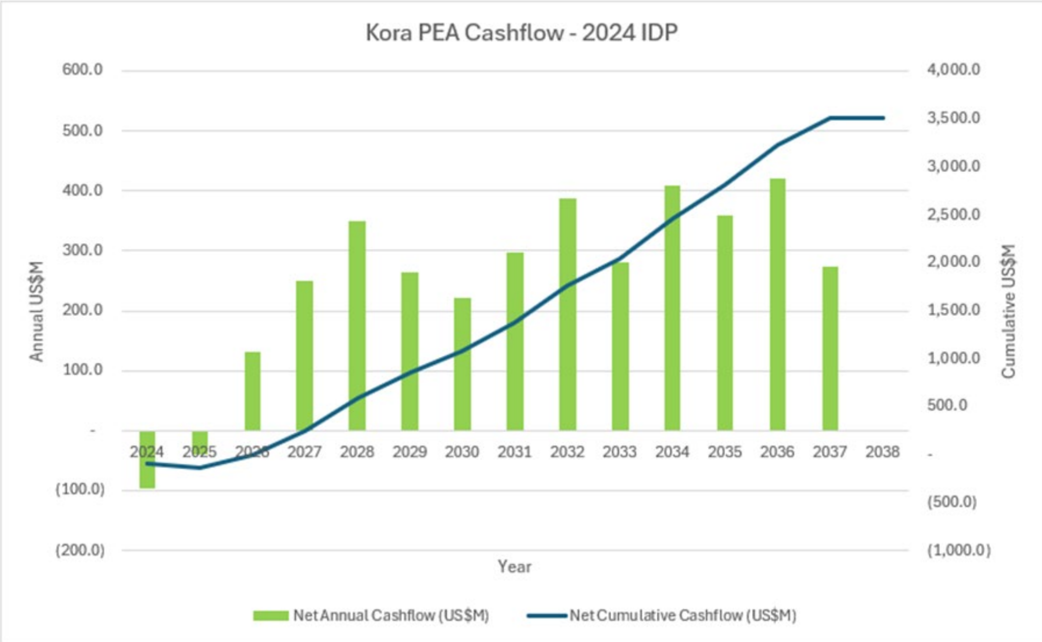


Figure 1-7 PEA Cashflow - 2024 IDP

1.14 Interpretation and Conclusions

1.14.1 Mining and Mineral Reserve Estimate

The DFS demonstrates robust economics, schedule and mine life. As an operating mine, there is increased confidence in execution of the LOM plan, and the accuracy of costs.

The proposed mining methods are well understood and utilised globally in underground mining. The proposed mine plan is technically achievable.

It is Entech's opinion that the Mineral Reserve estimation is reported in compliance with the NI 43-101 standards.

The mine plan assumes mining of Mineral Reserve material only and was shown to be economically viable with a reasonable degree of margin to buffer against unfavourable input movements.

1.14.2 New 1.2M tpa Treatment Plant

The 1.2M tpa processing plant design flowsheet incorporates a conventional single stage jaw crushing (200 tph) reporting to a crushed ore overflow surge bin and dead stockpile providing 12 hours of stockpile capacity. The primary crushed ore supplies a SAB milling circuit (150 tph) that includes an open circuit SAG mill and a closed circuit ball mill. The ball mill product reports to hydro-cyclones, with cyclone overflow reporting to the flotation circuit and the cyclone underflow stream being split between the gravity circuit, flash flotation cell and ball mill for grinding.

The gravity circuit comprises of a single batch centrifugal concentrator followed by two stages of gravity separation using wet shaking tables to upgrade the gravity concentrate, which is then calcined and smelted to produce gold doré. The flotation circuit includes flash flotation and conventional sulphide flotation, followed by thickening and filtering to produce a saleable gold-copper-silver concentrate. The circuit is based on conventional technology with the flowsheet largely similar to but optimized from the existing Kainantu processing plant. The key difference between the existing plant and the new treatment plant flowsheet configuration is the implementation of a single stage crushing, SAG and ball mill comminution circuit versus the existing two stage crushing and single stage ball circuit.

1.14.3 Tailings Storage Facility

As part of the production expansion, the management of tailings will utilize the existing TSF and the underground mine stopes. To ensure the TSF's readiness and compliance with both local and international standards, a specialized tailings consultant has been engaged. This review focuses on assessing the structural integrity, environmental compliance, and expansion capability of the TSF.

The tailings management strategies adhere to stringent guidelines, including:

- Australian National Committee on Large Dams (ANCOLD, 2019): This document provides comprehensive standards for the planning, design, construction, operation, and closure of tailings facilities. Adherence to these standards ensures robust structural integrity and operational safety.
- Global Industry Standards on Tailings Management (GISTM, 2020): These standards are designed to enhance the mining industry's practices by integrating social, environmental, and technical considerations. Implementation of GISTM guidelines has led to improvements in environmental monitoring systems and community engagement processes.

In addition to international standards, the tailings management practices are aligned with local regulatory requirements, ensuring that they meet specific local environmental and safety standards. The commitment to these standards demonstrates a proactive approach to responsible tailings management, which includes regular updates to practices as new regulations and technologies emerge.

1.14.4 Paste Plant

To improve underground geotechnical stability, facilitate complete ore extraction and dispose an estimated 74% of the Kainantu tailings underground, cemented paste backfill manufactured from fresh concentrator tailings is the preferred mine backfill method for both the DFS and PEA K92ML mining plans.

The topographic conditions and site layout at Kainantu present unique challenges for the paste system design. Through rigorous laboratory characterization and utilization of design principals, based on proven design methodologies, a robust paste system is proposed. This system can service the mining operation with the quantities and quality of fill required to enable the proposed mining strategy, while disposing of mine tailings in an environmentally friendly manner.

1.15 Recommendations

1.15.1 Mineral Resources and Exploration

- The entire mineralized district held under tenement should be assessed but with priority given to certain areas.
- Infill drilling should continue to upgrade both the Kora and Judd Mineral Resources. Drilling should be undertaken to the south of both Judd and Kora, following up significant isolated drill intercepts, with a view to expanding the current Mineral Resources.
- Reconnaissance drilling of other potential mineral lodes/systems identified from surface mapping and sampling is recommended.
- The Kora lode(s) is demonstrably richer in copper towards the south, thus potentially being closer to the original fluid source. This corresponds with an area of interpreted extensive dilation, where high grade gold/copper veins are haloed by long intervals of moderate grades. This corridor, linking Kora and Judd with the A1 porphyry target, known as Kora South and Judd South, should be a priority in the ongoing exploration program.
- Maniape and Arakompa both have historic resources which makes them a high priority for follow up exploration programs.

- The work program for each licence has been planned taking into consideration the current level of exploration on the tenement. Some programs will require detailed surface work prior to any drilling. Surface work should include assessment of lithocaps and vein expressions, as well as geophysical anomalies prior to drilling and are planned to be used in any future resource work.

1.15.2 Mining

Items for consideration are detailed in the following list:

- Optimization studies to increase NPV through scheduling and design improvements.
- Optimization of ore pass locations, and potential use of truck chutes at their termination points as opposed to rehandle with a loader.
- Continue to develop the mine with setting up required infrastructure i.e. air, water, power and ventilation.
- Prioritize decline and incline developments to link upper part (access from Portal 1) to lower part (access from Portal 2/3) of the mine together so multiple accesses to entire mine can be established.
- A dedicated geotechnical drilling program designed to cover critical infrastructure locations and the Kora orebody.
- Ongoing campaign geotechnical logging of resource definition drill holes, with focus on any holes intercepting mineralised zones Judd, K1 and K2 and the FGZ.
- Additional geotechnical testwork is required for all lithologies to confirm DFS assumptions for operational levels of robustness.
- A stress measurement program utilising the either WASM AE (Acoustic Emission) method or Hollow Inclusion Cell (HI-Cell) method is advised to be commissioned from the Kora mining area to assess the in-situ major principal stress magnitudes and directions at the Kora deposit.
- Detailed geotechnical mapping of ore drives should be routinely undertaken for input into stope design and determining a more accurate characterization of the rock mass.
- Numerical modelling of the stope extraction sequence needs to be refined once final stope designs are determined and re-run using the stress measurement test results to assess mining-induced stress re-distribution in terms of magnitudes and directions.
- A full review of the ground support systems should be undertaken as mining progresses to include updated geotechnical data to improve and justify the standards recommended.

1.15.3 1.2M tpa Treatment Plant

Complete the construction of 1.2M tpa Treatment Plant as per design and allocated timeframe.

1.15.4 Tailings Storage Facility

Based on the Definitive Feasibility Study for the Kora Expansion, the following recommendations are made to manage the tailings effectively and sustainably:

- Foundation assessment: Conduct comprehensive assessments to validate the suitability of the TSF foundations. This involves detailed geotechnical evaluations to address potential risks and ensure structural integrity under increased load.
- Maximization of Settled Density: Implement actions that maximize the settled density of tailings. Techniques such as adjusting the tailings' material properties or revising the deposition methodology should be considered to enhance the compactness and stability of the deposited tailings.
- Enhanced Drainage Systems: Consider the installation of additional drainage systems within the TSF to promote the consolidation of tailings. Improved drainage will facilitate faster water removal, which contributes to the overall stability of the tailings mass.
- Sustained Water Discharge: Implement systems to ensure sustained discharge of water from the tailings to minimize the volume of stored water within the TSF. Effective water management is critical to reducing the pressure on the embankment and enhancing the operational safety of the facility.

- Water Pond Management: Manage the size and location of the water pond within the TSF to ensure that tailings beaches are consistently maintained against the embankment. This practice is crucial in controlling the migration of the water body and preserving the structural integrity of the embankment.

These recommendations aim to enhance the operational efficiency and safety of the TSF, ensuring compliance with international standards and best practices in tailings management. Regular monitoring and adaptive management strategies should be employed to respond to any changes in tailings behaviour or external conditions.

1.15.5 Paste Plant

The filter plant has the potential to be a bottleneck for backfill operations. Pressure filters are known to require significant maintenance with lower availability compared to other filtration methods. K92ML would ensure that sufficient operational spares for the filters are always stocked on site to deal with all maintenance eventualities and the vendors should be tasked with providing a list of best practices for operations.

K92ML should implement a geotechnical program at the proposed “bus stop” area to determine final foundation requirements for the support of the tailings stockpile, wash bay and binder silo installations. In addition, enhanced fleet maintenance planning will be required to allow for the additional trucking requirements for hauling binder to site and servicing the extra wear that the haul fleet will experience due to the back haul of filter cake tailings from the mill to the bus stop. Also, the extra vehicles required to haul cake from the bus stop to the paste plant will introduce additional maintenance capacity requirements. This has been accounted for in the workshop design planning completed by K92ML.

The back haul of filter cake tailings using the ore haul fleet has the potential to introduce over-sized material into the backfill production stream. It will be necessary for K92ML to ensure that all truck operators are well trained on the use of the wash bay facilities. Proper procedure and training programs will help to prevent excessive fouling of the backfill stream and protect the piston diaphragm pumps from potential damage caused by oversized material entering the system.

Slag and cement supply streams will form a significant portion of the operational costs of running the backfill system. Volatility in the price of these commodities will therefore have a significant impact on the cost per tonne of backfill for K92ML. Kainantu staff should remain vigilant in looking for opportunities to find alternative sources for these inputs in case there is ever a disruption, or price increase from currently suppliers.

Locating the paste plant underground will involve different maintenance and operational challenges. K92ML must be prepared to service the maintenance needs of a remote plant. The paste plant will require a level of care and maintenance similar to the mill. Additionally, the operational challenges presented by pumping backfill up into the mine and the handling and stockpiling of filter cake will require proper training and management of operational procedures for upset and abnormal conditions. Operators should be trained on the intricacies of this installation configuration and be tested regularly for upset condition response preparedness.

1.15.6 Environmental Studies, Permitting and Social or Community Impact

- Geochemical characterisation of the ore body and waste rock should be reviewed, and an ore/waste acid mine drainage potential block model developed. With an extensive exploration program underway, drill core sampling is a cost-effective way to sample, and feed into a predictive ore/waste model. Potential acid generating wastes can be managed at the underground source rather than retrospectively on surface.
- Timely resolution of the 2020 MOA renegotiation with the Provincial Government and Landowners, with signing of the MOA still awaiting approval from the National Executive Council of the PNG Government pending the results of the PNG Land Titles Commission appeals process. This will result in flow of royalties (held in escrow) to the landowners and provide the basis for future negotiations as the mine develops.

- The Environmental Management System (EMS) and statutory Environmental Management and Monitoring Plan (EMMP) should continue to be updated to include the project expansion aspects and impacts. Environmental program specific management plans and SOPs should include new permit requirements.
- The mine closure plan will need to be reviewed and updated to reflect material changes following the expansion.

2. INTRODUCTION

2.1 Introduction

This report is an Independent Technical Report dated 1 January 2024 of the Updated Integrated Development Plan (IDP) for K92 Mining Inc.'s Kainantu Gold Mine Project (the 'Kainantu Project') in Papua New Guinea. The Updated IDP includes the Kainantu Stage 3 Expansion Definitive Feasibility Study (DFS) Case and the alternative Kainantu Stage 4 Expansion Preliminary Economic Analysis (PEA) Case.

In January 2024, K92 Mining Inc. (K92) requested Entech Mining (Entech), WSP Engineering Ltd. (WSP), ATC Williams Pty Ltd. (ATCW), EMM Consulting Pty Ltd. (EMM), and Metallurgical Management Services Pty Ltd (MMS) and GR Engineering Services Ltd. (GRES) to prepare a report in accordance with National Instrument 43-101 - Standards of Disclosure for Mineral Projects ('NI 43-101') incorporating the results of recently completed mineral resource estimates dated 12 September 2023 and mine scoping studies of the Kora Expansion Project (the Project).

H&S Consultants Pty Ltd was engaged to prepare a mineral resource estimate for the Kora deposit, including the K1, K2 and Judd deposits.

Entech were engaged by K92ML to prepare an updated mineral reserve estimate for the Kora deposit, including the K1, K2 and Judd deposits. This included a mining study for the development of the deposits including a financial model to provide guidance in relation to the economic viability of the Project.

MMS were engaged by K92ML to manage the metallurgical testwork programme including interpretation.

GRES have been engaged for the detailed design and construction of the new processing plant designed to treat 1,200,000 tpa of ore from the Kora deposit. Given this their insight has been used to provide accurate information needed to inform infrastructure, process recovery methods, and capital / operating costs.

WSP Engineering have been engaged by K92ML to carry out a front-end engineering on the selected mine backfill option of a paste plant.

ATC Williams was engaged to carry out a feasibility study on the tailings storage facility.

EMM Consulting was engaged to carry out an analysis of the environmental and community aspects relating to the project.

K92 (the Issuer) intends that this report be used as an Independent Technical Report as required under Part 4 'Obligation to File a Technical Report' of NI 43-101 to support publicly disclosed information.

2.2 Basis of Technical Report

This Technical Report has been compiled by K92ML from the sections prepared and signed off by the Qualified Persons (QPs - identified below), to prepare a Definitive Feasibility Study and a Preliminary Economic Assessment in accordance with Canadian National Instrument NI 43-101.

The qualified persons (QPs) responsible for sections in this Technical Report are as follows:

- Andrew Kohler (K92 Mining Inc.) responsible either wholly or partly for report Sections: 1 to 12, 23, 25, 26, and 27.
- Simon Tear (H&S Consultants Pty Ltd) responsible either wholly or partly for report Sections: 1, 14, 25 and 26.
- Evan Kirby (Metallurgical Management Services Pty Ltd) responsible either wholly or partly for report Sections: 1, 13, 25 and 26.
- Daniel Donald (Entech Mining) responsible either wholly or partly for report Sections: 1, 15, 16, 21, 24, and 25.
- Brendan Mulvihill (GRES Ltd.) responsible either wholly or partly for report Sections: 1, 17, 21, and 24.
- Isaac Ahmed (WSP Ltd.) responsible either wholly or partly for report Sections: 1, 13, 18, 24, 25 and 26.

- Ralph Holding (ATC Williams) responsible either wholly or partly for report Sections: 1, 18, 21, 24, 25 and 26.
- Nicholas Currey (EMM Consulting) responsible either wholly or partly for report Sections: 1, 20, 25 and 26.

Other sections have been provided by the Company.

2.3 Property Inspections by QPs

A summary of the QP site visits is detailed in Table 2-1.

Table 2-1 Summary of QP Site Visits

Qualified Person	Site Visit
Andrew Kohler	08/05/22 - 12/06/22 24/08/22 - 09/09/22 18/10/22 - 27/10/22 08/02/23 - 02/03/23 15/06/23 - 25/06/23 07/08/23 - 21/08/23
Simon Tear	21/10/18-23/10/18 21/06/23-25/06/23
Evan Kirby	09/06/22-12/06/22 29/05/24-02/06/24
Daniel Donald	29/05/24-31/05/24
Brendan Mulvihill	8/8/2023-23/08/23
Isaac Ahmed	N/A
Ralph Holding	2004
Nicholas Currey	N/A

2.4 Effective Dates

The Effective Date of this report is 1 January 2024. There were no material changes to the scientific and technical information of the Project between the Effective Date and signature date of this Report.

2.5 Abbreviations

Abbreviation	Description
3D	Three dimensional
a	annum
AACE	Association for the Advancement of Cost Engineering
AAS	Atomic Absorption Spectrometry
AE	Acoustic Emission
AMD	Acid mine drainage
ANC	Acid neutralising capacity
ANCOLD	Australian National Committee on Large Dams
ARD	Acid rock drainage
As	Arsenic
Au	Gold
AuEq	Gold Equivalent
BDP	Business Development Plan
BILA	Bilimoia interim landowner association

Abbreviation	Description
BLA	Bilimoia Landowners Association
BOO	Build Own Operate
BOOT	Build Own Operate Transfer
BQR	Budget Quotation Request
CA	Concession Agreement
CAE Fusion	Geological Data Management System
CAPEX	Capital cost estimate
CEPA	Conservation and Environment Protection Authority (PNG)
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CRM	Certified Reference Material
°C	Degree Celsius
COG	Cut-off grade
d	day
DB	Dry bulb
dB	decibel
DD	diamond drilling
DFS	Definitive Feasibility Study
DMIRS	Department of Mines, Industry Regulation and Safety (Western Australia)
DMT	dry metric tonnes
DO	dissolved oxygen
DPM	Diesel Particulate Matter
E&I	Electrical and Instrumentation
ECP	Environmental Code of Practice
EHP	Eastern Highlands Province
EIR	Environmental Inception Report
EIS	Environmental Impact Statement
EMP	Environmental Management Programme
EMS	Environmental Management System
EP	Environmental Permit
EPCM	Engineering, Procurement and Construction Management
ESIA	Environment and Social Impact assessment
F80	80% of a unit process feed particle size is below a given size, based on particle size distribution (PSD)
FGZ	Fault Gouge Zone
FIFO	Fly-in, fly-out
FoS	Factor of Safety
FS	Feasibility Study
g	grams
g/l	grams per litre
g/t	grams per tonne
GISTM	Global Industry Standards on Tailings Management
hr	hour
HQ	Exploration drill size (96 mm OD / 63.5 mm ID)
GAT	Gravity Amenability Test
GDMS	Geographical Data Management System
GED	General Exploration Drilling
GHG	Greenhouse gas
GP	General purpose (cement)
GPS	Global Positioning System
GST	Goods and Services Tax
ha	Hectare

Abbreviation	Description
HARD	Half Normal Distribution
HI-Cell	Hollow Inclusion Cell
HLS	Heavy Liquid Separation
HPL	Highlands Pacific Limited
hr	Hour
HV	High voltage
ICP-MS	Inductively Coupled Plasma Mass Spectrometry
IPR	Independent Peer Review
IRR	Internal Rate of Return
IRS	Intact Rock Strengths
ITC	Integrated Tool Carrier
JC	Joint Conditions (JC)
JS	Joint Spacing
K	PNG Kina
K92 / K92ML	K92 Mining Limited
km	kilometre
km^2	square kilometres
kPa	kilopascal
kV	kilovolt
kWh	kilowatt hour
l	litre
l/s	litre per second
LHOS	Longhole Open Stope
LHS	Longhole Stope
LOM	Life of Mine
LTC	Land Titles Commission
LV	Low voltage
M	million
m	metre
masl	metres above sea level
MCC	Motor control centre
MIK	Multiple Indicator Kriging
min	minute
ML	Mining lease
ml	millilitre
mm	millimetre
Mm^3	million cubic metres
mo	month
MOA	Memorandum of Understanding
MoM	Ministry of Minerals
MPa	megapascal
MRA	Mineral Resources Authority (PNG)
MRE	Mineral Resource Estimate
MS	Microsoft
MSO	Mineable Stope Optimiser
mS/m	milli Siemens per metre
Mt	million tonne
M tpa	million tonne per annum
NAF	non-acid forming
NAG	net acid generation
NPV	Net Present Value

Abbreviation	Description
NQ	Exploration drill size (75.5 mm OD / 47.6 mm ID)
OHL	Overhead line
OMC	Orway Mineral Consultants
OPEX	Operating cost estimate
oz	Ounce - 31.10348 grams
P&ID	Piping and Instrumentation Diagram
PAF	potentially acid forming
PSD	Particle Size Distribution
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PIR	Passive Infra-red
PLC	Programmable Logic Controller
PNG	Papua New Guinea
PPE	Personal protective equipment
ppm	parts per million
PQ	Exploration drill core size (122.6 mm OD / 85 mm ID)
P_{80}	80% of a unit process product particle size is below a given size, based on particle size distribution (PSD)
pH	Potential of Hydrogen - measure of acidity
QAQC	Quality Assurance Quality Control
QEMSCAN	Quantitative evaluation of minerals by scanning electron microscopy
QP	Qualified Person (in accordance with Canadian National Instrument NI 43-101)
RAR	Return air riser
RC	Reverse Circulation
RF	Revenue Factor
RFQ	Request for Quotation
RMR	Rock Mass Rating
ROM	Run-of-Mine
RQD	Rock Quality Designation
s	second
SAB	SAG and Ball mill
SAG mill	Semi Autogenous Grinding mill
SD	Standard Deviation
SG	Specific Gravity
SLC	Sub level caving
SMP	Structural, Mechanical and Piping
SPP	Supply and Procurement Plan
t	metric tonne (1,000 kg)
TDS	Total Dissolved Solids
tkm	Tonne-kilometer
TOR	Terms of Reference
tpa	Tonnes per annum
TSF	Tailings Storage Facility
TSS	Total Suspended Solids
TWA	Time weighted average
UG	underground
VAT	Value Adding Tax
VSD	Variable Speed Drive
WASM	West Australian School of Mines
WB	Wet bulb

Abbreviation	Description
WSF	Water Storage Facility
µS/cm	micro Siemens per centimetre
µm	Micron (1/1000 of a millimetre)

3. RELIANCE ON OTHER EXPERTS

The authors of this report are not qualified to provide comment on the legal issues associated with the Project, including any agreements, joint venture terms and the legal status of the exploration permits, and mining tenure included in the Project.

All input into this report has come from qualified persons, with no reliance on other experts.

4. PROPERTY DESCRIPTION AND LOCATION

The Kainantu property covers a total area of 836.8 sq km and is located in the Eastern Highlands Province of Papua New Guinea, approximately 180 km west-northwest of Lae (Figure 4-1). The project is located at the approximate centre of the Property, at 6°06′25″ S Latitude and 145°53′27″ E Longitude.

4.1 Tenure

The property comprises six exploration licences (EL470, EL693, EL1341, EL2619, EL2620 and EL2753), one mining licence (ML150), two mining easements (ME80 and ME81), and one licence for mining purposes (LMP78). Tenements are owned 100% by K92 Mining Limited (K92ML) but there is an understanding in-place for a 5% share to be divested to the local landowners and the Eastern Highlands Province. A tenement map is shown in Figure 4-1 and tenement details are summarised in Table 4-1. The Project as described herein is 100% owned by K92 Mining Limited (K92ML); a company incorporated in Papua New Guinea, which is 100% owned subsidiary of K92 Holdings International Limited (K92 Holdings).

K92 Mining Inc. (formerly Otterburn) is a company incorporated under the laws of British Columbia, Canada; the common shares of which are publicly listed on the Toronto Stock Exchange.

There has not been any title search or due diligence undertaken on the tenement titles or tenement conditions. The tenement's status has not been independently verified other than a viewing of tenement information on the PNG Mineral Resource Authority website.

K92ML is the registered holder of the following tenements in PNG (MRA, 2020), as issued by the applicable government authorities in accordance with the PNG Mining Act 1992 (the 'Mining Act'):

Mining Lease 150 (ML150) was renewed for a period of 10 years to 13 June 2034. All conditions of the lease renewal have been satisfied including commencement of production from the Kora deposit before 30 June 2018.

Mining Easements 80 and 81 (ME80 and ME81) was renewed for a period of 10 years to 13 June 2034.

Licence for Mining Purposes 78 (LMP78) was renewed for a period of 10 years to 13 June 2034.

Exploration Licence 470 (EL470), effective until 4 February 2023. K92ML have lodged an application for renewal for a further two years.

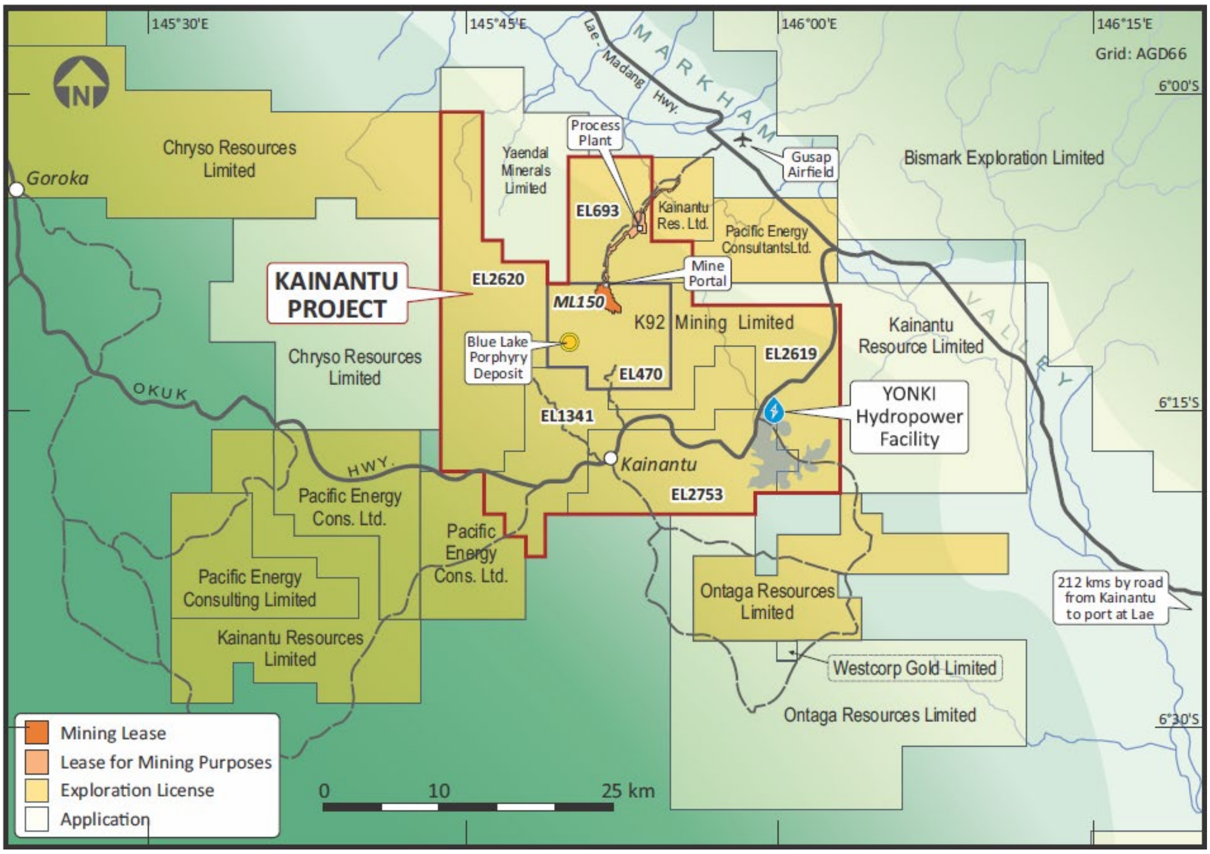
Exploration Licence 693 (EL693), effective until 4 February 2023. K92ML have lodged an application for renewal for a further two years.

Exploration Licence 1341 (EL1341), effective until 20 June 2024. K92ML have lodged an application for renewal for a further two years.

Exploration Licence 2619 (EL2619), effective until 22 January 2024. K92ML have lodged an application for renewal for a further two years.

Exploration Licence 2620 (EL2620); effective until 2 June 2025. K92ML have lodged an application for renewal for a further two years.

Exploration Licence 2753 (EL2753); effective until 20 December 2025.



Source: K92ML (2022)

Figure 4-1 Kainantu Project Location and Tenements

Table 4-1 Project Tenure Details

Tenement No.	Grant Date	Expiry Date	Renewal Application Date	Area (km2)	Owners#
EL470	05/07/1982	04/02/2023	03/11/2022	92.65	K92ML
EL693	29/12/1986	04/02/2023	03/11/2022	95.45	K92ML
EL1341	21/06/2004	20/06/2024	01/03/2024	146.63	K92ML
EL2619	23/01/2020	22/01/2024	13/10/2023	159.70	K92ML
EL2620	03/06/2021	02/06/2025	Current	201.19	K92ML
EL2753	21/12/2023	20/12/2025	Current	136.40	K92ML
ML150	14/06/2002	13/06/2034	Current	2.88	K92ML- 95% Landowners- 5%**
ME80***	14/06/2002	13/06/2034	Current	0.30	K92ML
ME81***	14/06/2002	13/06/2034	Current	0.35	K92ML
LMP78***	14/06/2002	13/06/2034	Current	2.09	K92ML

Source: K92ML (2024)

*** Ownership of ML150 currently 100% K92ML. 5% pledged under commercial terms to Landowners in the 2003 Memorandum of Understanding and ratified by the 2014 K92ML Purchase Agreement.*

**** ME80, ME81 and LMP78 are linked to the current ML150.*

4.2 Expenditure Commitments

Table 4-2 Expenditure Commitments

Tenement No.	Term End Date	Commitment Year 1* (PGK)	Commitment Year 2* (PGK)	Proposed Work Program Budget	
				Unit	Amount
EL470	04/02/2025	2,400,000	2,400,000	PGK	4,800,000
EL693	04/02/2025	500,000	500,000	PGK	1,000,000
EL1341	20/06/2024	600,000	600,000	PGK	1,200,000
EL2619	22/01/2024	350,000	350,000	PGK	700,000
EL2620	02/06/2025	150,000	250,000	PGK	400,000

4.3 Royalties

The Mining Act 1992 (Act) provides that all minerals at or below the surface of any land (i.e. gold, silver, copper and other minerals) are the property of the State. K92ML, pursuant to the Mining Lease from the State, owns what is mined from the orebody.

The tenements are subject to royalties and interests in favour of the Government of Papua New Guinea in accordance with the Mining Act 1992 (Act). The holder of a mining lease or a special mining lease under the Act is required to pay a royalty to the State equal to 2% of either:

The Free on Board (FOB) value of the minerals if they are exported without smelting or refining in Papua New Guinea.

The Net Smelter Return from the minerals if they are smelted or refined in Papua New Guinea.

No other royalty agreements exist over the tenement package.

While not strictly a royalty cost, the PNG government imposes a second cost on mining projects, that of the MRA Levy. This levy is 0.5% of mine revenue (there are no deductions allowed for concentrate transport, smelting and refining).

4.4 Other Significant Factors and Risks

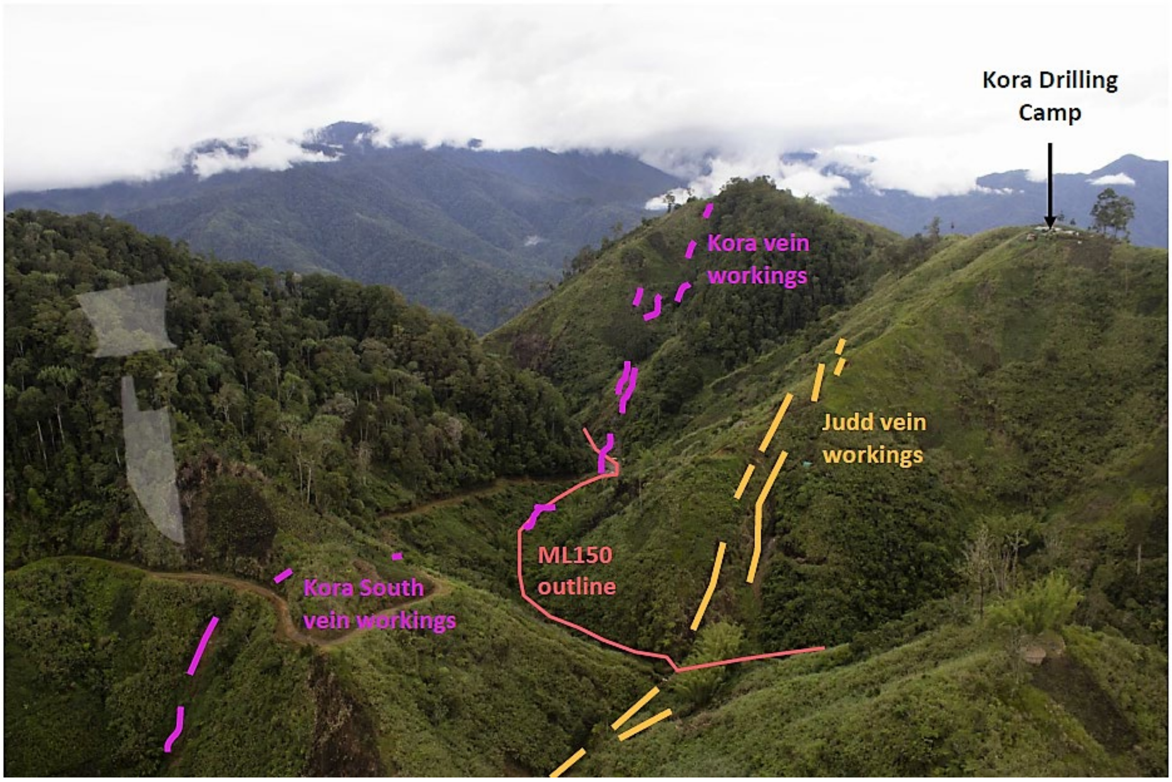
Under the laws and upon grant of a mining licence (ML) or a special mining licence (SML) the State may elect at its discretion to take, at sunk cost, up to a 30% participating interest in any major mineral development in PNG. Upon exercise of that option, the State will fund its share of capital and ongoing costs and the mine developer will be repaid its share of sunk costs. In respect of ML150, the State waived its right to acquire a 30% interest in the existing mining licence when they were first granted and has no similar rights under the ML renewal process. However, the State retains the option in respect of the Exploration Licences should any be converted into a Mining Licence or Special Mining Licence.

Environmental permitting, mine closure plans, and landowner compensation agreements are discussed in Section 20 'Environmental Studies, Permitting, and Social or Community Impact' of this report.

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

5.1 Physiography

The property lies within an area of mostly rugged topography, with transecting rivers forming lower lying areas. Also includes areas of low lying areas within the Markham Valley. Elevations range from 400 m to 3,130 m above sea level. Vegetation is mostly primary rainforest with areas of shifting cultivation in valley floors.



Source: K92ML (2021)

Figure 5-1 Kainantu Project Topography

5.2 Access

The property area is accessed by a two-hour drive along the sealed Lae-Madang Highway from Lae. Lae is the capital city of the Morobe Province and second largest city in PNG. It is serviced by daily flights from Port Moresby and other PNG centres and also, hosts the largest cargo port in PNG.

The property is serviced by a 10 km long formed access road from the Lae-Madang Highway, commencing at Gusap Airstrip to the Kumian Process Plant and Office facility. The access road crosses one single lane bridge at the Ramu River. From the process plant site, a formed haul road travels 6.5 km to the 800 Lower Portal of the mine. The haul road crosses three major single lane bridges.

Access and haul roads span 6 m width and are constructed within two Mining Easements (ME's 80 and 81) commencing at the Ramu Bridge. The haul road rises 391 m in elevation over its total length. These roads are graded and reformed on a continual basis and have subsequently not deteriorated significantly in high rainfall seasons.

5.3 Climate

The climate at Kainantu has the Köppen classification of Af (tropical rainforest) with hot temperatures and wet conditions. Daytime temperatures reach 30°C dropping to night-time lows of 20°C. A pronounced wet season occurs between November and April, although rainfall is common throughout the year. Rainfall averages 235 mm / mo during the November to April wet season, and 137 mm / mo during the dry season. Annual rainfall averages approximately 2,000 mm. Project operations and exploration activities can be impacted by weather activities. Cloudy conditions may prevent the operation of helicopters, and in some circumstances heavy rainfall or earthquakes can trigger localized landslides.

5.4 Local Resources

The property site offices are located 180 km from Lae, 21 km from Kainantu township and 56 km from Goroka. Goroka is the Capital of Eastern Highlands Province and contains Local and Provincial Level Government Offices.

5.5 Power

Yonki Dam provides water for the Ramu Hydro Power Station and the Yonki Toe of Dam Power Station operated by PNG Power Ltd. Currently the Ramu 1 Hydro Power station is supplying 54 MW from three generators on to the Ramu Grid while the Yonki Toe of Dam supplies 14 MW. They are supplemented by 4 MW from the Pauanda Hydro Power station, 10 MW from the Baiune Hydro Power station at Bulolo in Morobe Province and a combined thermal generation capacity of 20 MW from the diesel power stations in Lae, Madang, and the Highlands centres, giving a total generation capacity of 102 MW into the Ramu Grid (PNG Power website, 2014).

The primary source of power to the property is the PNG Power national grid (PPL) from the Ramu sub-station, located 20 km from the processing plant site. Power from the national grid is reticulated to site via 22 kV overhead line and services the plant, mine, and camp area. The property also has standby diesel generators capable of supplying the total requirements of the operation.

5.6 Gusap Airstrip

The Gusap Airstrip is a licenced grass landing strip located in the Ramu Valley and maintained jointly by the project and Ramu Agricultural Industries mainly for weekly charter flights and if needed emergency situations.

5.7 Infrastructure

The Kainantu mine is located within ML150 and the main Kainantu mine camp and processing plant are located within LMP78 which is located within EL693. The Property includes all mine infrastructure, exploration camps, exploration data and diamond drill core storage.

The property is well supported by regional infrastructure and contains all the necessary site infrastructure for mining operations.

Underground mining at Kainantu initially operated from 2004 to 2008. Following resumption of mining operations in 2016 underground mining infrastructure has been rehabilitated and refurbished. Since March 2019, the mining fleet has undergone significant expansion and modernization with an increase in the quantity of equipment, and a significant increase in size of equipment. The underground mine infrastructure was considerably upgraded and expanded during 2019 and this process is ongoing.

The Kainantu processing plant is located approximately 7 km from the opening of the 800 portal which accesses the Irumafimpa and Kora deposits. The plant was on care and maintenance between December 2008 and September 2016. Refurbishment of processing plant was completed in September 2016 and the first batch of underground ore from Irumafimpa was treated by K92ML in October 2016. In February 2018 K92ML declared commercial production at the Kainantu mine and mine production focused on the Kora North area. Upgrades of site infrastructure continued in 2019 and 2020. The treatment plant was expanded to a capacity of 400,000 tpa in 2020.

Further details of site infrastructure can be found in Section 18 Project Infrastructure.

6. HISTORY

There has been a significant amount of exploration on the property by various owners.

Modern exploration did not commence until the early 1980s. After the discovery of the Irumafimpa deposit, Highlands Pacific Limited (HPL) focused on high grade Au-telluride mineralization with little to minor work conducted on the porphyry Cu / Au targets. HPL commenced mining operations on the Irumafimpa deposit in 2004.

Barrick purchased the tenement package from HPL in late 2007 and concentrated on increasing Mineral Resources at Irumafimpa-Kora and discovering economic porphyry Cu-Au mineralization, this work successfully identified porphyry style targets however, little drilling was achieved, prior to the mining operation at Irumafimpa and regional exploration activities elsewhere being put on care and maintenance between January 2009 and August 2016 when K92ML commenced rehabilitation of the Irumafimpa mine and processing plant and later discovered Kora North mineralization that subsequently underpinned the mine development.

Further information on the past ownership and historic exploration activity at Kainantu is contained in the 'Independent Technical Report, Resource Estimate and Summary of Mining Facilities, Kainantu Project, Papua New Guinea, dated 1 May 2015' which is filed on SEDAR.

6.1 Historical Resource Estimates

Several historical estimates for the Irumafimpa, Kora and Eutompi deposits, were previously prepared before K92ML entered into an agreement to acquire an interest in the property that contains these deposits. After the Kora North discovery K92ML, by drilling, has now amalgamated Kora and Eutompi deposits with Kora North into what is now termed the Kora deposit.

Mineral Resources, reported in accordance with the 2004 JORC Code and Guidelines, were prepared by independent consultants including Hackchester Pty Ltd (2005) and Mining Associates Pty Ltd (2006).

Numerous historical estimates and financial models were prepared by Barrick for the Irumafimpa and Kora deposits. K92ML is not treating the historical estimates as current Mineral Resources or Mineral Reserves. These historic resources are not reported here as they have been superseded by the MREs for the Kora and Judd deposits which were reported in detail in 'Independent Technical Report, Mineral Resource Estimate Update, Kora and Judd Gold Deposits, Kainantu project, Papua New Guinea', effective date 20 January 2022 and in Section 14 of this report. The Irumafimpa MRE is reported in Woodward, (2015).

6.2 Historic Production 2006 to 2008

During the mining operation at Irumafimpa between 2006 and 2008, mining was predominantly shrink stoping with some longhole stoping. The method applied was based on the geological structure and varying vein widths. Multiple independent reviews have shown that previous operators had considerable difficulty with dilution issues during mining which has been mainly attributed to the geological complexity of the veins and a poor understanding of grade distribution within the veins.

Table 6-1 shows mill production for the life of the mine from 2006 to its closure in 2008. The grade control grades were of the order of 8 to 9 g/t Au whereas the back calculated mill head grade for 2008 was 5 g/t Au.

Table 6-1 Kainantu Mill Production 2006 to 2008

Year	Throughput (t)	Head Grade (Au g/t)	Gold Produced (oz)
2006*	104,272	8.00	26,819
2007*	141,452	7.00	31,835
2008**(6 mo)	61,532	5.02	9,939
LOM Total	307,256	6.94	68,593
* From Highlands Pacific annual reports.			
** Barrick Ownership (mining and processing ceased in December 2008).			

6.3 Care and Maintenance 2009 to 2016

In January 2008, Barrick sought to place the mine into care and maintenance. Barrick received approval to have the mine in care and maintenance via the variation to the approved purposes for Mining Lease No. 150 dated 13 February 2009, which was subsequently extended until February 2013, when the Mining Advisory Council determined that extension of care and maintenance was appropriate provided a Mine Closure Plan was submitted.

K92 Mining Ltd (K92ML) commenced the refurbishment and rehabilitation of the mine, process plant and related infrastructure in February 2016. The Company received approval from the Mineral Resources Authority (MRA) to recommence mining operations in October 2016.

Remedial work on the 800 Portal and Incline, the main mine access for the Irumafimpa mine, was completed in June 2016 with the upper working levels of the mine accessible and ventilation re-established. Refurbishment of the Kainantu Processing Plant was completed in September 2016 and the first batch of underground ore from Irumafimpa treated in October 2016. In early 2017, the mine shipped the first concentrates containing gold and copper to the Port of Lae for shipment overseas for smelting and refining. K92ML announced the declaration of commercial production effective 1 February 2018.

6.4 Production 2016 to 2023

K92ML restarted mining operations in the Irumafimpa mining area. Limited mining activities were undertaken in the lower parts of Irumafimpa during 2017, with mineralized material being mined from development headings and from stopes. A small amount of low-grade ore was also recovered from remnant stopes.

Table 6-2 shows mill production since March 2016. K92ML started the Kora mine project by completing the underground incline drive from Irumafimpa to Kora and commencing underground drilling.

In late 2017 initial exploration drilling to the south of Irumafimpa identified mineralization in the area between Irumafimpa and Kora; in the area initially referred to as Kora North. In September and October 2017 K92ML mined a bulk sample from the Kora North area and processed the material through the existing plant for metallurgical evaluation, with +90% recovery achieved for both gold and copper. In early 2018 mining activities ceased at Irumafimpa, and the focus of mining changed to development of the Kora North deposit.

Annual production for 2023 achieved 117,607 oz AuEq or 100,533 oz gold, 3,488 tonnes copper and 160,628 oz silver. In total gold Production since 2016 has achieved 532,860 oz of gold.

Table 6-2 Kainantu Mill Production 2016 to 2023

Year	Mill tonnes	Head grade Au g/t	Contained Oz Au
2016**	633	3.41	69
2017	61,932	4.47	8,900
2018 ***	79,487	19.1	45,810
2019	127,190	20.8	79,838
2020	230,365	14.0	95,109
2021	336,221	9.8	95,055
2022	448,087	8.3	107,546
2023	503,484	6.8	100,533
** K92ML Restart, rehabilitation, refurbishment and commissioning from March 2016			
*** K92ML Commercial Production from February 2018			

A general timeline of operations to date at Irumafimpa-Kora is shown in Table 6-3.

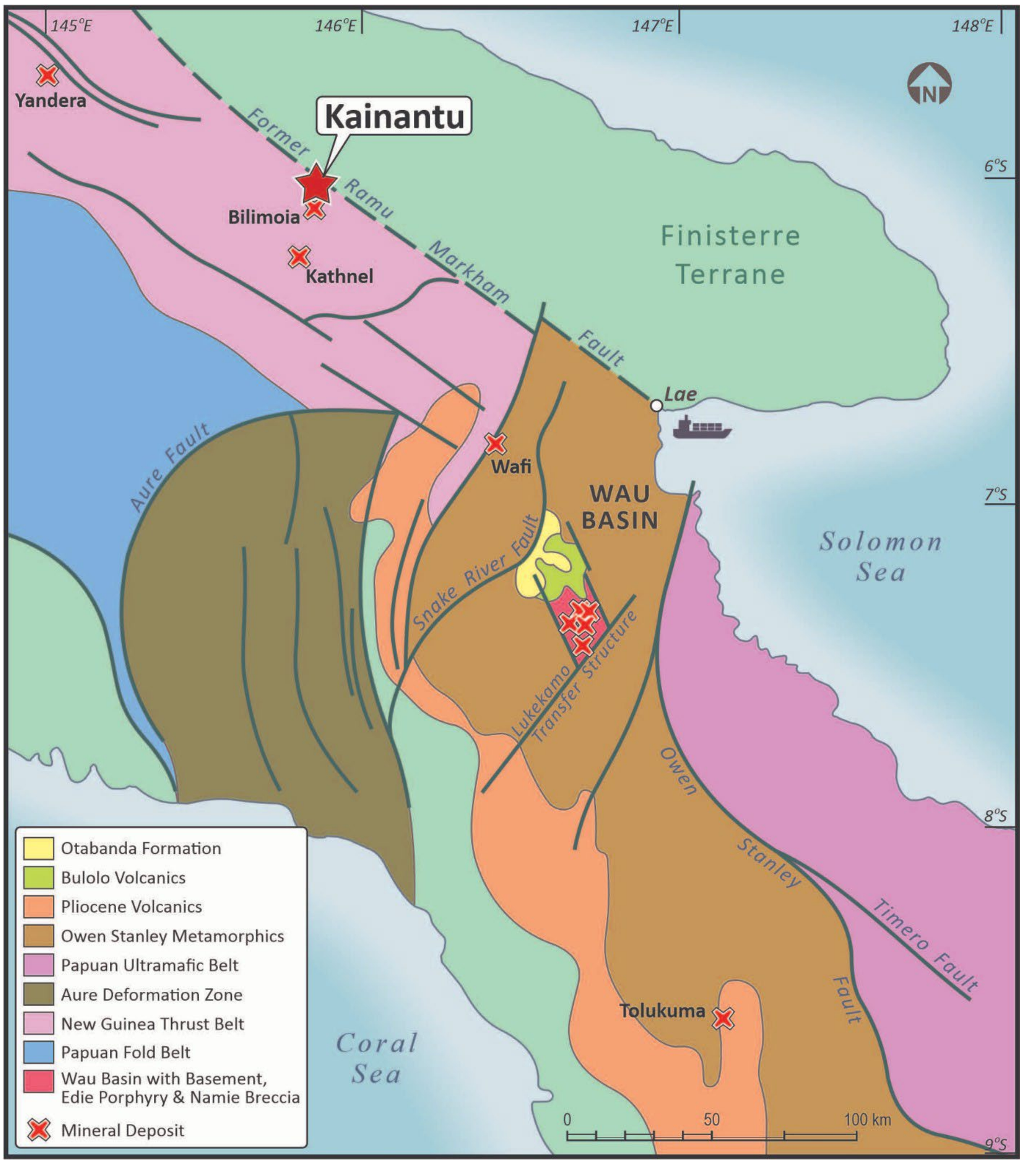
Table 6-3 Summary of Operations Timeline for the Kainantu Project

From	To	Mine Operations History (ML150)
January 2004		Highlands Pacific DFS approved by Mineral Resources Authority
2005	October 2007	Kainantu Gold Mine operated by Highlands Kainantu Limited (HKL)
November 2007		Barrick purchased the Kainantu project.
January 2008	December 2008	Mining was suspended from January to June 2008, restarted in July 2008 and was halted permanently in December 2008.
January 2009	December 2009	Exploration of epithermal and sulphide veins continued on the ML until June 2009, and then halted due to review of exploration priorities.
January 2010	December 2014	Project on Care and Maintenance, limited exploration on EL's. K92PNG acquired K92ML from Barrick (Niugini) Limited pursuant to an agreement dated June 11, 2014 which closed March 6, 2015.
January 2015	January 2018	Mining Lease granted. Operations restarted with rehabilitation of mine, refurbishment and re-commissioning of the processing plant.
April 2017	May 2017	K92 Mining Ltd discovers Kora North deposit, with KMDD0009.
February 2018	Current	K92ML declared commercial production at Kainantu mine and production focused on the northern Kora area.
July 2020	Current	Judd 1 Vein discovered, 1235 Drive commences as a vent drive, followed by drilling from underground on the Judd Vein system. Production continues from K1 and K2
January 2021	December 2023	Systematic surface, diamond drill programs. The KUDD and KODD drill hole series for the Kora Deposit.
January 2022	June 2023	Stage 2A production reached, of the Integrated Development Plan.
June 2023	Current	Progression to Stage 3 of the Integrated Development Plan.

7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Kainantu region is in the north-eastern flank of the northwest trending Papuan Mobile Belt which is a major foreland thrust belt (Rogerson et al., 1987). The regional structural package of the Kainantu district is bounded in the northeast by the northwest trending Ramu-Markham Fault, a major suture zone that marks the northern margin of the Australian Craton, and in the southeast by the Aure Deformation Zone (Figure 7-1). Many of the major structures in the New Guinea Thrust Belt represent crustal-scale thrust faults and host fragments of obducted oceanic crust. The belt is characterised by sub-horizontal to shallowly north-dipping late Miocene stacked thrust sheets of regionally metamorphosed and strongly cleaved Triassic to Eocene fine-grained sedimentary rocks and minor volcanic rocks. Following a middle Oligocene hiatus, siliciclastic sediments, carbonates, and volcanic rocks were deposited until thrusting began in the middle Miocene (Rogerson et al, 1987; Dobmeier et al, 2012) accompanied by middle Miocene intrusions. A mild orogeny in late Tertiary time folded and faulted Tertiary rocks and has continued to the present day (Dow and Plane, 1965). The belt is characterised by several north-northeast trending fault zones that commonly host major ore deposits (Williamson & Hancock, 2005).



Source: K92ML (2021)

Figure 7-1 Regional Geology of Papua New Guinea, showing location of Kainantu property

7.2 Property Geology

The Kainantu area is underlain by rocks of the Early Miocene Bena Bena Formation, comprising pelite, psammite, conglomerate and marl beds metamorphosed to greenschist or amphibolite grade.

(Table 7-1 and Figure 7-2). These are unconformably overlain by Miocene age Omaura Formation consisting of volcano-lithic sandstones and siltstones and numerous fossiliferous limestone lenses. The overlying Yaveufa Formation consists of basaltic and andesitic flows, agglomerates, volcanoclastic sandstone and limestone (Tingey and Grainger, 1976). The mid-Miocene Akuna Intrusive Complex consists of multiple phases ranging from olivine gabbros, dolerites, hornblende gabbros and biotite diorites to granodiorites. Late Miocene age Elandora Porphyry-related intrusions formed small high level crowded feldspar porphyry dykes and diatreme breccias associated with gold and copper mineralization. A north-northeast trending set of transfer structures transects the area, with associated mineralization, alteration and porphyry complexes often aligned along them.

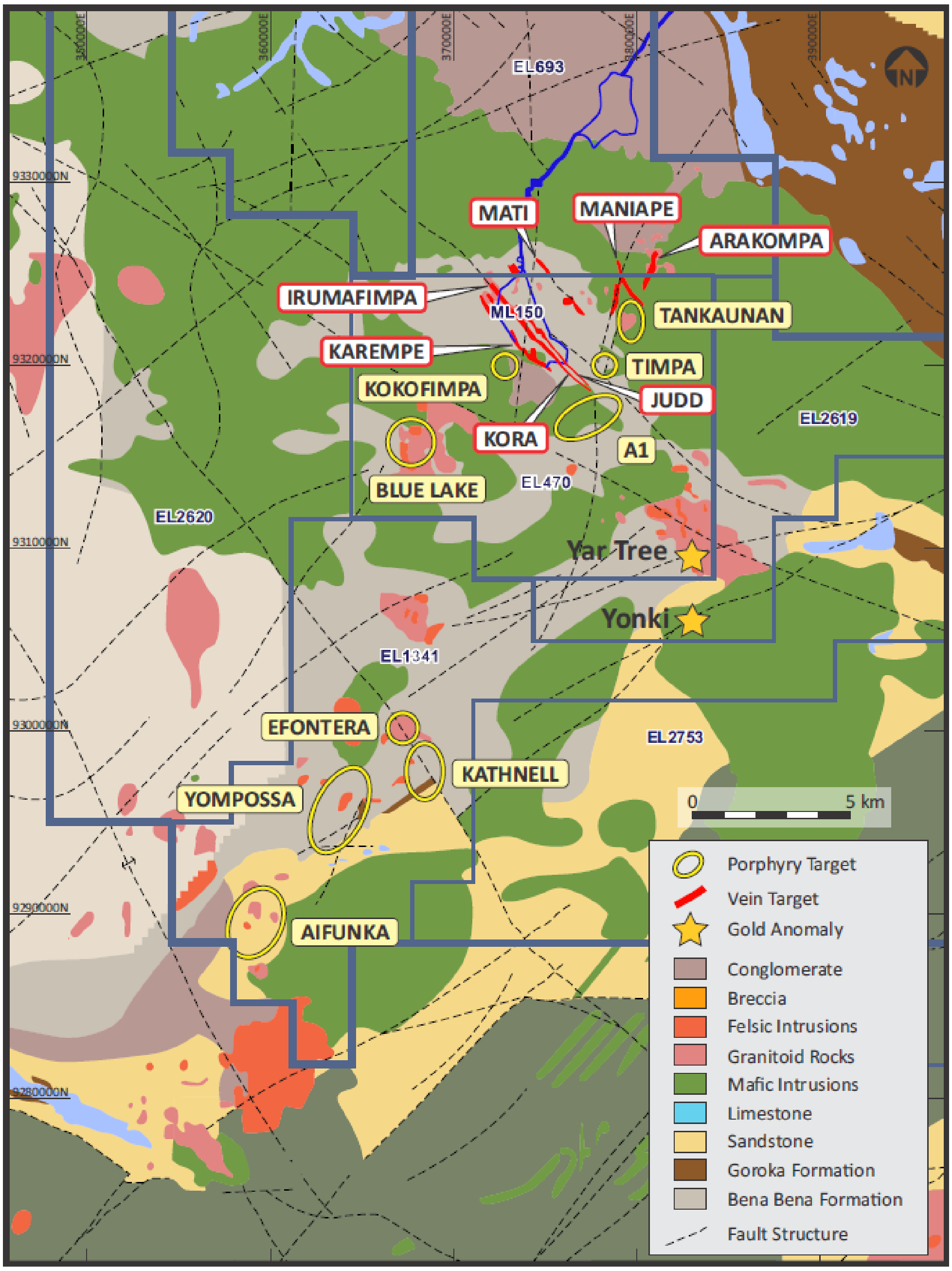


Figure 7-2 Geology of the Kainantu area

Table 7-1 Main regional rock units identified within the Kainantu area.

Age	Rock Units
Recent Quaternary	Kainantu Formation - basal fluvial conglomerate, lithified sandstone and mudstone overlain by well bedded tephra.
~~ Unconformity ~~~	
Late Miocene	Elandora Porphyry - intermediate dykes, sills and stocks.
Early Miocene	Akuna Intrusive Complex - range in composition from olivine gabbros through to granodiorites.
Early Miocene - Mid Miocene	Yaveufa Formation - basaltic and andesitic agglomerates, lithic tuffs, volcanoclastic sandstone and limestone.
Late Oligocene - Late Miocene	Omaura Formation - thin bedded to laminated calcareous siltstone and mudstone.
~~ Unconformity ~~~	
Early Mesozoic	Bena Bena Formation - pelite, psammite, conglomerate and marl metamorphosed to schist and phyllite. Steeply dipping sequence with shearing mineralization accompanying isoclinally folded faults and breccias. There are late stage dykes of andesite in places, intruding the Bena Bena formation, these are thought to be associated with the Elandora Porhyry.

7.3 Mineralization Overview

The dominant host rock of the Kora - Irumafimpa vein systems is the highly sheared and deformed Bena Bena Formation, composed of low grade metamorphosed phyllites and amphibolites, intruded by the Elandora porphyry at the northwest end of the vein system. Mineralization on the property includes:

1. Au-Cu-Ag mineralization in low sulphidation epithermal Au-telluride veins (Irumafimpa),
2. Au-Cu-Ag quartz-sulphide veins of Intrusion Related Gold Copper ("IRGC") affinity at Kora,
3. Indications of porphyry Cu / Au systems (Blue Lake) and
4. Alluvial gold.

The Kora - Irumafimpa mineral zone occurs in the centre of a large mineralized system, approximately 5 km x 5 km in area, that has been subject to drilling in parts and comprises several individual zones of vein and porphyry-style mineralization. The Kora - Irumafimpa (including Kora, Eutompi and Kora North) vein deposits has been demonstrated from drilling results and underground development to be part of the same structural system.

The Kora vein system is a broad northwest trending structurally controlled mineralized zone more than 2.5 km long and up to 60 m wide. Individual veins vary from less than one metre wide with pinch and swell features over short distances to more continuous veins up to several metres wide. The mineralization is characterised by auriferous quartz-pyrite (chalcopyrite) lodes. There are gold telluride lodes at Irumafimpa.

The Kora mineral zone, includes the old Kora and Kora North mineralization, and comprises a K1 footwall lode and a K2 hanging wall lode separated by varying widths of 0 to 15 m. In parts there is interstitial mineralization between K1 and K2 known as the Kora Link lode. About 90 to 150 m to the east lies the parallel Judd lode system. comprising from grid west to east the J1W lode followed by the main J1 lode and then the J2 and J3 lodes. The mineralization associated with the Judd lodes appears to be a structural vein system similar to the Kora zone.

Major vein systems, similar to Kora and Judd, are apparent at Arakompa and Maniape, mostly within EL693 but also extending into the northern part of EL470. Both Arakompa and Maniape were drilled with a small number of short holes in the late 1980s and early 1990s and historical internal company reports indicate coherent mineralization were identified. Both projects will be the subject of significant drilling campaigns in 2024.

Peripherally to Kora mineralization, exploration activities both historically and those done by K92ML have also, identified further areas of vein and porphyry-style mineralization. A substantially sized, gold/copper mineralized porphyry system has been delineated by K92ML at Blue Lake (Kotampa). The drilling, mostly at 200m spacing, demonstrated the presence of a prominent silica-clay lithocap, overlying mineralized propylitic (epidote-chlorite) altered porphyry of dacitic composition, with higher copper and gold grades hosted by potassic alteration intersected in a number of holes.

Other less advanced prospects on the property include evidence for epithermal Au veins similar to Irumafimpa, IRGC veins similar to Kora, porphyry Cu-Au systems, skarn Cu, Pb and Zn mineralization and alluvial gold.

The location of the deposits and prospects in relation to the property boundaries is shown below, in Figure 7-3 and a plan view of the Kora, Irumafimpa and Judd veins is shown below in Figure 7-4.

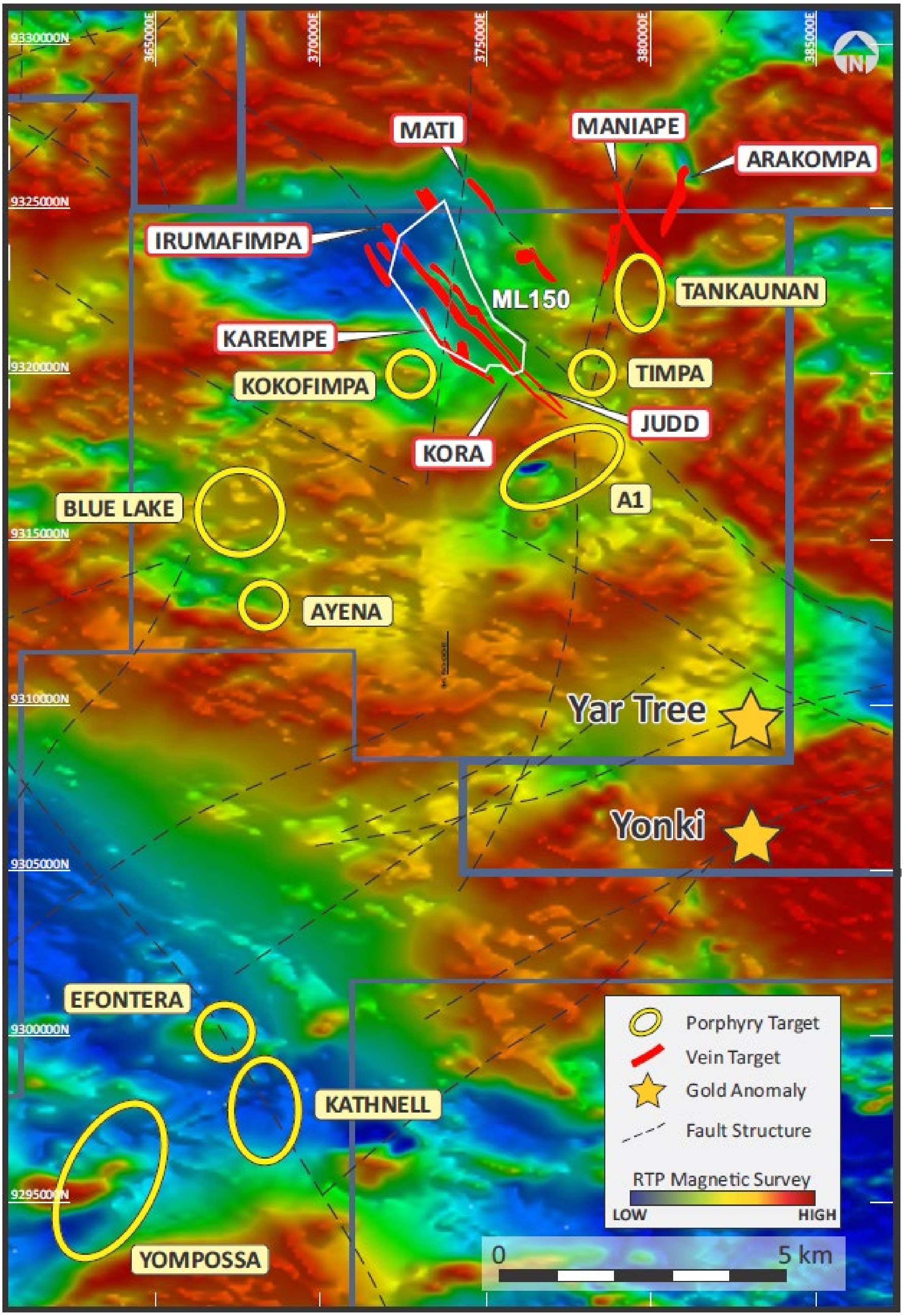
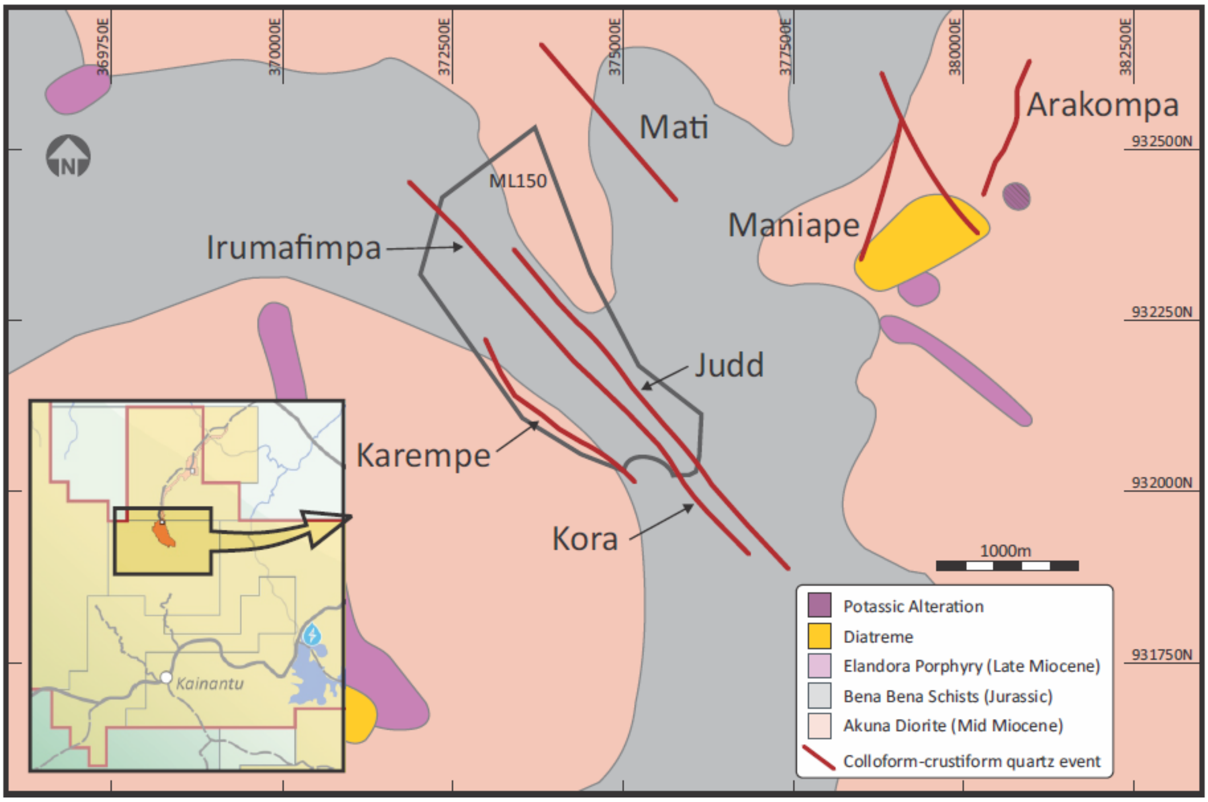


Figure 7-3 Kainantu property geology and known vein and porphyry deposits and prospects.

The prospects are summarised in Table 7-2 (K92ML 2024).



Source: K92ML (2021)

Figure 7-4 Location plan Kora, Irumafimpa and Judd vein systems

A summary of the mineralization style, host rocks, dimensions and geological continuity for the Kora-Irumafimpa vein deposits and the other vein and porphyry prospects for the Kainantu Project is shown in Table 7-2 and described further below.

Table 7-2 Summary of mineralization, host rocks, dimensions, and continuity for main Kainantu deposits and prospects

Deposit / Prospect	Mineralization	Host Rocks	Dimensions	Continuity
Irumafimpa	Epithermal Au-Te mineralization	Quartz veins in chlorite-sericite schist	Strike length of 1 km 0.5-2 m wide open at depth	Drilling indicates continuity of mineralization on a broad scale. Gold mineralization is discontinuous
Kora (Including Kora, Eutompi, Kora North and Kora south)	Vein Au-Cu intrusive vein style (described in Section 7.4)	Quartz veins in chlorite-sericite schist. Brecciated sometimes ductile deformation	>2.5 km strike (includes Kora South) x 60 m wide System is open along strike and at depth	Drilling shows strike and depth continuity at a kilometre scale. Gold mineralization is discontinuous.
Judd	Vein Quartz sulphide veining IRG Au-Cu (described in Section 7.4)	Quartz veins in chlorite-sericite schist.	2.5 km strike (includes Judd) structural system with multiple Veins as defined	Surface continuity along strike unknown due to poor

Deposit / Prospect	Mineralization	Host Rocks	Dimensions	Continuity
	(Mineral Resources reported in Section 14)		by surface mapping and sampling and sporadic drilling. Mineralization open along strike and to depth	outcrop exposure
Karempe	Vein Epithermal Au (best intercept; 2.45 m @ 39.82 g/t Au, from 238.65 m, including 0.75 m @ 125.40 g/t Au from 239.0 m, in drill hole KRDD0005)	Quartz veins in granodiorite and chlorite-sericite schist.	2 km+ strike and 1-2 m wide veins, within corridor, as defined by surface mapping and sampling. Mineralization open along strike and to depth	Surface continuity along strike unknown due to poor outcrop exposure
Arakompa	Vein Epithermal Au	Quartz veins in Akuna Diorite	3 km strike, with multiple high grade gold-rich veins amid a broad (100 m+) corridor NNE trending No deep drilling.	Continuity is partially obscured at surface, due to poor outcrop exposure
Maniape	Vein Epithermal Au	Bena Bena Metamorphics, Akuna Diorite,	Strike length 1 km Near surface zone of mineralization of at least 700 m strike x 34 m wide x 125 m depth defined by surface sampling and diamond drilling	Continuity of near surface mineralization confirmed by drilling
Mati / Mesoan	Vein Epithermal Au (Rock chips average of 28 g/t Au and a maximum of 131 g/t Au)	Bena Bena Metamorphics, Akuna Diorite,	1 km strike mineralized zone defined No drilling	Surface continuity along strike unknown due to poor outcrop exposure No drilling
A1 (reconnaissance stage)	High-sulphidation and porphyry Cu-Au Brecciated vuggy silica-pyrite-enargite mineralization and anomalous molybdenum in soils Historic float sample of massive enargite-pyrite returned 16.6% Cu and 12 g/t Au.	Bena Bena Metamorphics, Akuna Diorite, Feldspar porphyry and breccias	Undefined	Undefined

Deposit / Prospect	Mineralization	Host Rocks	Dimensions	Continuity
Kokofimpa	Porphyry Cu-Au	Akuna Intrusive Complex and Elandora porphyry intrusions within the Bena Bena Metamorphics	3 km x 3 km Defined porphyry system with multiple magmatic phases with minimal drilling in centre of prospect.	Undefined
Tankaunan	Porphyry Cu-Au	Akuna Intrusive Complex and mid-late Miocene Elandora Porphyry intrusions within Bena Bena Metamorphics	Extent of systems needs to be defined by first pass 400 x 400 m drilling.	Undefined
Timpa	Porphyry potential postulated Cu-Au-As in Soils Advanced argillic alteration Quartz Breccia (monomict, quartz cemented, with shallow quartz infill textures; soil sampling shows the breccia is anomalous in Au, As, Bi, Sb, W)	Bena Bena Metamorphics and breccia	Quartz breccia is 500 m x 100 m. Other mineralization Undefined	Undefined
Aifunka	Skarn (Porphyry-related) Cu and Au Au (Barda reefs)	Mineralization is hosted in calc-silicate bands spatially associated with the brecciated porphyry dyke contacts. Underlain by the Omaura Sediments and Akuna Intrusive Complex with Elandora Porphyry.	Undefined	Undefined

Deposit / Prospect	Mineralization	Host Rocks	Dimensions	Continuity
Yompossa	Porphyry Cu-Au (60 m @ 0.3% Cu and 0.1 g/t Au from 105 m in BHP01; 118 m @ 0.1% Cu and 0.1 g/t Au in KSDD0004)	Underlain by Bena Bena Formation and Omaura Formation. Contains feldspar porphyry intrusions interpreted to be associated with Elandora Porphyry	Anomaly is 500 m x 600 m and is open to the NE. Potential for further mineralization below existing drill holes.	Undefined
Kathnel	Base metal epithermal veins (Pb-Zn-Cu-Au)	-	Undefined	Undefined
Efontera	Porphyry Cu-Au	-	Undefined	Undefined
Blue Lake	Porphyry Cu-Au (549 Mt @ 0.61 g/t AuEq, for 10.8 Moz AuEq). Tear et al., 2022. Proof of a large mineralized system.	Feldspar porphyry intrusions within Akuna Granodiorite	Au/Cu geochem in soil anomaly is 1.2 x 0.8 km Porphyry has dimensions of 1,500 m (X) by 1,300 m (y) by 1,100 m (Z) with a modest plunge to grid south-east	Completely open to the south-east

7.4 Kora -Irumafimpa Vein Systems

The Kora -Irumafimpa vein systems are interpreted to contain two stages of mineralization based on work completed by Corbett (2009). His study used the Irumafimpa mine working exposures and Barrick's Kora and Eutompi drill results. The earliest phase is pyrite/chalcopyrite-rich veining. This is overprinted by a quartz-rich, Au-dominant crustiform quartz vein to breccia system with high grade gold associated with tellurides (e.g. Calaverite AuTe). The alteration and mineralization paragenesis recognised in the Kora-Irumafimpa vein system is summarised below in Table 7-3.

Table 7-3 Mineralization and alteration paragenesis in the Kora-Irumafimpa- vein system

Stage	Name	Description
Stage 1	Silicification and fuchsite alteration	silica, fuchsite
Stage 2	Sulphide-rich Cu-dominant	quartz, pyrite, chalcopyrite, bornite
Stage 3	Quartz-rich Au-dominant	quartz, gold tellurides (calaverite and kostivite), native gold
Stage 4	Quartz Cu	quartz, pyrite, chalcopyrite, bornite

Stage 1 is the earliest period of alteration and is characterised by silicification and fuchsite alteration of phyllitic wall rock.

Stage 2 at Irumafimpa comprises coarse-grained idiomorphic quartz and pyrite (typically euhedral) veins with base metals. Volumetrically this early mineralization appears to be the most abundant mineralization. At Kora and Judd the mineralization comprises massive pyrite veins to pyritic breccias, grading to pyrite-chalcopyrite-bornite veins characterised by elevated Zn, Pb, Sn, W, Bi, and Sb. Higher copper grades, in the order of 2% Cu, occur at Kora (Coote, 2019). There appears to be a lateral zonation to lower copper grades at Irumafimpa.

Stage 3 mineralization is the dominant gold-bearing stage and is characterised by crustiform, vuggy and colloform quartz veins, quartz breccias, and xenomorphic pyrite. At Irumafimpa most of the gold occurs as the gold tellurides, calaverite and kostivite, which are concentrated at vein margins, whilst at Kora and Judd, gold tellurides are minor and gold occurs as fine-grained particles, linked to pyrite margins and is free milling. Significant native gold has been locally observed and is probably a result of either oxidation of tellurides at Irumafimpa, or as primary native gold at Kora and Judd.

Stage 4 is manifested as local brecciation and deposition of low temperature quartz along with minor copper mineralization.

Underground face examples of the K1 mineralization are included in Figure 7-5. The upper picture is from Level 1305 and the lower picture is from Level 1130.

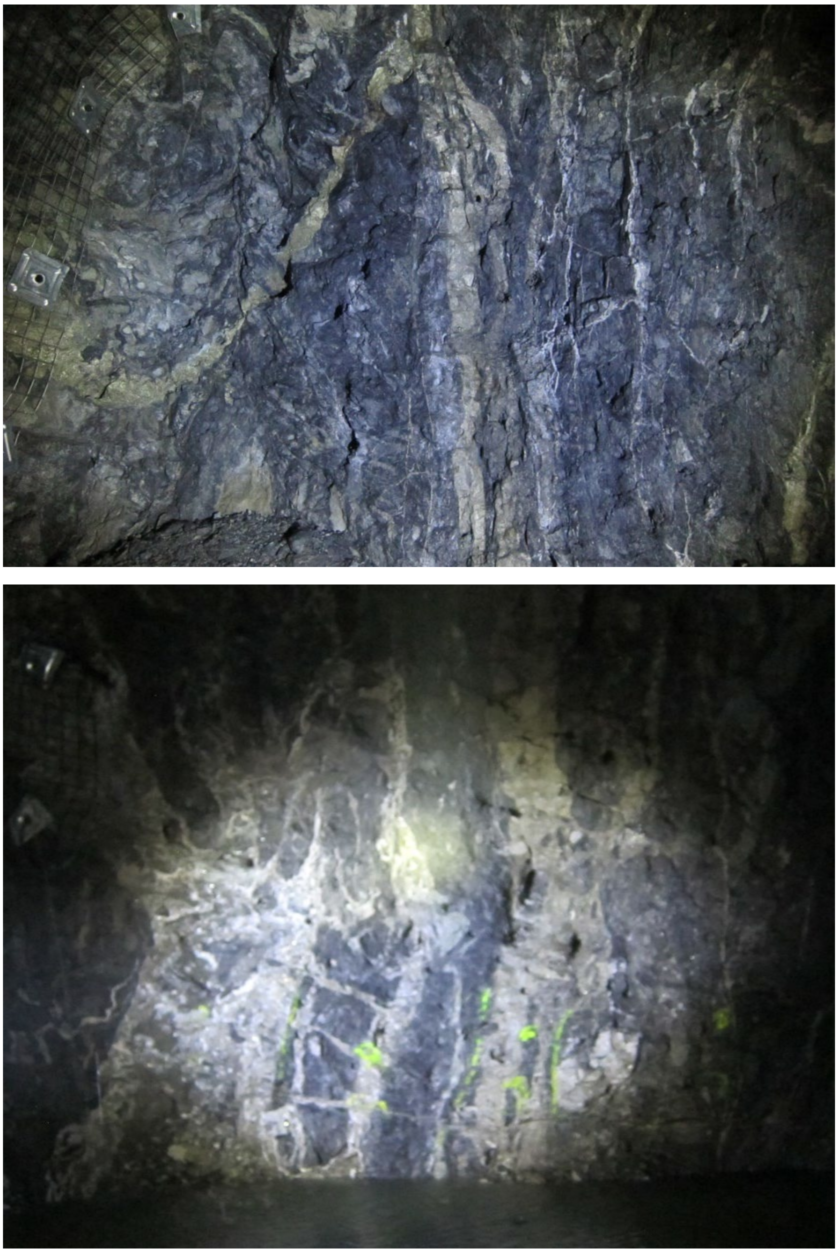


Figure 7-5 K1 Lode showing Quartz-Pyrite Veining (H&SC 2023)

7.4.1 Host Rocks

The dominant host rock of the Kora -Irumafimpa and Judd vein systems is the highly sheared and deformed Bena Bena Formation, composed of low grade metamorphosed phyllites and amphibolites, intruded by the Elandora porphyry at the northern end of the vein system.

7.4.2 Controls to Mineralization

The structural history of the Kora -Irumafimpa area has been documented by Blenkinsop (2005), this body of work was done prior to the discovery of Kora North. The Kora -Irumafimpa vein systems are made up of the contiguous Kora, Eutompi, Kora North and Irumafimpa structures. The Kora -Irumafimpa vein system generally comprises breccia veins with abundant clasts of both altered wall rock and earlier stages of vein mineralization. These are tectonic in nature and are vuggy and annealed in places, however, at Irumafimpa they are mostly hydrothermal veins of epithermal origin. Vein formation was multistage, with at least four identifiable episodes of alteration and mineralization (Table 7-3).

The Kora-Irumafimpa vein system is interpreted to be a long-lived structure, which was reactivated at several different stages in time. The mineralization shows modest variations in dip from sub-vertical to locally -60° dip and strike. This is believed to be due to the formation of dilation and rotation from pre-existing structures that later were enhanced by further structural dilatancy from continued mineral fluid injection.

Late-stage faults with clay gouge postdate the mineralization. These usually occur on the vein margins but can cause local disruption of the veins.

A crude plan or cross section schematic representation of the Kora mineral zone in miniature is presented in Figure 7-6. The picture shows a structural dilatant zone with early pyrite mineralization cut by later quartz veining with the two bounding quartz veins symbolic of the auriferous K1 on the right and K2 on the left.

Figure 7-6 Schematic Representation of the Kora Mineral Zone (H&SC 2018)

7.4.3 Dimensions and Continuity

The geological continuity of the lode system is excellent, whereas the gold grade continuity is variable throughout the lodes as is typical of this style of mineralization. The current interpretation for the Kora mineral zone is based on the 2020-2021 drilling results and supported by the 2021-2023 period of drilling. The Kora mineral zone, as represented by the parallel lodes K1 and K2, has a strike length of 2,250 to 2,500 mm and a dip length of 1,050 m to 1,350 m. The K1 lode phases out in the south particularly at depths below the 1,205 level, whereas the K2 lode remains open to the south at both shallow and deeper levels. This has allowed for K2 to be considered the dominant lode for the vein system.

The 2021-2023 drilling period allowed significant resource expansion to the south and from the surface, down dip, marrying up with the drilling from underground. Ever since the 2020-2021 drilling period, the Kora Link lode has been interpreted as an amalgamation of smaller mineralized veins into a single zone juxtaposed with both K1 (on its hanging wall) and K2 (on its footwall).

7.5 Judd Vein System

The Judd vein system is located approximately 90 to 150 m east of and parallel to Kora on ML150. The system consists of four known, narrow veins, <5 m wide, with significant inter-vein separation. The veins are steeply dipping and with a similar strike direction as Kora. Mineralization is similar to Kora i.e. structurally controlled auriferous quartz-pyrite (chalcopyrite) veins and breccias. From west to east the veins are J1W, J1, J2 and J3 with J1 being the dominant vein.

The Judd vein system was partially tested by Barrick as part of its drilling programme testing for the Kora lodes at depth. This sporadic drill testing returned a maximum intersection of 3 m @ 278 g/t Au. Core samples illustrate two types of Judd vein mineralization, a quartz dominant, Au-rich component and a sulphide dominant, Cu-rich vein style (Figure 7-7). Surface mapping and sampling has indicated a mineralized strike length of approximately 2.5 km.

BKDD0002 126.3 – 127.3m
0.9 g/t Au, 69.8 g/t Ag, 7.49 % Cu

BKDD0002 113.6 -114m
1,870 g/t Au

Figure 7-7 Judd mineralization styles (sulphide dominant elevated copper grades, left photo and quartz dominant gold enriched on the right)

The J1 lode is a 4 to 5 m wide, intrusion-related quartz-sulphide vein system and at the close-off date of the MRE 2023, has been subject to development drives on five levels (1235, 1265, 1285, 1305 and 1325 levels) totalling 2,265 m from 402 face sample lines. This work has shown significant mineralization and geological continuity with variable grade continuity. J1 has made a significant contribution to the process plant mill feed, from both development and stoping. The lode has consistently achieved similar recoveries for gold and copper to that achieved for the Kora material. Ever since the Judd maiden Mineral Resource in early 2022, K92ML have carried out diamond drilling from both underground and surface to define and extend the Judd vein system in the development area. Underground diamond drilling has concentrated on the J1 vein, with some holes being extended to intersect the other Judd veins (J2 and J3) previously documented by Barrick and HPL and which remain under-explored.

7.5.1 Host Rocks

The dominant host rock of the Judd vein system is the highly sheared and deformed Bena Bena Formation, composed of low grade metamorphosed phyllites and amphibolites, intruded sometimes by thin intermediate intrusives of dacitic composition in sub-concordant positions.

7.5.2 Controls

The Judd veins are intrusion-related vein style and are typically brecciated with abundant clasts of both altered wall rock and earlier stages of vein mineralization. The veins are tectonic in nature and are sometimes vuggy in places. An underground face example of the J1 mineralizationis included as Figure 7-8.

Figure 7-8 J1 Lode showing Quartz-Pyrite Veining (H&SC 2023), (width of image 3 m)

The structures containing the quartz-sulphide mineralization at Judd strike NNW and dip at 75 to 800 to the west. This is likely to be a structure which was reactivated at several different stages in time. The mineral lodes show modest variations in dip and strike and this is believed to be due to the formation of dilatancy from pre-existing structures that were later enhanced by continued mineral fluid injection.

7.5.3 Dimensions and Continuity

The dimensions of the Judd mineralization interpreted for the September 2023 MRE are 800 m in strike length and 500 to 700 m in the dip direction. The Judd vein system has also been interpreted where it has been sporadically intercepted in diamond drill holes from surface and underground in the ML150 and passing into EL470.

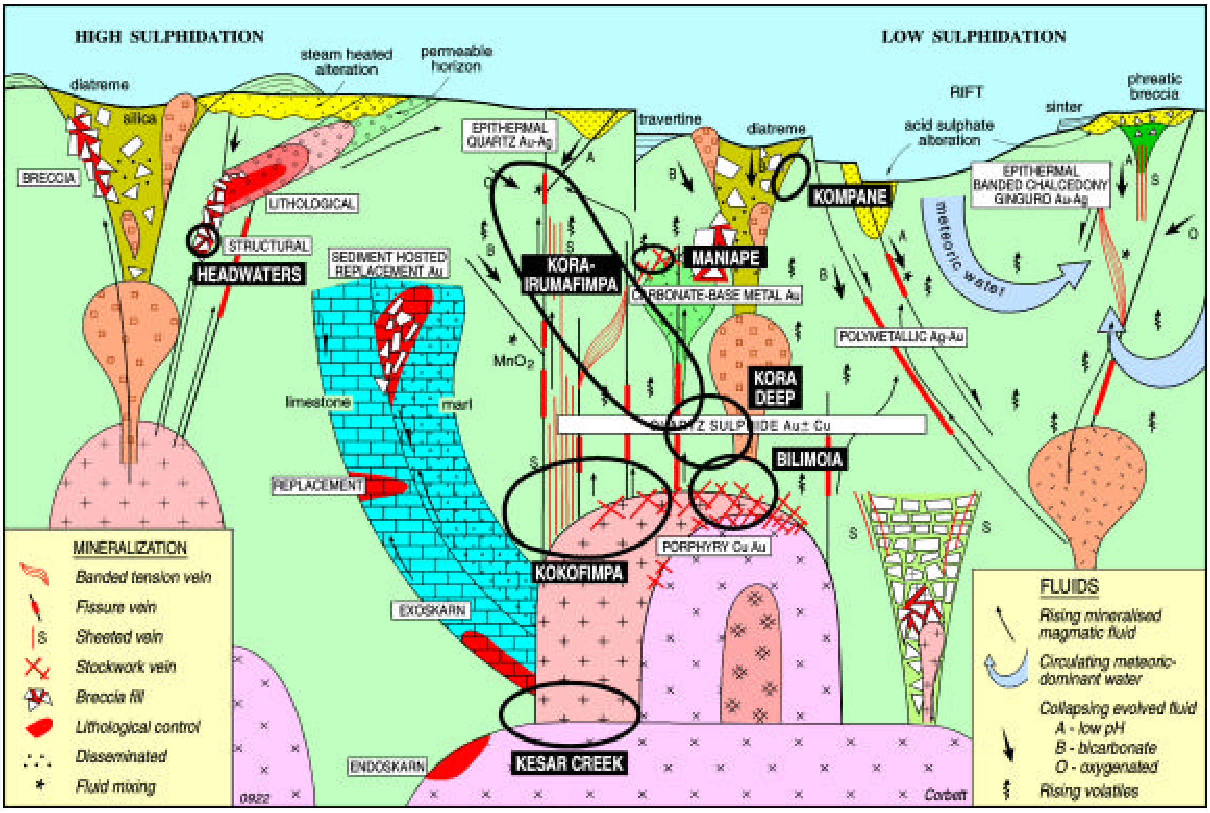
The J1 gold grade continuity and mineralization pinches and swells along the drive. The gold grades vary with ranges of similar grades typically extending between 5 and 30metres along the drive, the variability is usual for this type of mineralization.

8. DEPOSIT TYPES

Gold-copper deposits within the SW Pacific Magmatic Arcs have been classified into groups by Corbett and Leach (Corbett and Leach, 1998; Corbett, 2009):

- Porphyry-related (including gold skarn).
- High sulphidation gold-copper.
- Low sulphidation (including sediment-hosted replacement).
- Intrusion-related Gold Deposits.
- Carbonate-base metal Au Deposits.

Telescoping may overprint the varying styles of low sulphidation gold mineralization upon each other or upon the source porphyry intrusion. Some of K92ML's Kainantu prospects occur in Corbett's model and are highlighted in Figure 8-1.



Source: modified by K92ML after Corbett 2009

Figure 8-1 Conceptual model for porphyry and related low and high sulphidation mineralization.

9. EXPLORATION

9.1 Historic Exploration

A summary of historic exploration on ML150 (Irumafimpa, Kora, Judd, and Karempe) including drilling is reported in Section 6 of this report. Further exploration information at other prospects at Kainantu is described in Section 6 of the "Independent Technical Report, Resource Estimate and Summary of Mine Facilities, Kainantu Project, Papua New Guinea" dated 15 April, 2016 and in Section 6 of the "Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment for Expansion of the Kainantu Mine to Treat 1M tpa from the Kora Gold Deposit, Kainantu Project, Papua New Guinea", dated 27 July 2020 which are filed on SEDAR.

9.2 Exploration by K92 Mining 2015-2023

In September 2016 K92ML recommenced underground exploration work targeting the Irumafimpa deposit and the Judd vein system.

In May 2017, underground drilling by K92ML commenced at Kora North. Diamond drill hole, KMDD0009, intersected an extension of the Kora / Eutompi vein system approximately 130 metres from the Kora Incline. Following discovery of the Kora North mineralization and the delineation of an initial Mineral Resource, the emphasis of K92ML's drilling has focused on extending the Kora North mineralization above and below the 2018 Mineral Resource. Ever since 2017, drilling in conjunction with underground mining has continued with expansions to the Kora Mineral Resources. Drilling from 2020 to the 2022 MRE concentrated on infilling the Resource to upgrade it to Measured and Indicated category for the DFS. Post the 2022 MRE the drilling has concentrated on resource expansion and overall, there was a small increase in the Measured and Indicated resource category for the Kora Lodes, coupled with an expansion of the Inferred Resource category to the south. Also, a large focus was placed on the J1 Lode, this was infilled to increase the Measured and Indicated category as well as expand the Inferred Resource.

In addition to the Kora Mine exploration K92ML's exploration team has prioritised several targets in the general periphery of the mine (listed below in Table 9-1) for follow-up work.

Table 9-1 K92ML Priority Exploration Targets

Porphyry Targets	Epithermal Targets/Deposits
Tankaunan	Irumafimpa Extension (Kokomo)
Kokofimpa	Kora (includes Kora South)
Timpa	Judd (includes Judd South)
A1 (Headwaters)	Karempe
Blue Lake (Kotampa)	Maniape
Efontera	Arakompa
Kathnell	Mati / Mesoan
Yompossa (Yanabo)	
Aifunka	

9.3 ML 150 (Kora-Judd)

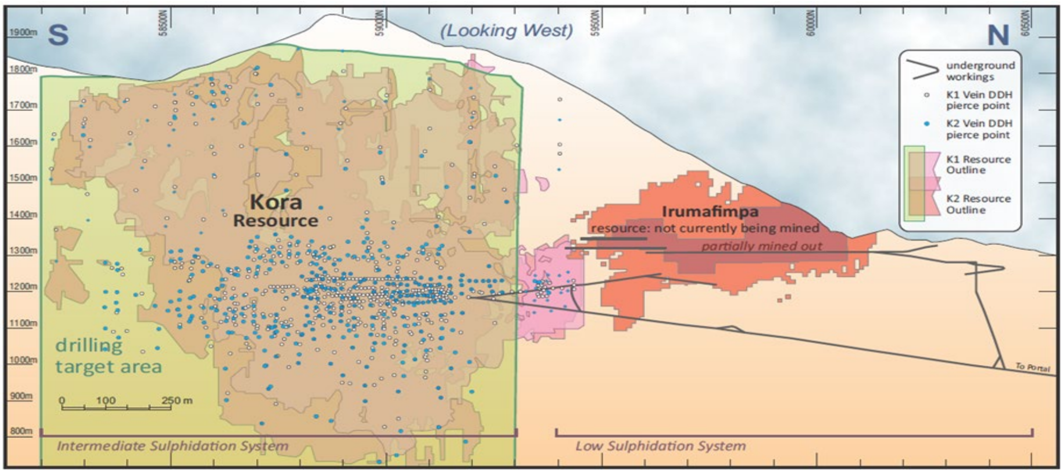
In September 2016 K92ML instigated two diamond drill rigs to commence work underground at Irumafimpa. In 2017 mining and drilling was put on hold at Irumafimpa to give priority to mining and exploration activities at the Kora North deposit because of logistics, higher grades of gold and copper and improved processing properties of the ore material.

In May 2017, K92ML commenced underground diamond drilling at Kora North. Diamond drill hole, KMDD0009, intersected what K92ML interpreted as an extension of the Kora / Eutompi vein system approximately 130 m from the Kora Incline. The intersection of 5.4 m at 11.68 g/t gold, 25.5 g/t silver and 1.33% copper was approximately 500 m along strike to the north and 150 m down dip from the closest point of what was the then defined as the Inferred Resource of the Kora deposit. Follow-up drilling expanded the discovery such that a Kora North Mineral Resource could be delineated in September 2018.

K92 Mining Limited’s previous underground diamond drilling has been described in the Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment of Irumafimpa and Kora Gold Deposits, Kainantu Project, Papua New Guinea, dated 2 March 2017, the Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment of Kora North and Kora Gold Deposits dated 7 January 2019 and the Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment for expansion of the Kainantu mine to treat 1M tpa from the Kora Gold Deposit, Kainantu Project, Papua New Guinea, dated 27 July 2020. Also, the Independent Technical Report Mineral Resource Estimate Update Kora and Judd Gold Deposits, Kainantu Project, Papua New Guinea, dated 20 January 2022 which are all filed on SEDAR.

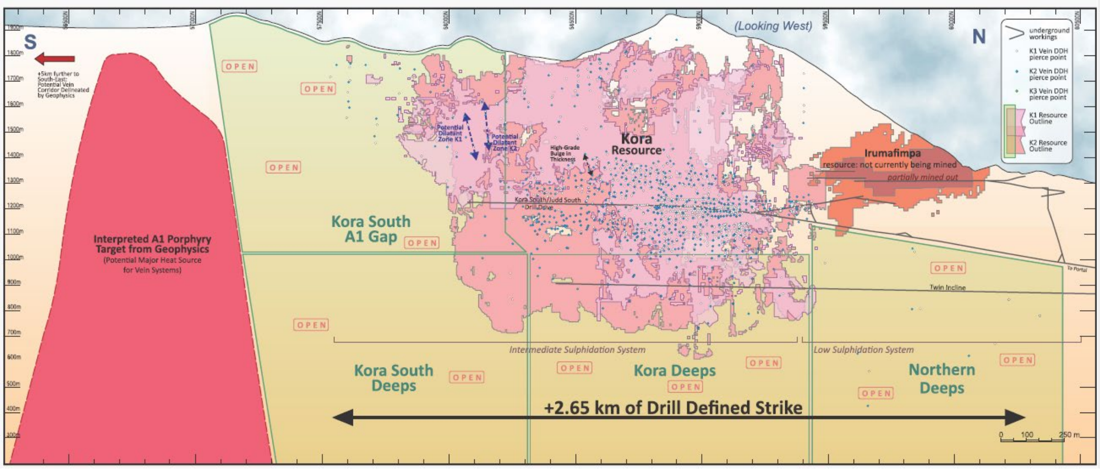
Since 2018 both surface and underground diamond drilling has continued, which has resulted in a substantial expansion of the Mineral Resource announced for all MRE updates in 2020, 2021 and now in 2023. Mineral Resources for both Kora and Judd have increased each time in the face of mine depletion.

Figure 9-1 is a schematic longitudinal section showing the location of the Kora diamond drilling as of October 2021 at the time of the previous MRE and Figure 9-2 shows the Kora drilling as of September 2023. For the current, September 2023 MRE, drilling was designed to be of sufficient density to raise confidence in the resource classification and expand the resource primarily at depth and to the grid south.



Source: K92ML (2021)

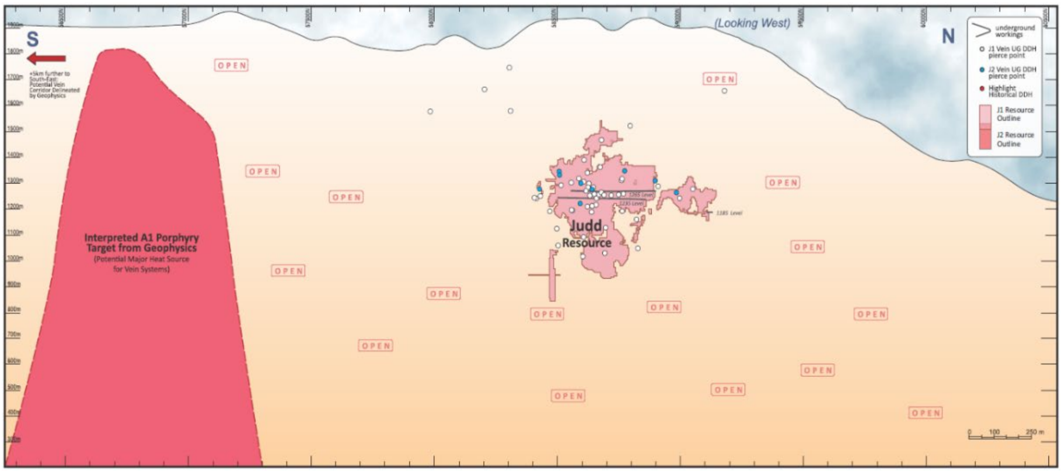
Figure 9-1 Resource Long Section - Irumafimpa, Kora and the drilling target area, the Kora Resource Estimate as at October 2021



Source: K92ML (2023)

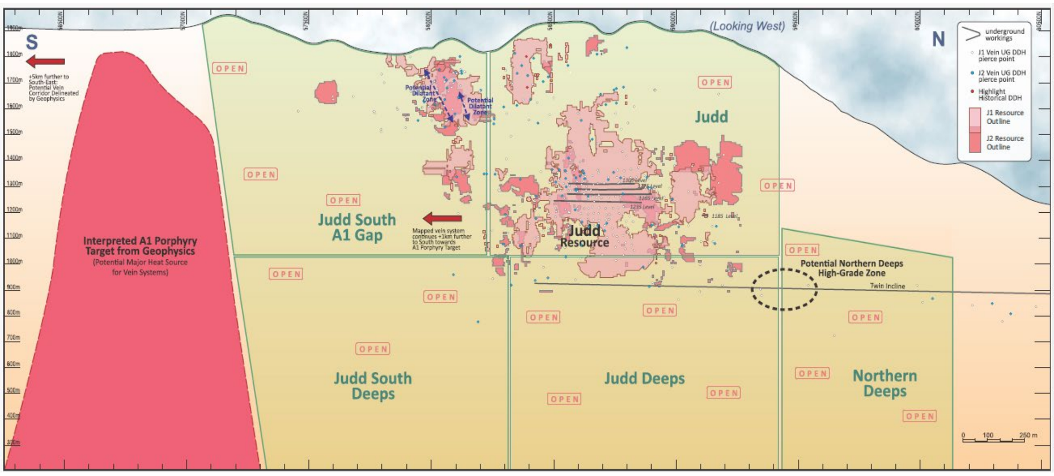
Figure 9-2 Resource Long Section - Irumafimpa, Kora, drilling target area and l intercept pierce points as of September 2023

Judd drilling first culminated in an MRE in December 2021. This is displayed in Figure 9-3 below. The updated J1 Resource long section for the September 2023 MRE is in Figure 9-4. The J2 Maiden Resource is also displayed in Figure 9-4 below.



Source: K92ML (2021)

Figure 9-3 Resource Long Section - J1 including pierce points and resource outline, as of December 2021



Source: K92ML (2023)

Figure 9-4 Resource Long Section - J1 and J2 including pierce points and resource outlines, as of September 2023

9.4 EL470 - Kora South

The Kora structure has not been closed off to the south and at depth. It has been traced along strike with drilling for more than 1km beyond the ML150 mining lease boundary and is believed to extend for a further 1km beyond that still within K92ML's exploration license EL470. This area is referred to as the Kora South prospect. The prospect has numerous artisanal workings and mineralized outcrops, some of which line up with the projected strike of the Kora structure. Underground and surface drilling, south of the ML150 boundary on EL470, commenced on the Kora South prospect in late 2021.

The extent of the structure has been highlighted by a recent geophysical survey, as previously discussed in the Kora MRE 2021, and recounted here.

The program was flown in November 2021 and engaged Expert Geophysics Limited (EGL) to conduct the helicopter-borne MobileMT electromagnetic and magnetic survey. MobileMT is the latest generation of airborne AFMAG technologies, designed in 2017 by the inventor of the ZTEM system. MobileMT measurement frequency range is 25 Hz - 30,000 Hz, while ZTEM range is 25 Hz - 720 Hz, thus delivering a much greater depth range of investigation. Electromagnetic and magnetic data was collected along east-west survey lines, nominally spaced at 200 m, and north-south tie lines nominally spaced at 2,000 m.

The airborne deep penetrating geophysics program completed over the entire property, showed good correlation between known mineral deposits and conductive bodies. Figure 9-5 shows a zone of high Apparent Conductivity contours extending from Kora and Judd into EL470 for several kilometres and Figure 9-6 is a longitudinal section illustrating the highly conductive zone north and south of the mining centre at Kora.

The results demonstrate an extensive untested potential strike length to Kora-Kora South and Judd-Judd South vein systems beyond the A1 Porphyry to the southeast for several kilometres. New vein and porphyry targets on K92ML's licenses were also identified as well as highlighting known mineralized porphyries, including A1, Blue Lake, Tankaunan and Timpa.

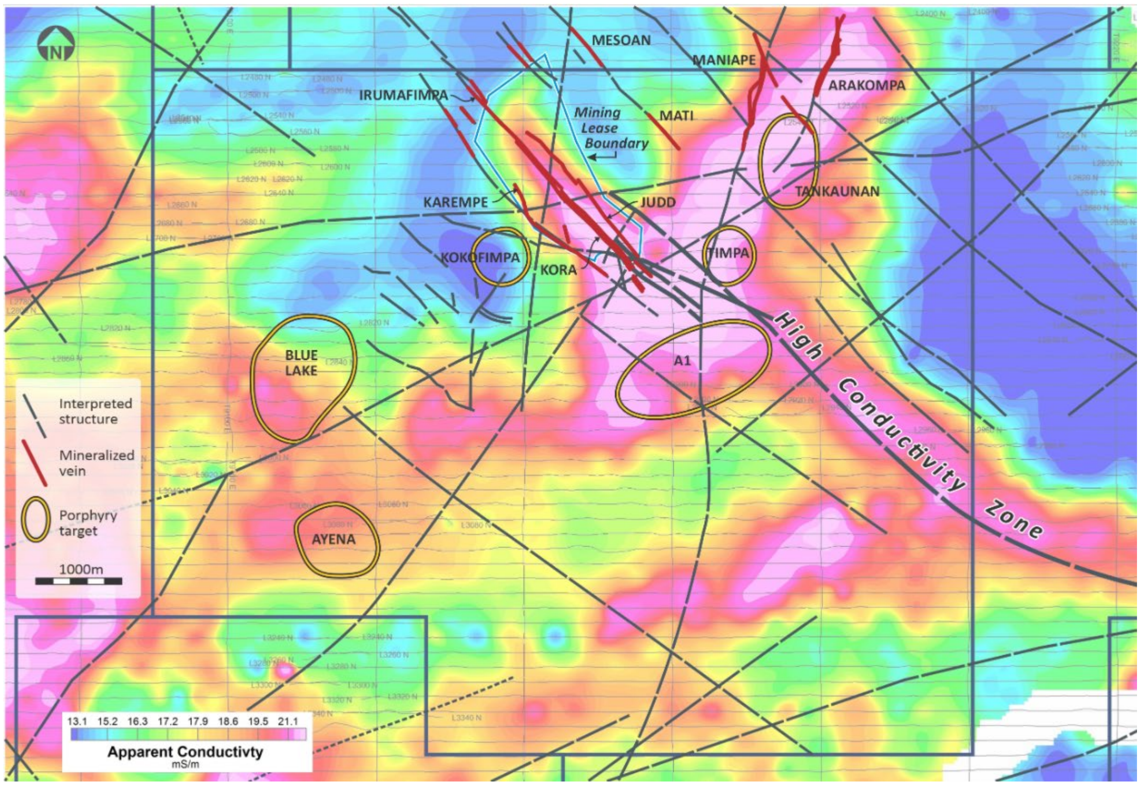


Figure 9-5 Plan View of Apparent Conductivity Contours

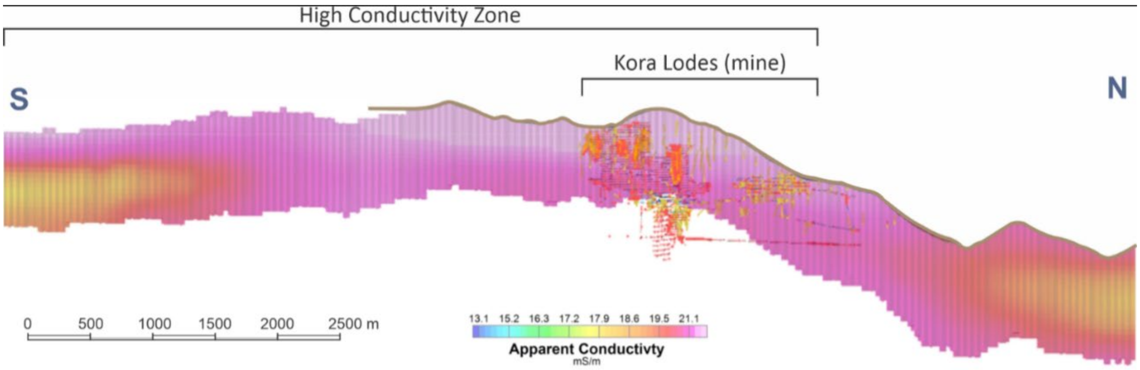


Figure 9-6 Longitudinal Section of Apparent Conductivity Contours

10. DRILLING

10.1 Kora / Judd-Drilling

The updated Mineral Resources for Kora and Judd detailed in Chapter 14 of this report is based on diamond drill hole samples from both surface and underground drilling along with face sampling of underground development drives.

Table 10-1 provides summary details of the sampling for the overall deposit area. The majority of the recent K92ML drilling (December 2021- September 2023) has been for resource expansion for Inferred Resources. Infill drilling to increase the amount of Measured and Indicated Resource for the MRE has been moderate but has provided replenishment after depletion. The drilling achieved resource expansion by extending the Kora mineralization along strike to the south. Drilling into Judd was aimed at both infill and expanding the Mineral Resource from both underground and surface.

Table 10-1 Summary Details of Sampling Methods

Company	Year	Location	Type	No of Holes	Metres	Ave Length (m)
Barrick	2008 - 2015	Surface	DD	24	10,690	445.4
		UG	DD	6	808	134.7
Highlands & Others	1990 - 2007	Surface	DD	79	16,596	210.1
	(Irumafimpa)	UG	DD	562	26,514	47.2
			Total	671	54,608	
K92ML	Oct 2017 - Sept 2018	UG	DD	83	9,564	115.2
			Face	461	1,499	3.3
K92ML	Oct 2018 - Feb 2020	Surface	DD	16	7,390.	461.9
			DDwedge	3	719	239.9
		UG	DD	96	21,225	221.1
			DDwedge	7	935	133.6
			Face	312	1,658	5.31
K92ML	Mar 2020- Oct 2021	Surace	DD	21	6,155	293.1
		UG-Kora	DD	231	36,153	156.5
			DDwedge	3	472	157.2
			Face	509	2,799	5.5
	Mar 2020- Dec 2021	UG_Judd	DD	49	8,936	182.4
			Face	193	1,060	5.5
K92ML	Oct 2021- Aug 2023	Surface	DD	53	22,134	417.6
		UG-Kora	DD	184	38,232	207.8
			Face	425	2,439	5.7
	Dec 2021- Aug 2023	UG-Judd	DD	159	31,089	195.5
			Face	209	1,205	5.8
			Kora Total Drilling	727	154,477	
			Kora Total Face	1,707	8,394	
			UG-Judd Total Drilling	208	40,025	

Company	Year	Location	Type	No of Holes	Metres	Ave Length (m)
			UG- Judd Total Face	402	2,265	
			Grand Total Drilling	1,576	237,612	
			Grand Total Face Sampling	2,109	10,659	
			(DD = diamond drilling; UG = underground)			

Drilling and sampling methods are described in detail below.

The K92ML underground diamond drilling programs for Kora and Judd deposits have utilised up to 6 owner operated LM90 drill rigs as well as using rigs supplied by Quest Exploration Drilling (QED) Contractors (PNG). QED provided two drill rigs, the Atlas Copco Diamec U4 and U6 drilling machines for this purpose. QED also carried out the surface drilling using Atlas Copco CS6 and CS1000 rigs, coupled with two owner operated drill rigs. These are Multi Power Discovery HD - Heli-portable deep hole drilling rigs. All drilling was carried out on a day and night 12-hour shift basis.

The underground diamond drilling utilises several core sizes, namely LTK60, NQ, NQ2 and HQ. For the surface drilling all holes were collared to various depths in PQ, followed by HQ and NQ casing off where needed to maintain competent samples through the lode system.

K92ML follows an established drilling protocol in which the driller prior to drilling is given a drill hole work plan specifying the expected lode target positions, hole depth, azimuth, dip, core size and drilling method to use, the plan also highlights any safety concerns such as proximity to workings. Rig setups are supervised and checked by K92ML geologists prior to the commencement of drilling. All drilling was monitored to ensure that all precautions were taken so that the diamond core recovered from the barrel was maintained in the best possible condition to maximise the information obtained by minimising breakage and core loss.

All underground diamond drill hole collar positions are surveyed in, by qualified surveyors prior to drilling and picked up after hole completion. This was done using a Leica Total Station TS09 Plus instrument, from K92ML's involvement in the project in 2016 to late 2021. From late 2021 to now, a Leica TS16 has been used. The hole design is uploaded into the instrument and set out in the field. A reference (azimuth) line is marked at the foresight and backsight. The drill collar is also marked. The drilling dip is positioned using a clinometer. A geologist checks the alignment of the rig on each hole before drilling is allowed to start. For hole pickup, a rod is inserted by the surveyor, into the hole at the collar and two points along the rod are surveyed to determine the initial drilled azimuth and dip. All surface drilling hole collars have been set up by a geologist using a tape and compass. A Garmin GPSMAP 64sx using circa 10-minute, waypoint averaging is used by the geologist to survey the collar coordinates. As a check, approximately 10% of these have been picked up by a qualified surveyor using a Leica TS16 instrument. However, recently, for both underground and surface collar setups an Azimuth aligner tool (Reflex TN14 Gyrocompass) is being used for setup in the azimuth as well as the dip direction of the hole collar.

Down hole surveys were taken for every diamond drill hole. Azimuths and inclination were measured using a Reflex EZ TRAC XTF capable of single and multi-shot operations. (This instrument has been used since K92ML involvement with the Project and began use in 2016. However, recently a Reflex GYRO SPRINT-IQ (OMNI42) is being trialled on underground diamond drill holes. During the drilling if hole conditions allowed, using the EZ TRAC XTF, single digital survey shots were done at 3, 9, 30, 60, 90m and so on including an end of hole shot. On the way back out of the hole, if hole conditions allowed, a multi-shot digital survey was carried out at 3m intervals along the entire hole. To avoid erroneous readings aluminium extension rods were used to place the instrument away from magnetic interference caused by the drilling equipment. Azimuth and dip results from surveys are automatically calculated and displayed on the handheld device for the geologist to use, for checking the downhole deviation is going as planned. Subsequently, the handheld device uploads the data into the cloud and is emailed to the Geology Team. After verification of the results, the Database Geologist loads the data into the database.

No core orientations were done for the underground drilling by K92ML as the results of the closed spaced drillholes clearly showed the geological continuity of each lode. Some core orientations were performed for HQ and NQ core on the holes drilled from surface by K92ML, however orientations were ceased because of issues correlating the orientation line. HPL and Barrick did minimal orientated core from surface and none from underground, siting difficulties of maintaining the orientation line between orientation readings because of the brecciated, often broken-up nature of the core samples through the mineralized zones.

Diamond core was laid out in the core trays by the drilling contractor / company driller, beginning in the upper left corner of each tray with respect to the long axis of the trays. The core trays were labelled with the hole number, tray number, start and finish depths. Plastic and wooden core blocks marked the end of each run with its downhole depth. Any core loss was recorded on the core block and run timesheet by the driller. Core was then removed from the drill site by the drilling teams and taken to the core shed for processing.

At the core shed the core was measured for core loss and RQD determination. A reference line is marked on the core and one metre intervals are annotated onto the core. The core is then logged according to a set of pre-defined codes, in particular for lithology, alteration, veining and sulphide mineralogy. Figure 10-1 contains a flowsheet for the K92ML core handling procedure:

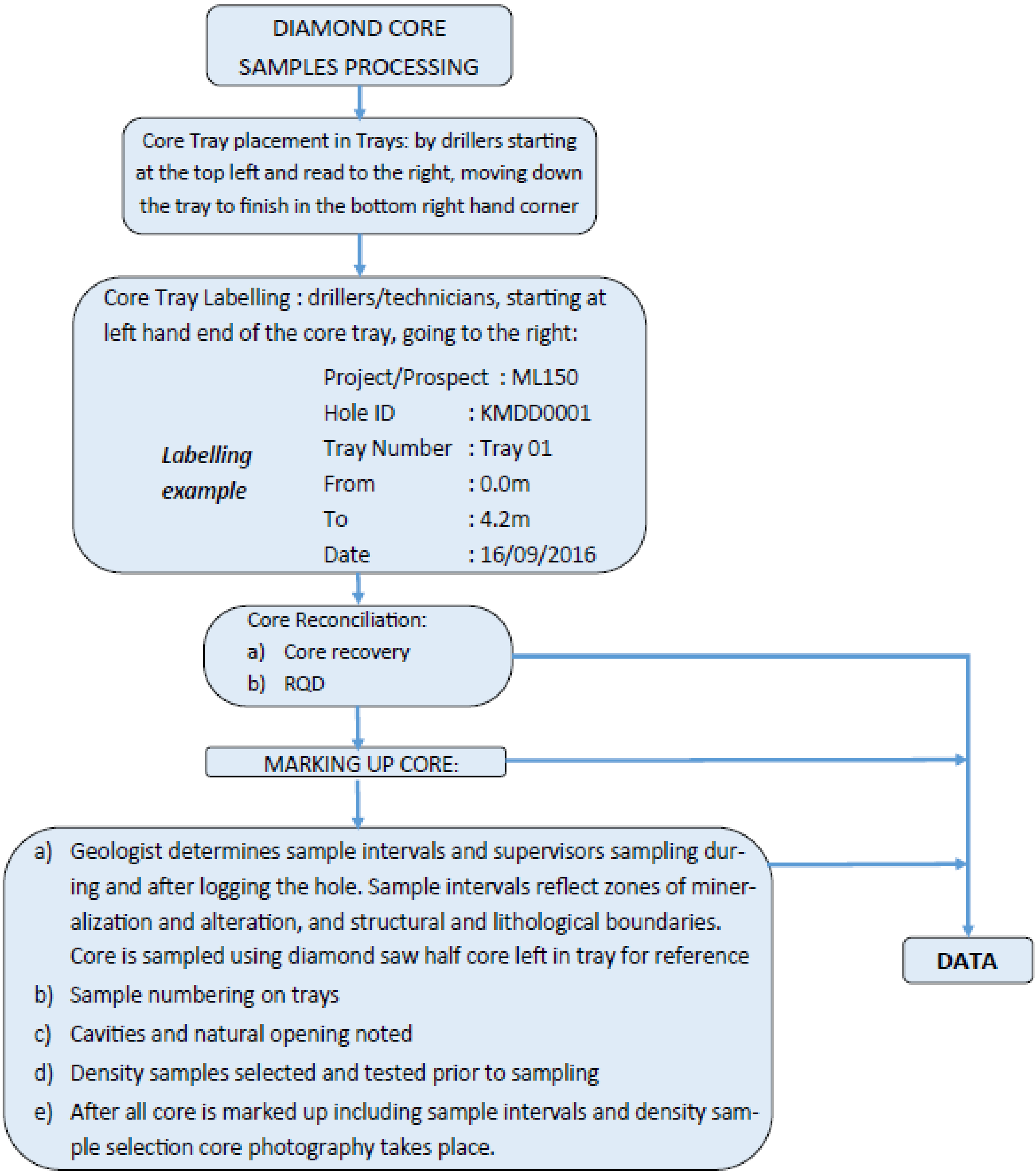


Figure 10-1 Flowsheet diagram of the diamond core handling process

Face sampling and geological mapping is done after each cut along the drive, with the cut interval along the drive being nominally 3.5-4 m using Jumbos or (historically) 1.5m using airlegs. The face mapping and channel samples are taken across the strike of the vein system under geological control, at right angles to the drive, with sample intervals ranging from 0.1m to 1m in length. (Example below in Figure 10-2).

The Barrick/Highlands drilling was reviewed and endorsed by Nolidan Mineral Consultants in 2015. Drillcore handling and sampling procedures used by Barrick are reported in Section 10.2.

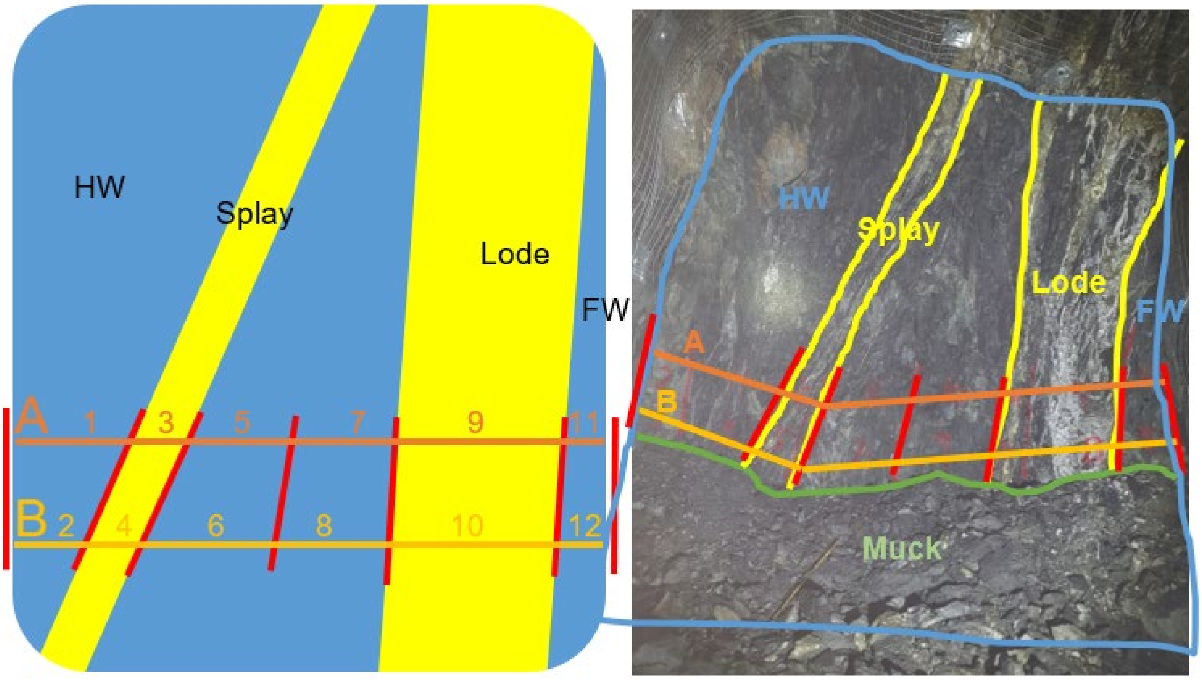


Figure 10-2 Face sample example with lode and the two sample channels and sample intervals

10.2 Core Recovery Analysis

K92ML, HPL and Barrick have followed the same recovery method analysis. The recovery data for each core run was recorded onto paper logs and entered into the MS Access drillhole database. Equal length weighted composites of 1m were generated for gold (g/t) and recovery (%) values for the K1, K2, Kora Link and Judd, J1, J1W, J2 and J3 lode structures. Selection of the lode intervals was based on the geological wireframes of each lode used in the resource estimation process, which were based on logged geology, logged vein tags (field 'vein_id'), and metal assay grades and trends located in the assay table of the drillhole database. This was done for each lode structure and enabled the comparison of gold grade versus the core recovery.

All K92ML drill recovery data has been used to construct the grade/recovery graphs. Comparisons of core recovery percentages between K92ML, HPL and Barrick are satisfactory.

10.2.1 K92ML K1 Lode

The K1 lode core recovery averaged 94% for 4,665 composited samples with no significant relationship between gold grade and core recovery (Figure 10-3). 3,650 samples have >85% core recovery.

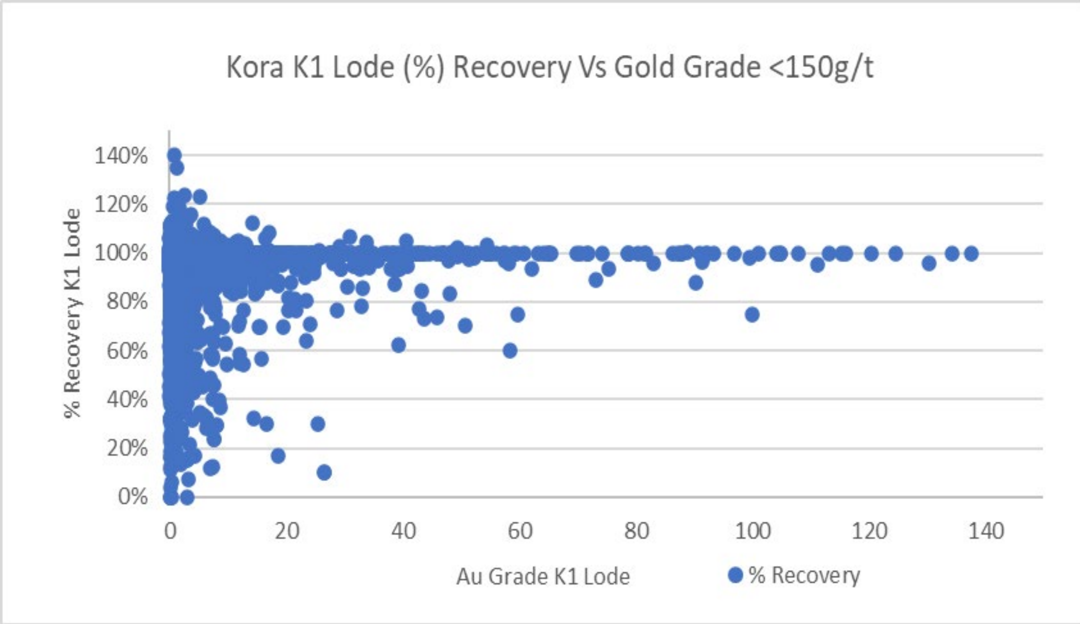


Figure 10-3 K1 gold grade versus core recovery

10.2.2 K92ML K2 Lode

The K2 lode core recovery averaged 91% for 5,039 composited samples with no significant relationship between gold grade and core recovery (Figure 10-4). 3,911 samples have >85% core recovery.

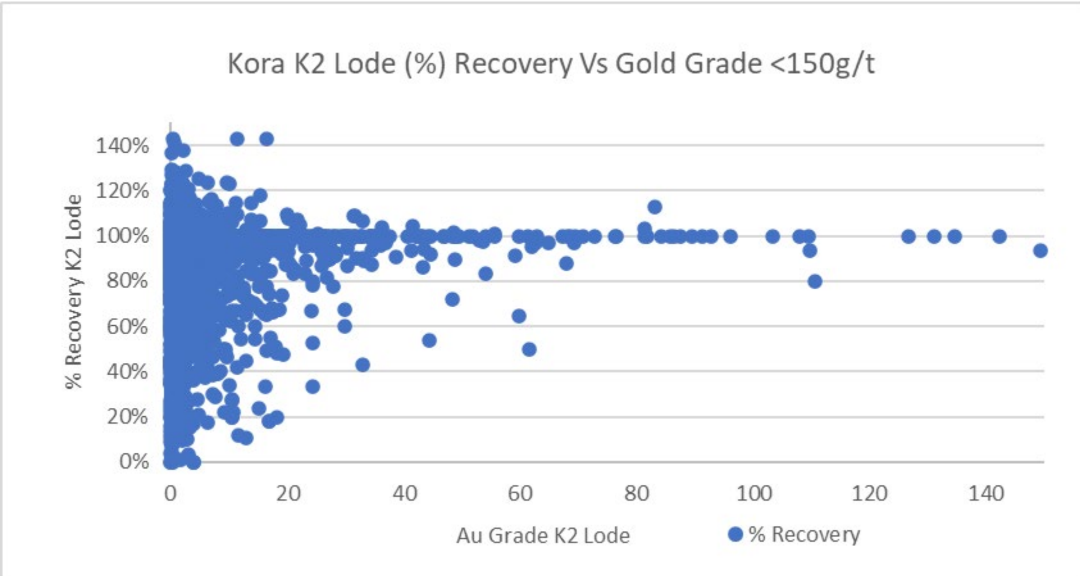


Figure 10-4 K2 gold grade versus core recovery

10.2.3 K92ML Kora Link

The Kora Link lode core recovery averaged 93% for 3,893 composited samples with no significant relationship between gold grade and core recovery (Figure 10-5). 3,190 samples have >85% core recovery.

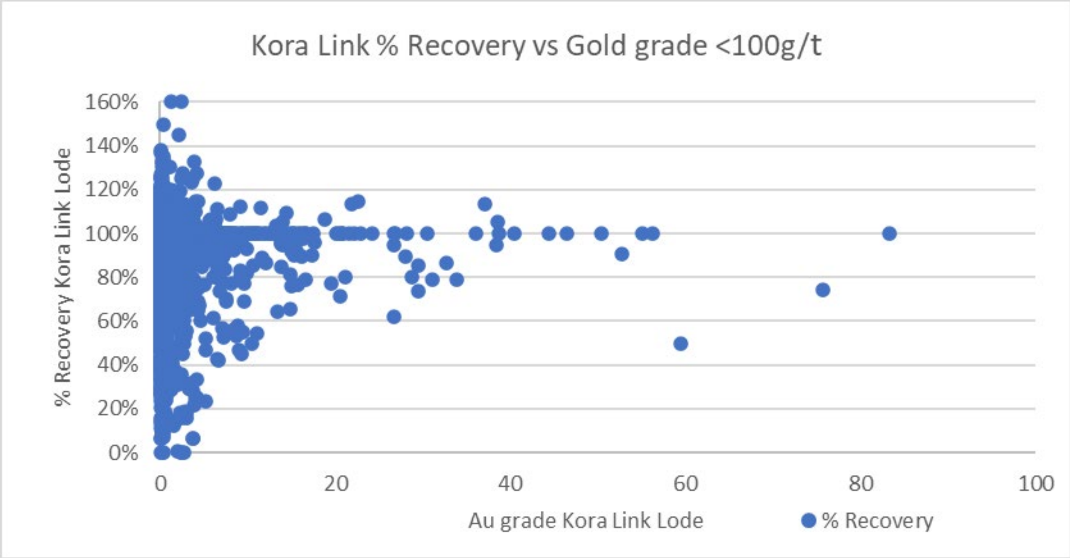


Figure 10-5 Kora Link gold grade versus core recovery

In summary, there is no relationship between core recovery and gold grade for the K1, K2 and Kora Link lode structures.

10.2.4 K92ML Judd J1 Lode

The Judd J1 lode core recovery averaged 96% for 1,788 composited samples with no significant relationship between gold grade and core recovery (Figure 10-6). 1,630 samples have >85% core recovery.

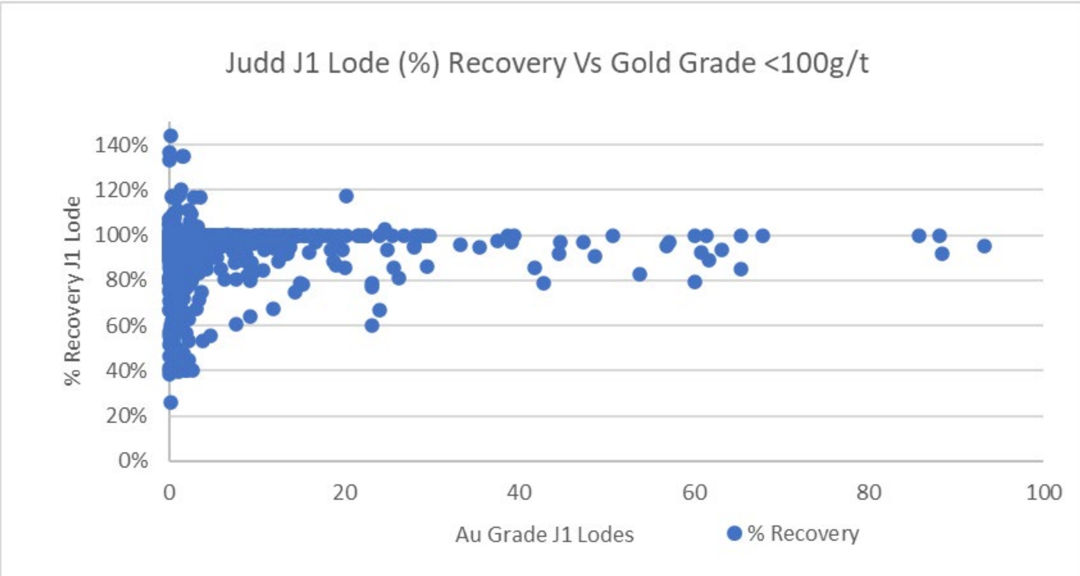


Figure 10-6 J1 Lode gold grade versus core recovery

10.2.5 K92ML Judd J1W Lode

The Judd J1W lode core recovery averaged 95% for 131 composited samples with no significant relationship between gold grade and core recovery (Figure 10-7). 118 samples have >85% core recovery.

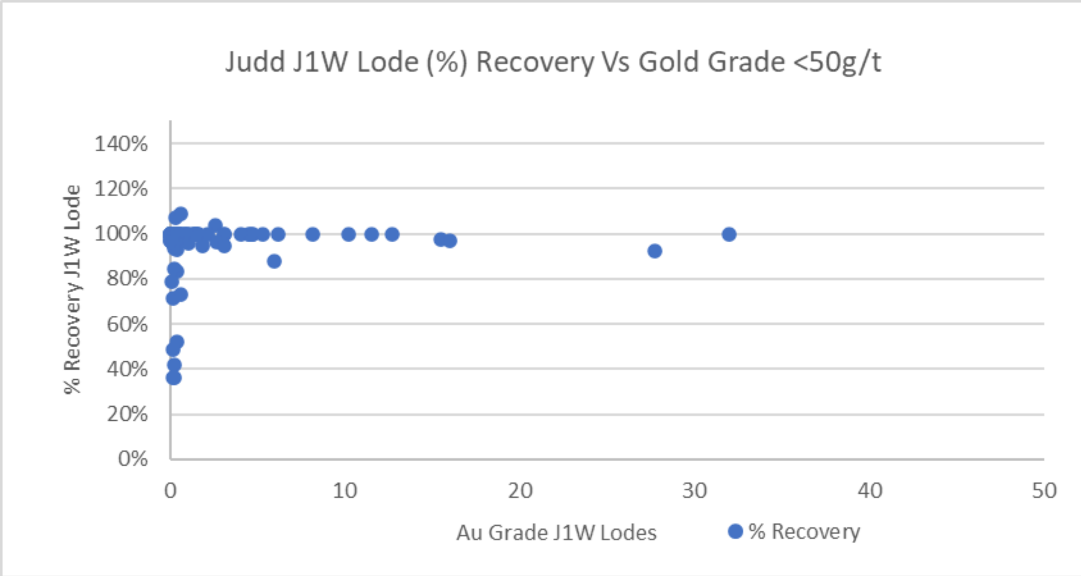


Figure 10-7 J1W Lode gold grade versus core recovery

10.2.6 K92ML Judd J2 Lode

The Judd J2 lode core recovery averaged 95% for 922 composited samples with no significant relationship between gold grade and core recovery (Figure 10-8). 815 samples have >85% core recovery.

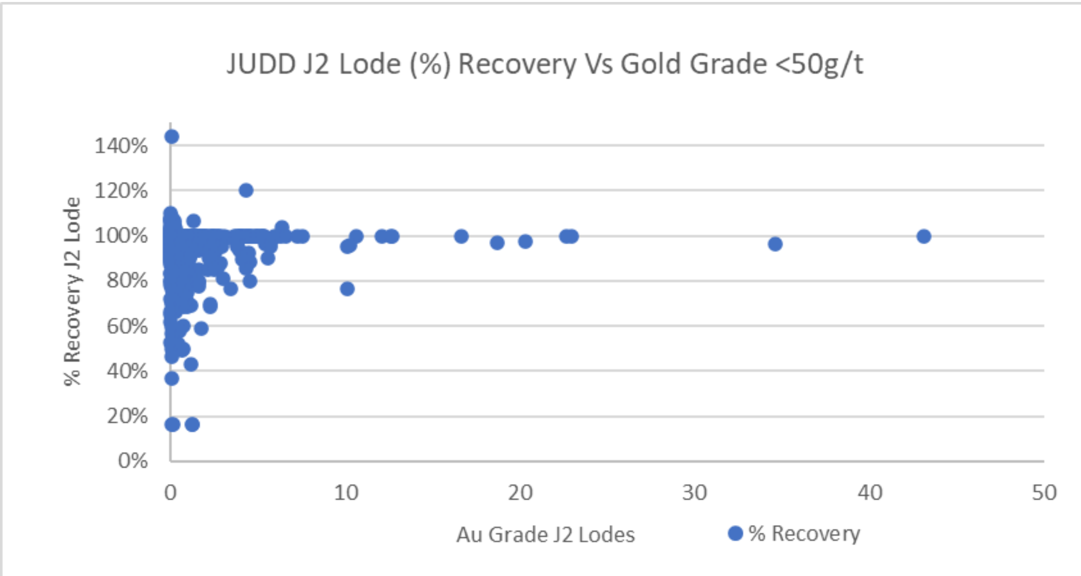


Figure 10-8 J2 Lode gold grade versus core recovery

10.2.7 K92ML Judd J3 Lode

The Judd J3 lode core recovery averaged 96% for 122 composited samples with no significant relationship between gold grade and core recovery (Figure 10-9). 110 samples have >85% core recovery.

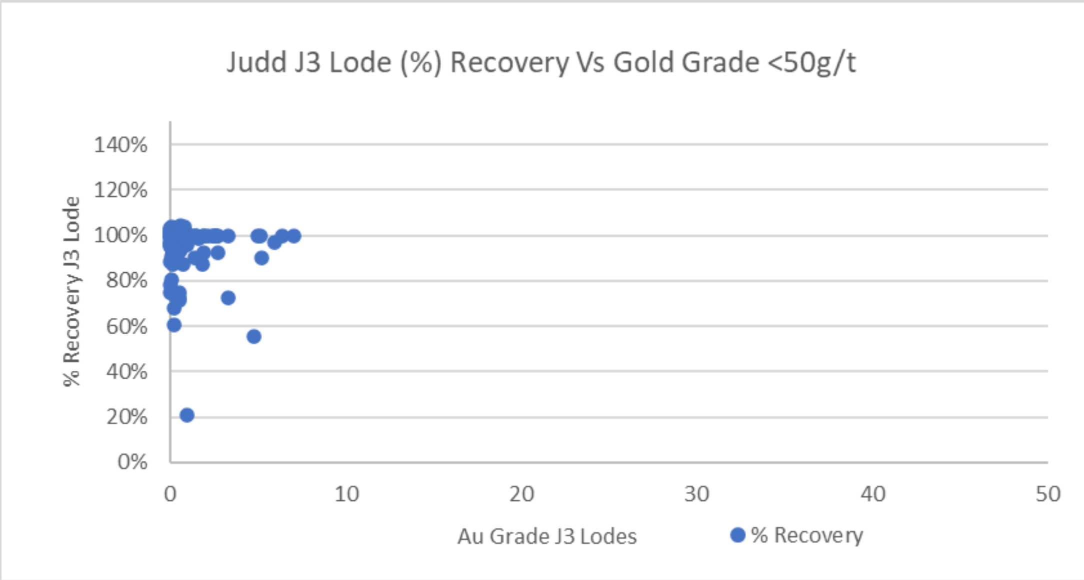


Figure 10-9 J3 Lode gold grade versus core recovery

In summary, there is no relationship between core recovery and gold grade for the J1, J1W, J2 and J3 lode structures.

Overall, the average recovery for the Kora lodes was, 92.6% and for the Judd Lodes was 95.5%.

10.2.8 Highlands Pacific

Analysis of the core recovery data for the HPL drilling indicated no significant relationship between gold grade and core recovery for the K1 lode (Figure 10-10) or K2 Lode (Figure 10-11).

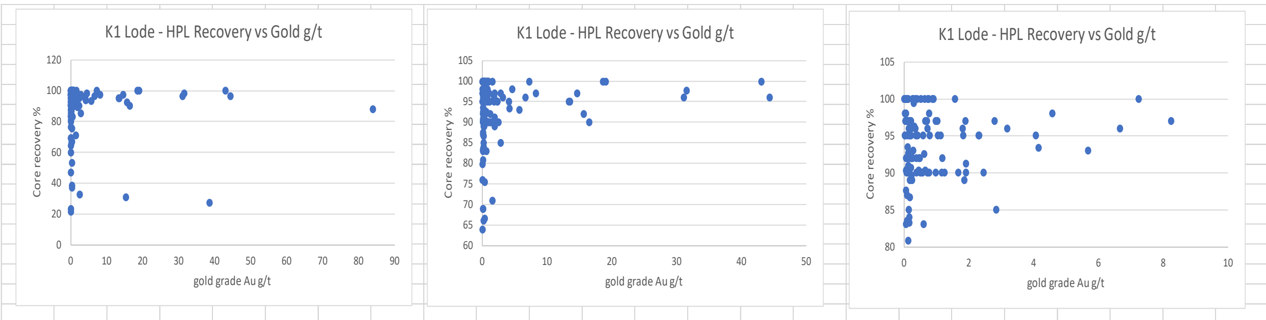


Figure 10-10 Highlands Pacific Gold Limited core recovery K1 lode

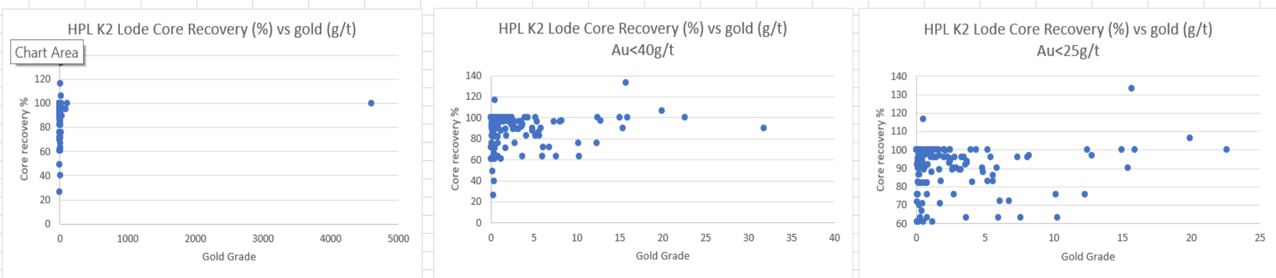


Figure 10-11 Highlands Pacific Gold Limited core recovery K2 lode

HPL's average core recovery for K1 was 89% and for K2 was 91%

10.2.9 Barrick Gold Limited

Analysis of the core recovery data for the Barrick drilling indicated no significant relationship between gold grade and core recovery for the K1 and K2 lodes (Figure 10-12).

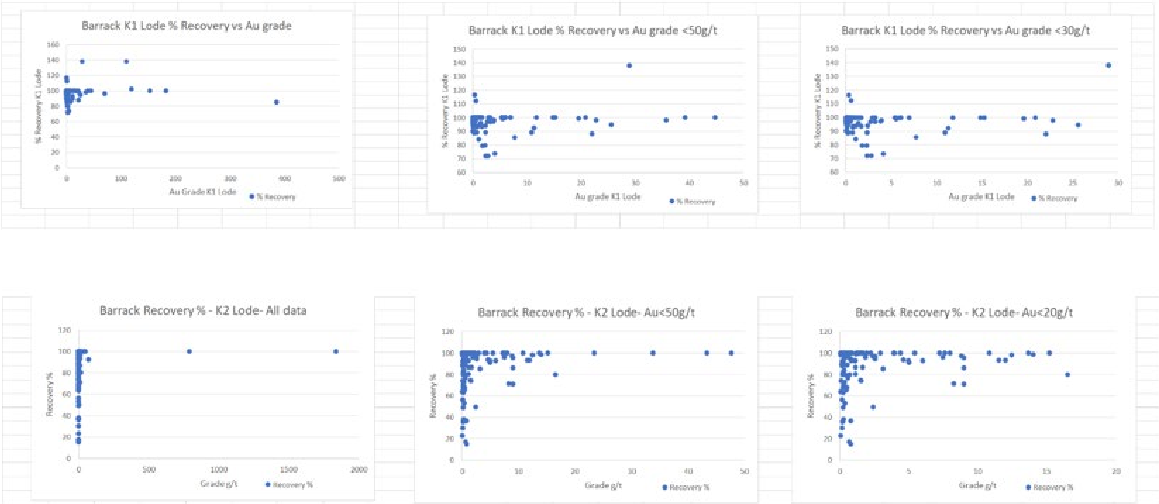


Figure 10-12 Barrick Gold Limited core recovery for K1 and K2 lodes

In summary there is no gold grade bias associated with core recovery, in particular there is no subset in the data of high gold grades associated with low recoveries.

Barrick's average core recovery for K1 was 97.8% and for K2 was 89.3%.

10.3 Documentation and Storage

The drill data is recorded into a series of MSExcel worksheets, including core recovery, RQDs, density measurements, geological logging and sample intervals. Data is logged directly into the LogChief software using tough book computers at the coreyard, once validated by the Database Geologist and Senior Geologist the data is then entered into the site DataShed database and then exported to an MSAccess database for subsequent drill planning, geological interpretations and resource estimation.

Core is stored in core trays on weatherproof pallets. Pallets are nominally loaded to 1.5 m high and contain 3 columns of core trays on each one, with an aluminium lid placed on each column for waterproofing. Core trays are strapped to the pallet with metal straps and tarpaulins are used to cover the pallets to prevent weathering. Pallets of core for a single hole are then taken to a lay down area and stored for future reference. Some core is also stored in sea-containers with this form of storage limited to the surface drill campaigns and older historical core from Barrick Gold's drilling.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Highlands Pacific

Highlands Pacific had documented drilling protocols and procedures for both surface and underground diamond drilling. In summary core was carefully reconstructed in the trays after transport with a centre reference line marked on the core for sampling purposes. Mineralized zones were sampled as sawn half core with 5 m either side of the mineralized zone also being sampled. Maximum sample interval was 1 m with a minimum of 0.5 m. Intervals were determined under geological control by a qualified geologist. The entire half-core sample was dried in a monitored oven between 95°C and 113°C. After 10 hours, the samples were removed from the oven and allowed to cool, then crushed to 2mm and reduced to minus 75 micron with an LM5 pulveriser before the 250 gram assay portion was split off via the Laboratory riffle splitter. All samples were analysed at Astrolabe's Madang Laboratory. Gold was determined by using a 50g fire assay with an AAS finish and other elements such as Cu, Pb, Zn and As were determined via a three acid digest and an AAS finish.

11.2 Barrick

The drill core handling procedures used by Barrick were reported by Nolidan Mineral Consultants in 2015. His findings are as follows: "All drill core was logged, photographed (wet and dry), then cut and sampled at Barrick's Kumian core yard. Logging data entry was completed using an in-house developed version of the AcQuire software. After logging, core was half-cut using diamond saws, and continuously sampled into numbered calico sample bags. The samples were submitted to the sample preparation facility of Intertek Laboratory Services in Lae (PNG). Sample preparation involved drying the samples at 105°C, crushing in a jaw crusher with 95% of the sample passing <2 mm, riffle splitting and pulverising to 95%passing <75 µm."

11.3 K92ML

11.3.1 Sampling

A sampling flow diagram for both UG and surface drilling is presented in Figure 11-1.

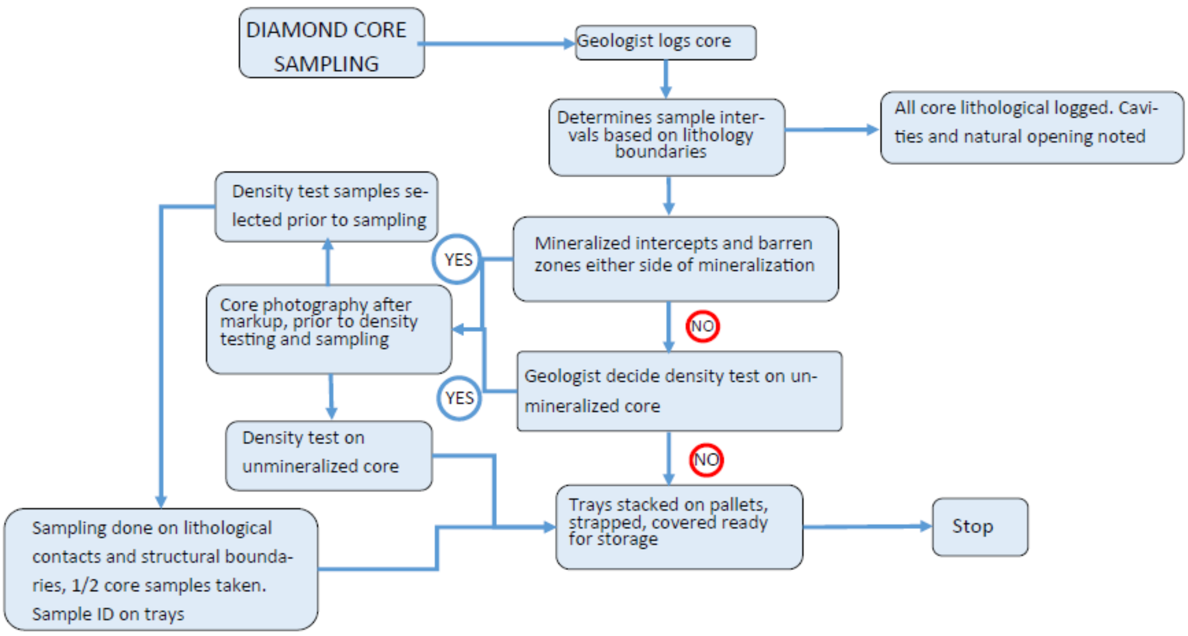


Figure 11-1 Diamond Core Sampling Process Flow Diagram

The core tray labelling and core placement in the tray is done by the drilling and field assistants' team at the drill site, the core is removed to the core yard where the geologist logs the core, determining sampling intervals according to the geology intersected. Once sample intervals are determined they are marked on the core by the geologist along with the sample number using a white paint marker or red chinagraph pencil. A minimum sampling width of 0.1 m and a maximum of 1 m were used in mineralization and occasionally a 1.5 m sample interval was used in unmineralized core. The smaller sample intervals were utilised to sample individual sub-veins/stringers and sulphide intercepts. Core was sampled as a minimum to greater than 10m either side of each mineral lode, including stringer-style mineralization away from the lodes. Unmineralized areas at the start of the hole, at the end of the hole and between K1 and J1 lodes were usually left unsampled. All mineralized occurrences were sampled. Core sampling was almost always continuous between K1 and K2 where Kora Link resides, however, in the north end of the Kora deposit, Kora Link is narrow in width and in these areas the barren core between K1 and K2 was not sampled.

Sampling of the core involved sawn half core cut along the reference line. For LTK60, NQ, NQ2, HQ and PQ core the left hand (looking down hole) half core is sampled. The remaining half of the core is returned to the core tray. Core samples were placed in numbered calico and plastic bags and a numbered sample ticket placed in each calico bag for dispatch to the on-site assay laboratory. This laboratory is managed by Intertek Testing Services (PNG) LTD, an independent accredited laboratory.

K92ML has a standard underground face sampling procedure in place. Face samples are taken across the full face of both the exposed lode system and any waste rock, under geological control. Sample intervals range from 0.1 m to 1 m in width depending on the geology. Two samples are taken per interval at waist and knee height and the corresponding widths recorded. Samples are approximately 3.5 kg in weight. Widths and dimensions of the mineralization were documented on a face sketch with the location of the face sketches determined by the geologist and surveyor using the surveyed stations along the drive. The two samples for each interval are assayed separately for Au, Cu and Ag and the results averaged out using length weighting and channel orientation before entry into the database.

K92ML has a documented QAQC procedure that included the insertion standards, blanks and duplicates into the sample suite for each hole (for all drilling since 2017) and for the face sampling (since 2018). HPL's drilling QAQC results were satisfactory. However, Barrick's QAQC results could not be found.

A twin drill hole programme, composed of four holes drilled from surface was completed by K92ML in the 2020-2021 period, to measure for any sampling or assay bias with the Barrick surface holes. The outcomes confirmed that there was no significant bias between Barrick assays compared to those derived from K92ML drill holes.

11.3.2 Sample Preparation

Intertek Laboratory Services based in Lae, Morobe Province, PNG provides K92ML with on-site laboratory facilities. The sample preparation for both the drill core and face samples is described below and in a flow diagram presented in Figure 11-2.

- Samples are sorted and dried at 105°C overnight.
- Jaw crushed to 5 mm.
- Secondary jaw crush using a Boyd crusher to 2 mm and then rotary split to give 1 kg.
- Pulverisation using an LM2 mill of the 1 kg sample to 90% passing 106 microns.
- Duplicate splits 1 in 30, using a micro riffle splitter for splitting pulp samples.

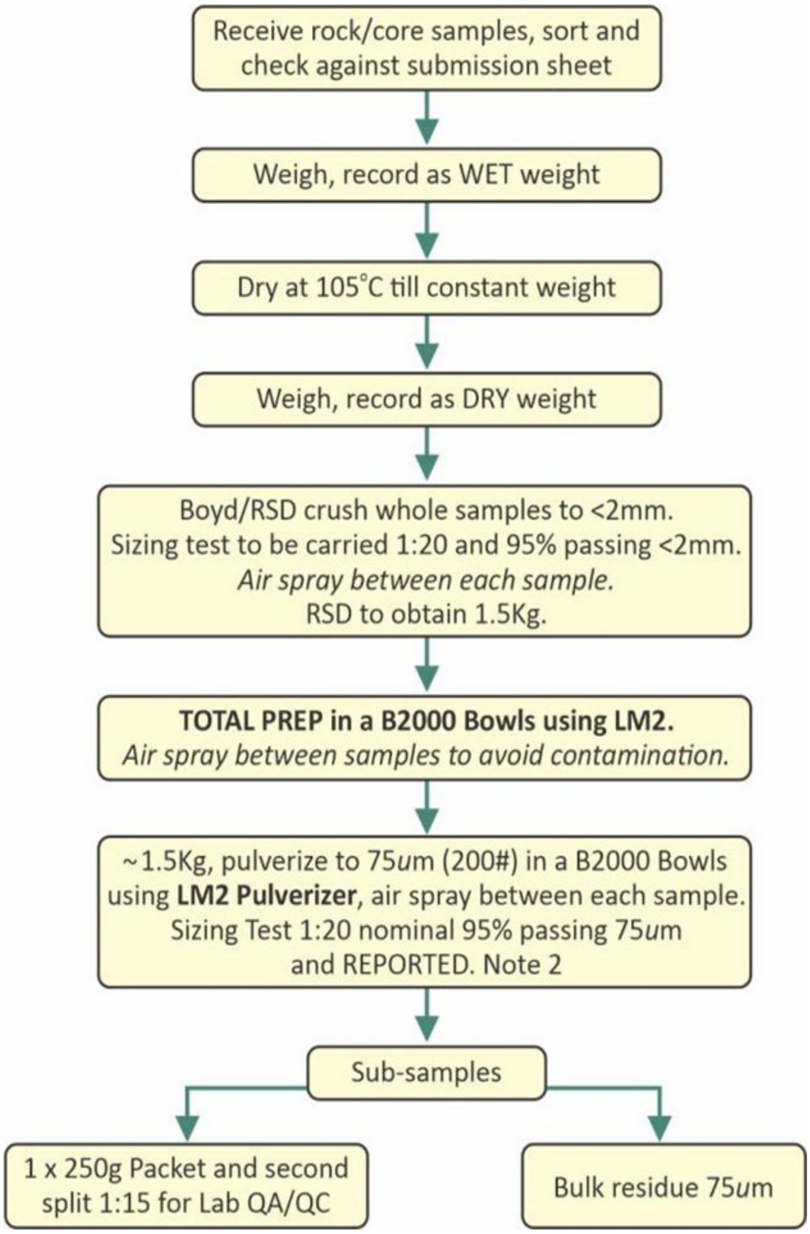


Figure 11-2 Sample Preparation Procedure

11.3.3 Density Measurements

Historic Density Measurements

Previous density testwork is summarized in the Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment of Irumafimpa and Kora Gold Deposits, Kainantu Project, Papua New Guinea, 02 March 2017 (Woodward, A.J, Desoe C., and Park L.J., 2017).

During the initial 2002 Feasibility Study HPL carried out density determinations using water immersion method (density =weight in air overweight in air minus weight in water), on 35 samples sourced predominantly from the Irumafimpa exploration adit. Density of these samples ranged from 2.9 t/m^3 to 3.7 t/m^3 with HPL using a default density of 2.9 t/m^3. This incorporated a correction for voids which constitute approximately 10% of the total volume of the Irumafimpa lodes (SRK, 2006). Historic resource estimates by Hackchester (2005) and Mining Associates (2006) used an average density value of 2.9 t/m^3.

Barrick made 428 density measurements of drill core from Kora. These were mostly waste material but included 5 intersections of the interpreted Mill and Robinson veins of Irumafimpa, located to the north of Kora, which has not been mined since 2017. Densities were determined using the immersion in water method with the Archimedes Principle used to determine density, via the formula, density =dry weight of the sample / weight in air minus weight of the sample in water, (Bond, R., Dobe, J., & Fallon, M., 2009). Average values from measurements for the lode material ranged from 2.58 t/m^3 to 2.77 t/m^3.

In the 2015 MRE vein blocks in the Irumafimpa deposit were assigned a density of 2.9 t/m^3 and vein blocks in the Kora deposit were assigned a density of 2.8 t/m^3.

In 2018 density testing of core from diamond drilling at Kora North consisted of 154 samples concentrated on the mineralized zones. This used the geologists' discretion to take representative core samples of the lodes and were usually between 3-10 cm in length. Samples were taken across the K1 and K2 lodes. This work indicated an average density of 2.8 t/m^3 however, samples tested were spatially limited in their extent.

Density testing between October 2018 and March 2020 were summarised for K1 and K2 lodes in Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment for expansion of the Kainantu mine to treat 1M tpa from the Kora Gold Deposit, Kainantu Project, Papua New Guinea, dated 27 July 2020. The work indicated the average density value for K1 Lode is 2.84 t/m^3 and for K2 is 2.93 t/m^3.

For the previous 2022 MRE K92ML provided some analysis of the density data which resulted in average density values for the four lodes and waste rock. Part of this analysis involved the exclusion by K92ML of low density values generally <2.3 t/m^3. This amounted to the removal of approximately 6.5% of the K1 density sample data, 10% for K2, 15% for the Kora Link and 11.5% for Judd. The impact of removing the low values had been a 1.5 to 3.5% increase in the average density value and is not considered significant, although it may not be considered as best practice especially when the values appear to be part of a single population. The default values used in the previous 2022 MRE were, for K1 and K2 being 2.84 t/m^3, for the Kora Link 2.74 t/m^3 and for J1 2.71 t/m^3.

11.3.4 Current Density Measurements

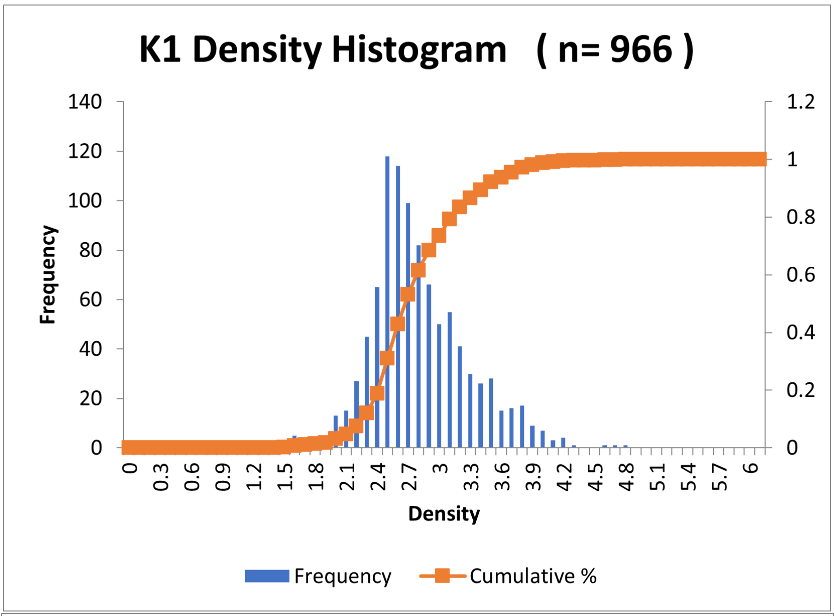
The K92ML density determination method for the Kora and Judd drill core samples has been consistent since 2018. Density test samples are selected by the geologist during the logging process. Sticks of core are taken out of the core tray, preferencing solid pieces of core of 10-25 cm length, they are weighed before drying in the oven at 105 degrees Celsius for 12 hours. The samples are then weighed dry, then weighed while submerged in water, then weighed after the submersion in water (The Archimedes Method), and moisture content and density calculated. Trials using cling film to wrap the core prior to submersion in water were abandoned as this tended to add buoyancy to the sample giving it less weight in water. Typically, moisture content of samples was less than 1% by weight.

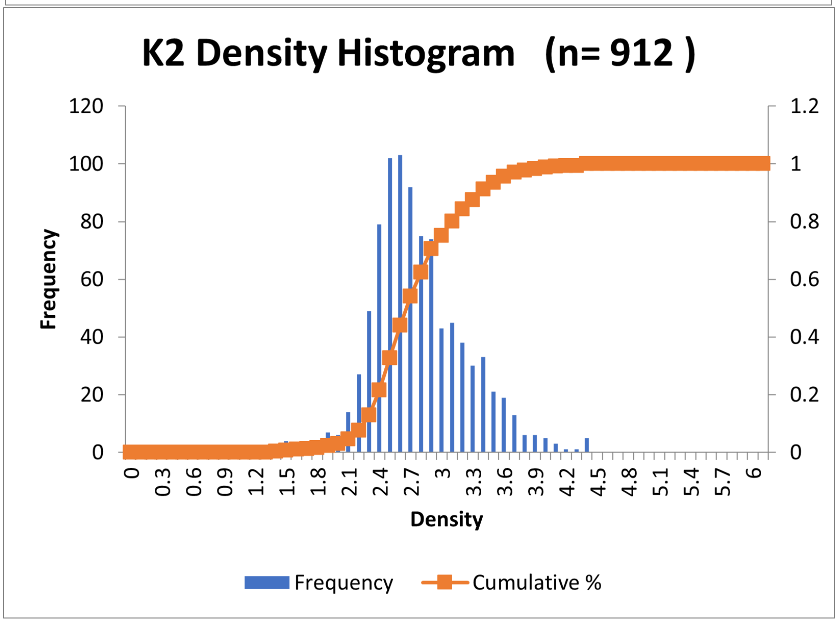
There is a risk to this density measuring method in that often there can be a bias towards selecting more competent pieces of core, and vuggy core or areas of poor recovery are not always represented. K92ML has also measured density on 1 m lengths of cut core and sometimes for whole core trays, the latter of which is particularly useful for broken core / oxidised / poor recovery zones. For the 2023 MRE a total of 6,608 density measurements were available via the supplied MSAccess database. This is a significant increase on previous MRE numbers such that it will allow for modelling of the density data for each lode rather than simply ascribing a default density for each lode. The sample data was reviewed and obvious errors in density measurements were removed. Table 11-1 contains detail on the number of samples for each lode with the obvious too high or too low values removed. There are still one or two low values but numerically these are insignificant with virtually no impact on the overall or even local density.

Table 11-1 Summary Statistics for Lode Density

Density	K1	K2	KLK	J1	J2	J3	J1W
Mean	2.759	2.735	2.634	2.623	2.542	2.755	2.214
Median	2.670	2.660	2.520	2.580	2.465	2.695	2.685
Standard Deviation	0.471	0.462	0.409	0.460	0.346	0.385	0.681
Sample Variance	0.222	0.214	0.167	0.211	0.120	0.148	0.463
Coeff of Variation	0.171	0.169	0.155	0.175	0.136	0.140	0.307
Minimum	1.385	1.343	1.860	1.205	1.744	2.240	1.439
Maximum	4.710	4.396	4.800	4.390	3.890	3.400	2.980
Count	966	912	381	519	160	24	9

Histogram plots of the density sample data for the K1, K2 and J1 lodes are very similar i.e. a normal distribution with a positive skew Figure 11-3. This would seem to suggest that the low values are part of a normal distribution, although geological intuition might suggest that the low values were too low and were in fact in error.





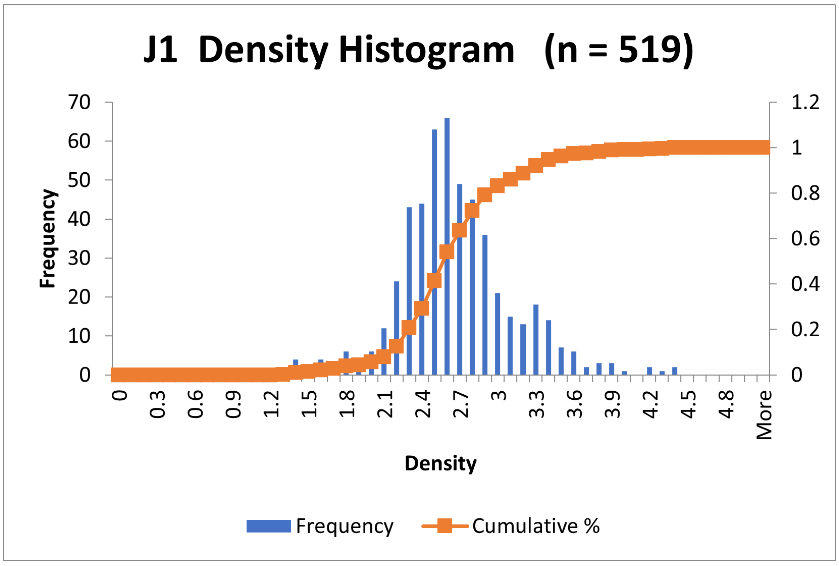


Figure 11-3 Histogram Plots of Density Sample Values for the K1, K2 and J1 Lodes

Figure 11-4 is a long section plot of the density values for K1. It shows a cluster of data associated with general mine area and localised patches of density in the peripheral zones.

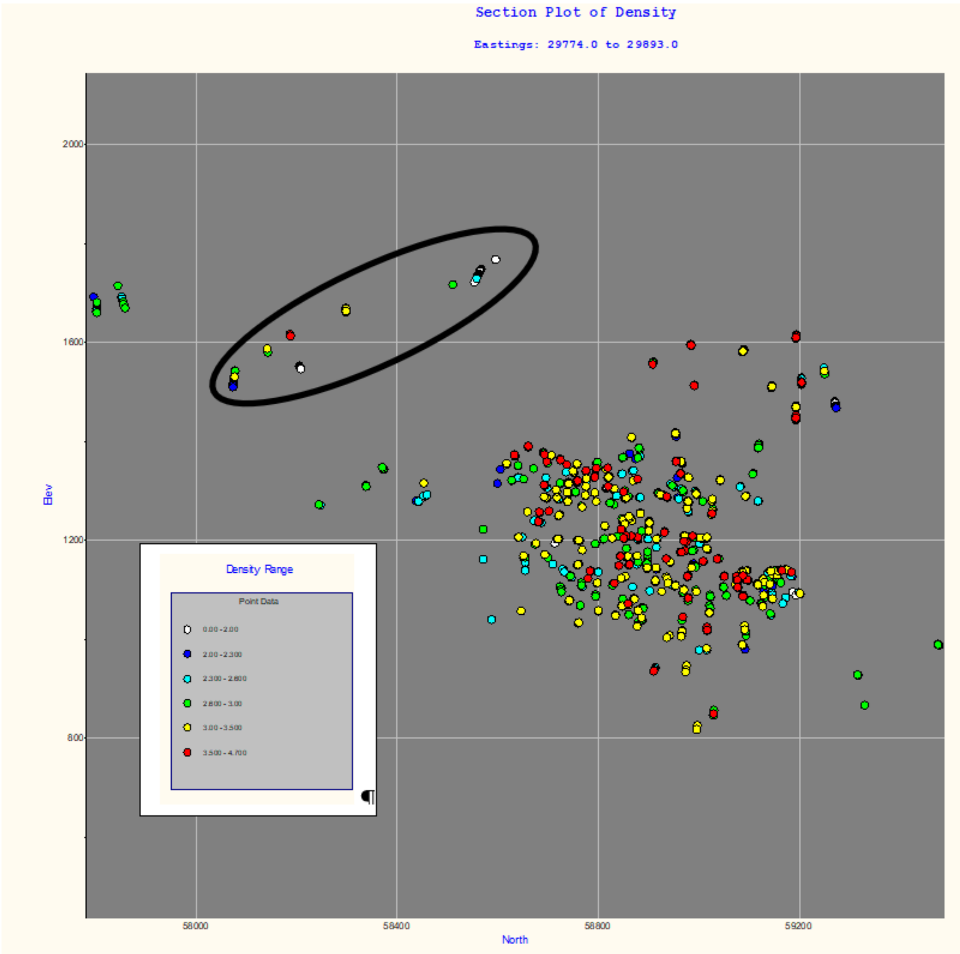


Figure 11-4 Density Sample Distribution for K1 Long Section

The plots for K2 and J1 lodes indicate similar patterns to K1 i.e. a cluster of low grade values in the top left corner of the long section and the occasional low value in the general mine area.

Following the completion of the resource estimation and with some further density data analysis by H&SC it transpired that there were a significant number of relatively low values for what seemed to be fresh rock, too many to be simple human error. The black ellipse in Figure 11-3 shows the location of the suspect values in amongst acceptable values for the K1 lode; K2 had a similar pattern for the density. The suspiciously low-density values are coincident with higher 'more normal' density values and were from certain continuous sections in surface exploration holes. Whilst it is not easily visible there are also low density values in the general mine area which might suggest that the low values elsewhere are reasonable. Having a large number of suspect data makes it difficult to decide which data to omit and thus H&SC decided to retain all data and model accordingly.

Subsequent discussion with K92ML personnel during the grade interpolation process suggested that these low values were in error. The main area of suspect data is the upper southern end of the deposit and so to counteract the possible erroneous density values a default density value of 2.73 t/m^3 was ascribed to blocks of the various lodes for a defined upper zone, replacing the modelled value.

Analysis of the density sample data for K1 lode by progressively removing low values provided a measure of the impact on the average density value. The results are in Table 11-2 which shows that removal of values <2.4 t/m^3 (K92's preferred cut off point from 2021) resulted in a 5% increase in the average density value, however this meant that about 18% of the density data would be classed as in error. This is a significant number and suggests a fundamental problem with the density measuring system. H&SC reviewed the density measuring set up for the mine exploration which appeared rudimentary but satisfactory but did not review the Exploration team's methodology.

Table 11-2 Density Summary Statistics for Different Low Value Cut offs for K1

Density K1	Global	<2.0 t/m^3	<2.2 t/m^3	<2.4 t/m^3
Mean	2.759	2.790	2.820	2.886
Median	2.670	2.690	2.710	2.770
Standard Deviation	0.471	0.443	0.428	0.410
Sample Variance	0.222	0.196	0.183	0.168
Coeff of Variation	0.171	0.159	0.152	0.142
Minimum	1.385	2	2.2	2.4
Maximum	4.71	4.71	4.71	4.71
Count	966	937	897	794

Chapter 14 contains further detail on the density model.

It is planned to carry out more density tests and to include a second check method of testing density using the 'cylinder water displacement' method and, to carry out routine weighing of a bulk sample of known volume, freshly excavated from an ore drive, to give a bulk density result.

11.3.5 Carbon Sulphur

- Read by combustion furnace.

11.4 Sample Analysis

The analytical methods are detailed as follows:

11.4.1 Gold

- Fire Assay Method with a 30 g charge (FA30).
- Samples are fired with a modified fire assay flux, prills digested at 100°C with aqua regia and read on an atomic absorption spectrometer (AAS).

11.4.2 Copper/Silver

- 3 acid digest at 180°C (Nitric, Perchloric & Hydrochloric mix).
- Diluted with water and read by AAS.

11.4.3 Fluorine

- Following a carbonate fusion, samples are read on a specific ion meter referenced against standards.

11.5 QAQC Programme and Results

Since its acquisition of the project in 2016, K92ML has implemented a QAQC programme. K92ML's documented QAQC programme for the drilling comprises standards, blank standards, laboratory duplicates, 2nd laboratory checks and twin holes. The face sampling prior to September 2018 had no QAQC samples inserted although the sample suites were inserted in between drillhole sampling suites which did have QAQC samples. From September 2018 to 2020 the same QAQC protocols were applied to the face sampling, however, in subsequent periods because the face sampling QAQC showed similar trends to the core sample QAQC results, it was decided by K92ML to desist from using QAQC samples for the face sampling and rely on the core sample QAQC.

A definition of QAQC terms used in this report is supplied in Table 11-3.

Table 11-3 List of QAQC Terms

Item	Description
Standard	A Certified Reference Material (CRM) used to measure accuracy of the sample prep and analysis of the samples.
Blank Standard	A sample with nil or negligible (i.e. undetectable) concentration of the tested element(s). Blank standards are used to monitor for contamination at the various stages of processing and assaying.
Field Duplicate	A second drillhole sample collected in the field. This sample provides a measure of the homogeneity of the sampled material, and short scale grade continuity
Laboratory Duplicate	This sample provides a check on sample homogeneity from the sample prep stage and the repeatability of the sample extraction. Sample collected as a sub-sample of the originally submitted 2-3 kg sample after crushing and pulverizing. Sometimes referred to as a pulp duplicate
2nd Laboratory Checks	Check assays on all shipments are yielded from samples submitted to more than one laboratory (organization) for validation of consistency. Sometimes known as umpire laboratory checks.
Laboratory Replicate	A second measurement (often analytical reading) of the same sub-sample after sample digest; a check for sample prep homogeneity and machine calibration.
Twin Hole	A repeat hole located in very close proximity to an original hole i.e. <5 m spatial difference. It is used to validate the primary drilling and a check that the sampling is representative and provides a measure of short scale grade continuity.

The K92ML 2018 QA/QC programmes comprised the use of standards including blank samples, laboratory duplicates and second laboratory check assays. The QAQC outcomes for the drilling were reported by H&SC in the relevant resource estimation reports. There were concerns over possible low-level contamination associated with the sample preparation. This was remediated with an improved dust extraction system in the sample preparation location in the following year.

The K92ML 2018-2020 QA/QC programmes comprised the use of standards including blank samples, laboratory duplicates and second laboratory check assays.

The K92ML 2020-2021 QA/QC programmes comprised the use of standards including blank samples, certified reference materials, laboratory duplicates and second laboratory check assays. The QAQC also included a diamond hole twinning programme of 4 pairs that was completed by K92ML in 2021 twinning previous historical surface drilling.

The K92ML 2021-2023 QA/QC programmes comprised the use of standards including blank samples, laboratory duplicates and second laboratory check assays.

Standard insertions and check sample selections for the K92ML drilling, 2016-2023 drilling are listed below:

All QAQC samples have an insertion rate that is done on a batch process, batches are made up of samples from a single hole and are composed of up to 200 samples in a single batch. The QAQC insertion rates are broken down as listed below:

- Blanks: one inserted after 20th original sample as the 21st sample.
- Standards: Gold, 1 after every 20 original samples as the 22nd sample.
- Standards: Base metal, 1 after every 10 original samples as the 11th sample.
- Duplicate as the 23rd sample.

11.5.1 Standard (Certified Reference Material, CRM Overview)

In the 2017 to 2018 period K92ML used two standards for gold only (G914-4, G915-8).

After the 2018 K92ML purchased a range of standards (Certified Reference Material) from Geostats Pty Ltd. The standards are certified for gold and in some instances, for copper and silver. Another set of standards were purchased from Gannet Holdings Pty Ltd for gold only. The standards comprise low, medium (head grade) and high-grade values and were submitted to the laboratory as part of the sample suite.

As professionally prepared standards could not be hidden amongst the core samples they were submitted routinely as the 22nd sample in the sample sequence to the onsite laboratory (Intertek Laboratories).

Details of the standards are in Table 11-4.

Table 11-4 Table of Certified Reference Materials (Standards)

CRM name	Certified for	Mean value	Lower bound	Upper bound	Supplier
G914-4	Gold (g/t)	0.2	0.18	0.22	Geostats PTY LTD
G915-8	Gold (g/t)	24.66	22.19	27.13	Geostats PTY LTD
G312-5	Gold (g/t)	1.60	1.44	1.76	Geostats PTY LTD
G915-2	Gold (g/t)	4.98	4.48	5.48	Geostats PTY LTD
G916-6	Gold (g/t)	30.94	27.85	34.03	Geostats PTY LTD
G314-2	Gold (g/t)	1.58	1.42	1.76	Geostats PTY LTD
G904-7	Gold (g/t)	1.6	1.44	1.76	Geostats PTY LTD
G910-6	Gold (g/t)	3.09	2.78	3.40	Geostats PTY LTD
G919-5	Gold (g/t)	11.3	10.17	12.43	Geostats PTY LTD
G919-10	Gold (g/t)	7.58	6.82	8.34	Geostats PTY LTD
G301-9	Gold (g/t)	5.67	5.1	6.24	Geostats PTY LTD
ST643	Gold (g/t)	4.94	4.45	5.43	Gannet Holdings PTY LTD
ST621	Gold (g/t)	33.24	29.92	36.56	Gannet Holdings PTY LTD
ST614	Gold (g/t)	1.00	0.90	1.10	Gannet Holdings PTY LTD
ST589	Gold (g/t)	2.42	2.18	2.66	Gannet Holdings PTY LTD
ST725	Gold (g/t)	12.38	11.14	13.62	Gannet Holdings PTY LTD
ST695	Gold (g/t)	33.50	30.15	36.85	Gannet Holdings PTY LTD

CRM name	Certified for	Mean value	Lower bound	Upper bound	Supplier
ST732	Gold (g/t)	4.98	4.482	5.478	Gannet Holdings PTY LTD
ST720	Gold (g/t)	0.30	0.27	0.33	Gannet Holdings PTY LTD
GBM309-4	Copper (%)	2.2334	2.0101	2.4567	Geostats PTY LTD
	Silver (g/t)	42.3	38.1	46.5	Geostats PTY LTD
GBM915-16	Copper (%)	2.296	2.066	2.526	Geostats PTY LTD
	Silver (g/t)	51.2	46	56.5	Geostats PTY LTD
GBM910-4	Copper (%)	0.5412	0.48708	0.59532	Geostats PTY LTD
	Silver (g/t)	1.8	1.62	1.98	Geostats PTY LTD
GBM303-6	Copper (%)	1.3967	1.25703	1.53637	Geostats PTY LTD
	Silver (g/t)	5.5	4.95	6.05	Geostats PTY LTD
GBM910-6	Copper (%)	1.0084	0.90756	1.10924	Geostats PTY LTD
	Silver (g/t)	3.6	3.24	3.96	Geostats PTY LTD
GBM315-10	Copper (%)	0.2646	0.23814	0.29106	Geostats PTY LTD
	Silver (g/t)	4.7	4.23	5.17	Geostats PTY LTD
GBM910-13	Copper (%)	0.2306	0.20754	0.25366	Geostats PTY LTD
	Silver (g/t)	2.0	1.8	2.2	Geostats PTY LTD
GBM920-10	Copper (%)	2.1707	1.95363	2.38777	Geostats PTY LTD
	Silver (g/t)	41.9	37.7	46.1	Geostats PTY LTD
GBMS911-3	Gold (g/t)	1.33	1.20	1.46	Geostats PTY LTD
	Copper (%)	0.77	0.73	0.80	
	Silver (g/t)	1.7	1.53	1.87	
GBMS304-4	Gold (g/t)	5.67	5.10	6.24	Geostats PTY LTD
	Copper (%)	0.98	0.93	1.03	
	Silver (g/t)	3.4	3.06	3.74	
Blank	Gold (g/t)	0.1	0	0.2	Sourced locally from barren sediment
	Copper (%)	0.01	0.00	0.02	
	Silver	1	<1	2	

11.5.2 Gold Standards

Examples of the gold assay results for the standards used by K92ML are included in this section. The Blue Lake QAQC data are included on the various graphs for standards, blanks and duplicates, because the Blue Lake samples were also submitted to the on-site Laboratory and form part of the data set.

Figure 11-5 shows over the time period 2018 to 2023, the results for the low-grade standard. The central horizontal line on the graph is the certified value and the upper and lower horizontal lines are the ±10% variance of the certified value and are placed on the graph to give an indication of the accuracy of the reported laboratory results and to show if there are any biases in the data. The X axis represents time, with the higher numbers being more relatively recent.

The results for the low-grade standard are reasonable considering the low value for the standard with no significant bias. There is some lack of precision in the data as shown by the spread of data above and below the ±10% lines.

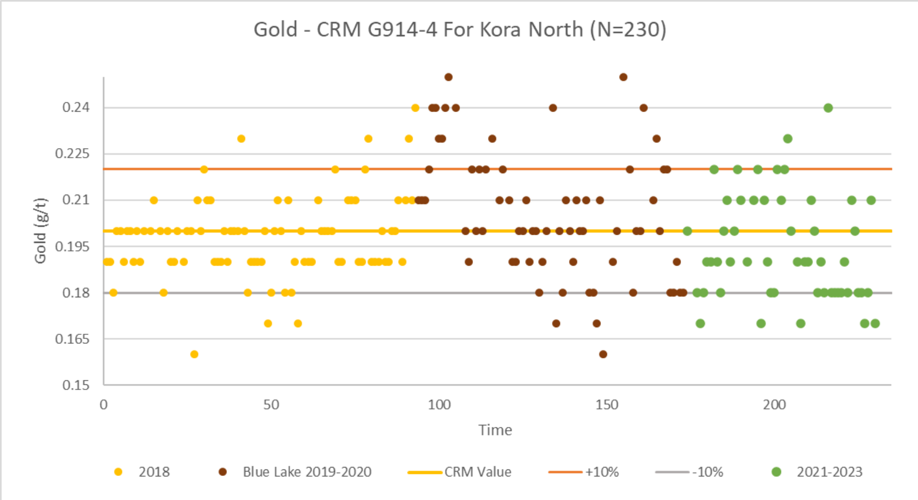


Figure 11-5 Low Grade Standard (G914-4)

Figure 11-6 shows for 2018 the results for the high-grade standard G915-8.

The results for the high-grade standard shown below, are satisfactory, with no real significant bias. Precision is quite reasonable.

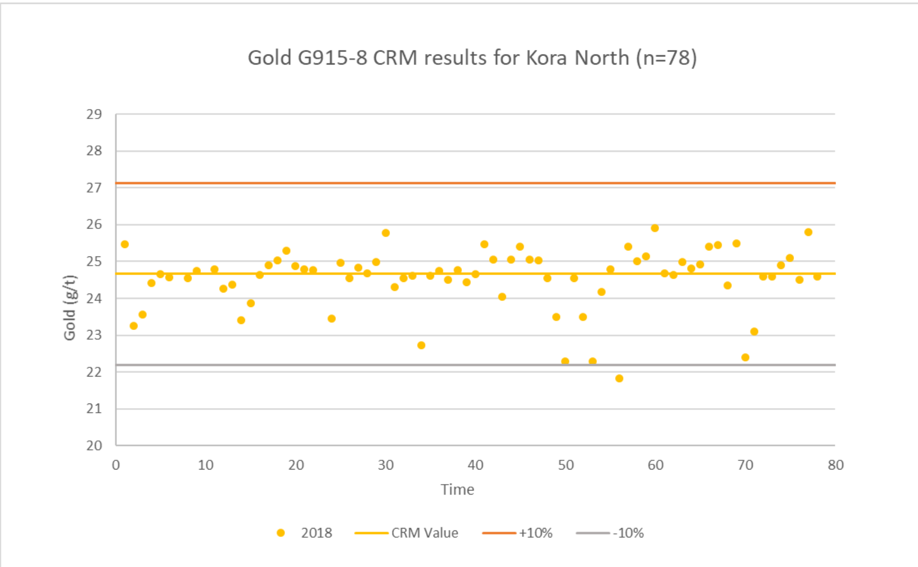


Figure 11-6 High Grade Standard (G915-8)

Figure 11-7 shows the assay values for low grade standard ST710 over the time period 2021 to 2023. There appears to be no significant bias with reasonable precision.

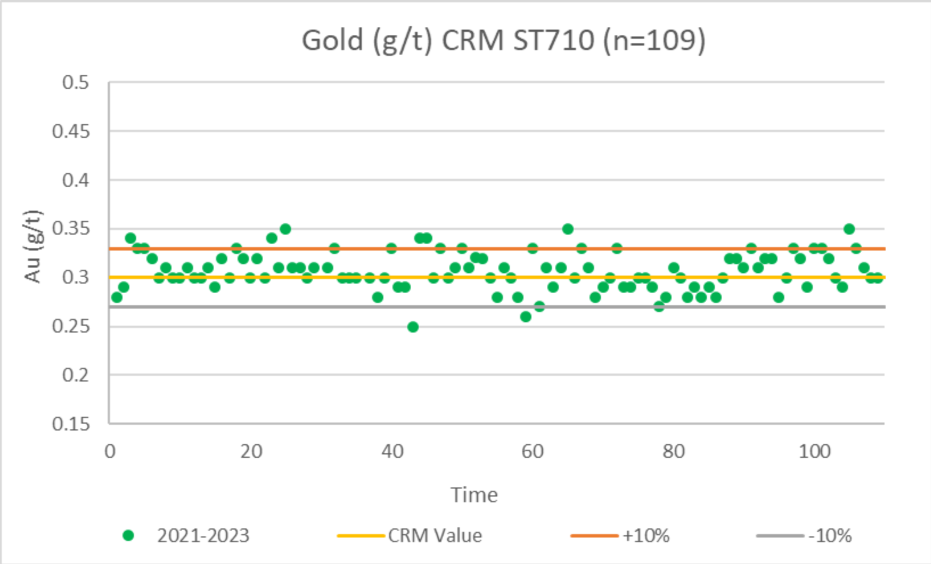


Figure 11-7 Low Grade Standard (ST710)

Figure 11-8 shows the results for the low grade standard G312-5 for all deposits over the time period 2019 to 2023. The results show no significant bias with reasonable precision. Judd in the legend on the graph refers to Judd in the 2020-2021 MRE period and in the 2021-2023 MRE period Judd has been combined with Kora. This was done to simplify the graphs as the Judd QAQC results were no different to Kora.

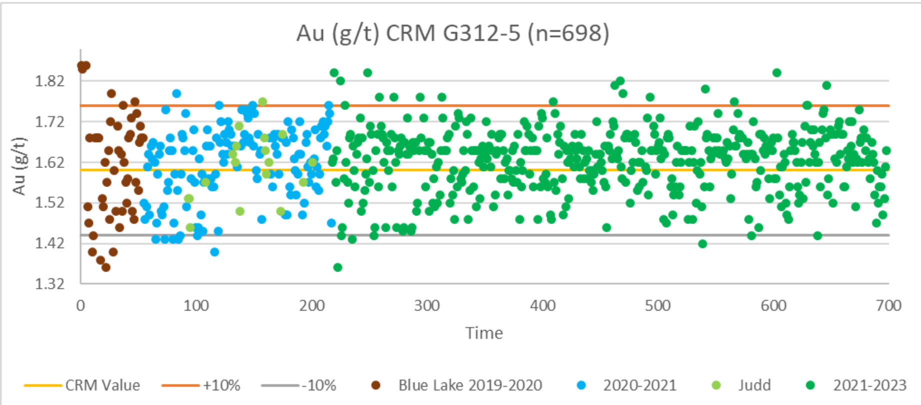


Figure 11-8 Low Grade Standard (G312-5)

Figure 11-9 shows the results for the low-grade gold standard, ST614 and indicates a slight positive bias with some cyclicity i.e. indicating over-reporting of assay results. However, the bias is within acceptable limits of variation and the precision is good.

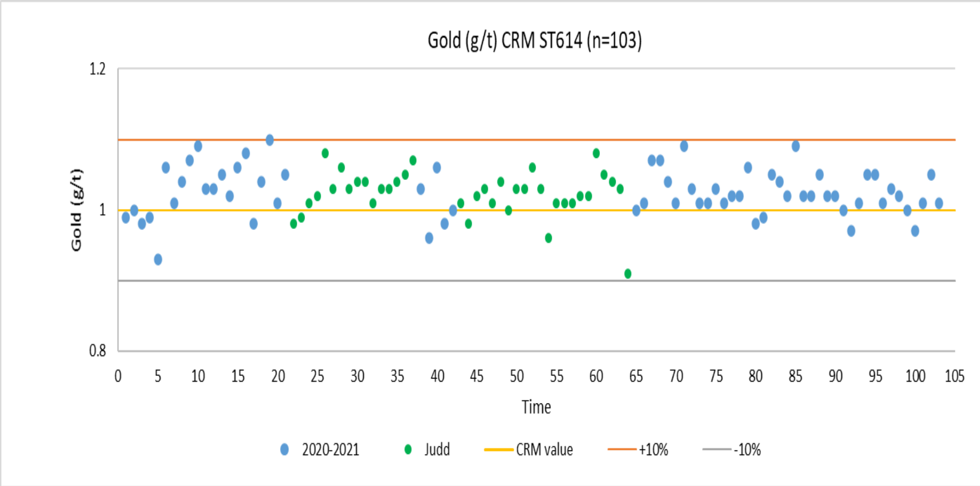


Figure 11-9 Low Grade Gold Standard ST614

Figure 11-10 shows the results for the likely gold cut off grade standard G312-5, over the time period 2018 to 2020. The results indicate good accuracy and precision.

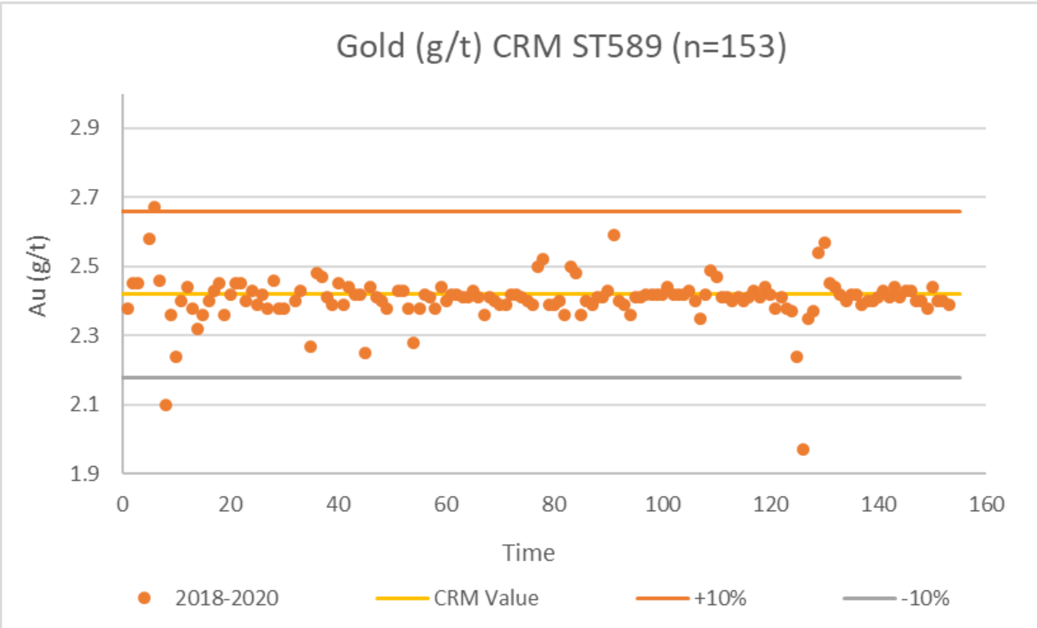


Figure 11-10 Low Grade Gold Standard ST589

Figure 11-11 shows results for a likely mine cutoff grade standard G915-2 for the time period 2018 to 2023. There is a very slight positive bias in the results, however, they are within acceptable limits. Some variability in the 2018-2020 time period, however, overall, the gold standard G915-2 shows good accuracy and precision which seems to have improved over time.

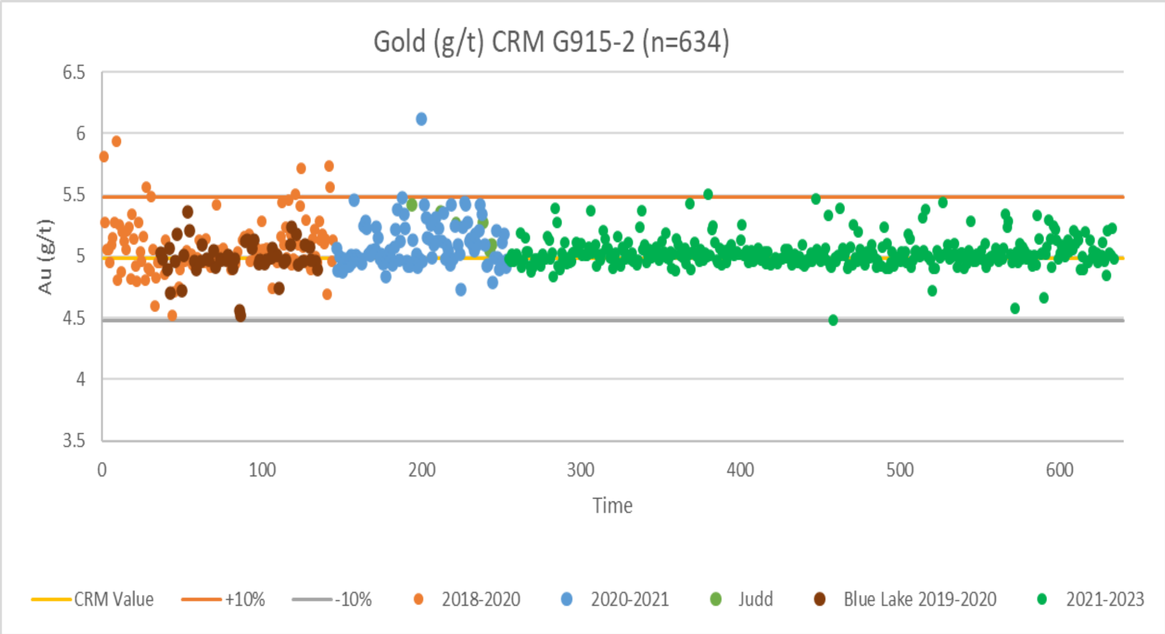


Figure 11-11 Gold Standard (G915-2)

Results for the head grade standard, CRM ST725 (Figure 11-12), for the time period 2018 to 2021 show a varying minor bias over time with under-reported values early on switching to over reported values in later times, but all within acceptable limits. Thus, the results are reasonably accurate, but precision has worsened over time.

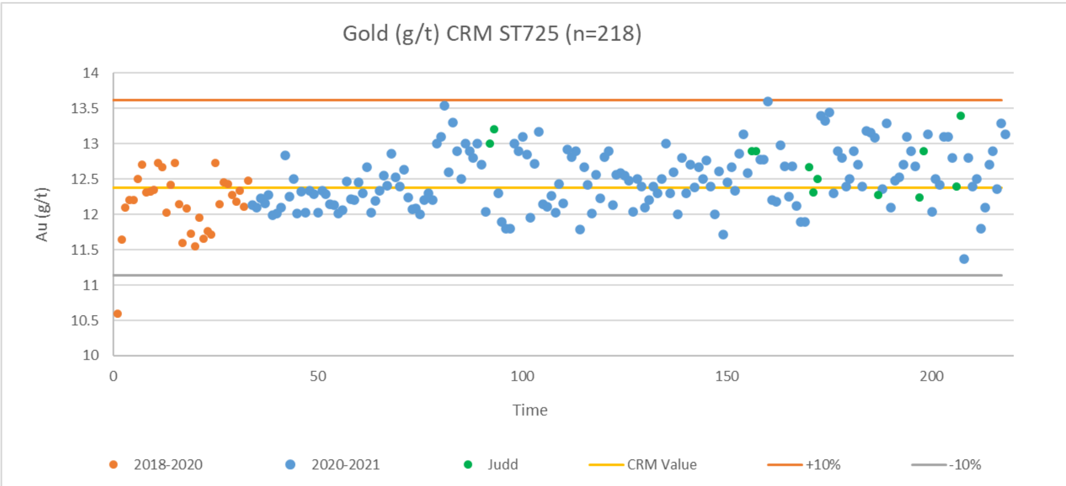


Figure 11-12 High Grade Standard (ST725)

The reported results for CRM G919-5 (Figure 11-13), indicate a variable bias over the time period 2021 to 2023. There is a positive bias initially before reverting to a negative bias with an accompanying lack of precision. However, all results are just within acceptable limits of variation. Continued use of this standard needs to be considered.

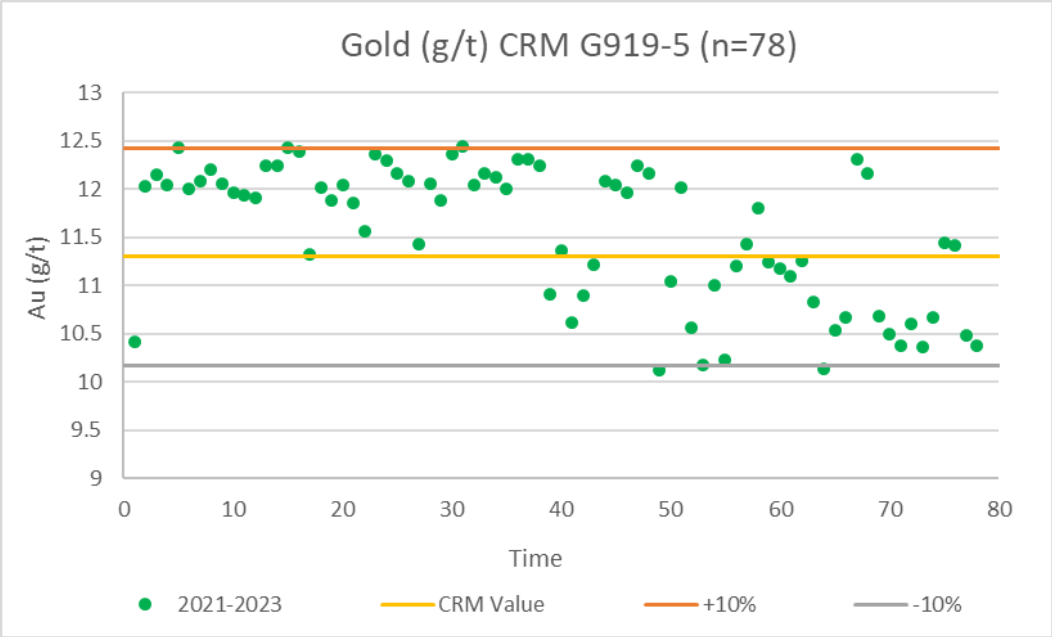


Figure 11-13 High Grade Standard (G919-5)

Assay results for the high grade standard G916-6 (Figure 11-14), show an overall consistent positive bias of about 5% in the reporting although there is reasonably good precision. However, all results are within acceptable limits.

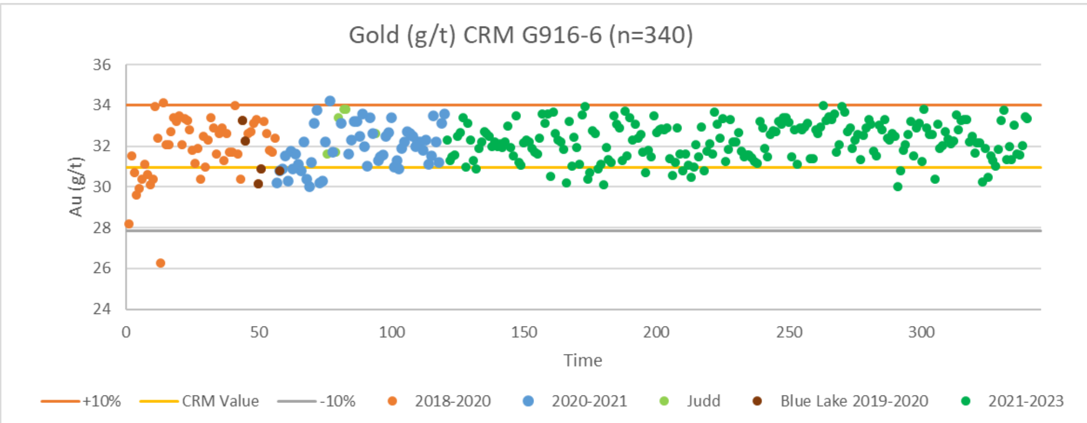


Figure 11-14 High Grade Standard (G916-6)

11.5.3 Copper Standards

Copper standard GBM315-10, Figure 11-15, showed an initial small positive bias in the 2018-2020 data but performance changed to a slight negative bias in the 2021-2023 data. The biases are not considered significant noting the low grade of the standard with precision being reasonable.

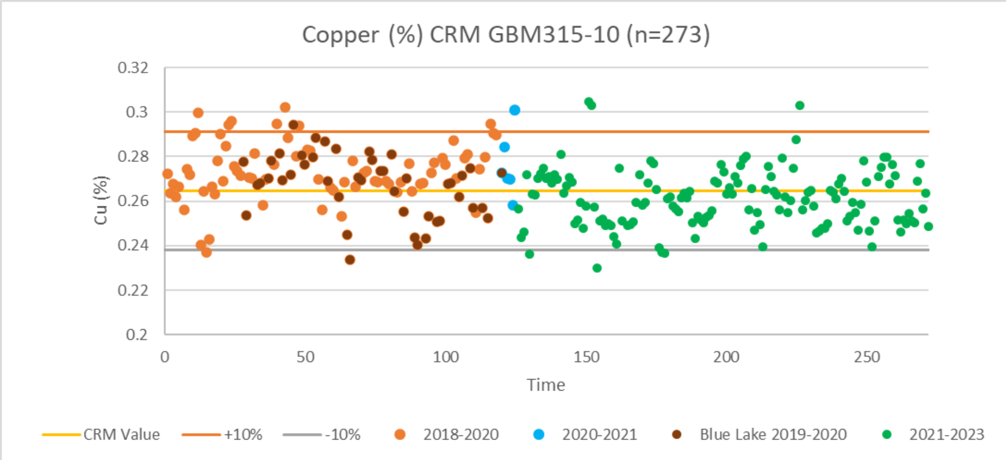


Figure 11-15 Low Grade Copper Standard GBM315-10

Copper standard GBM910-4, Figure 11-16 is a higher-grade standard that shows a generally consistent negative bias of about 4-5%. However, the precision is reasonable and all results are within acceptable limits.

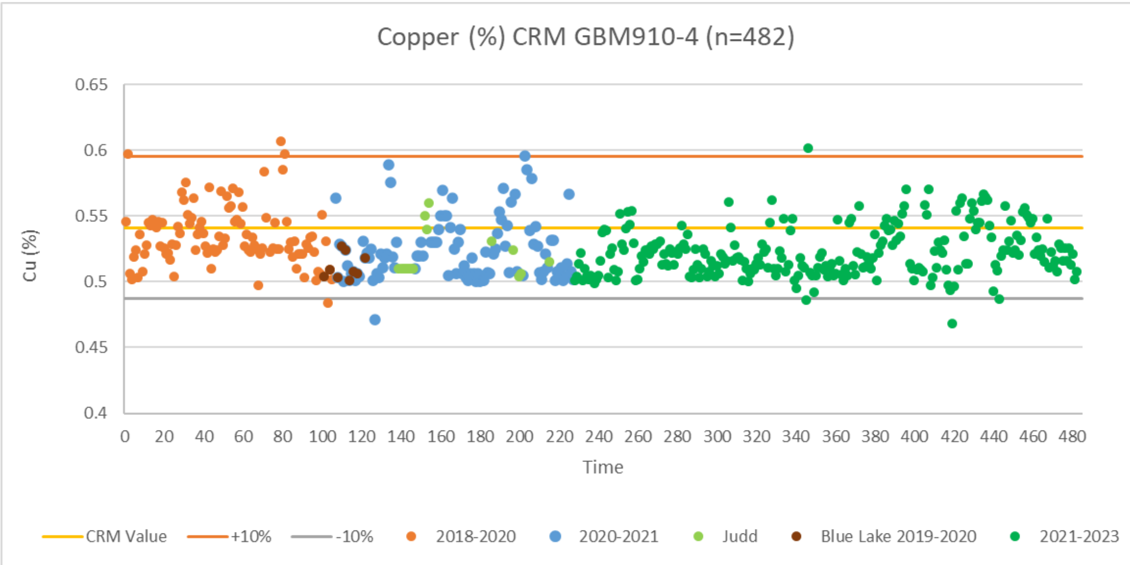


Figure 11-16 Copper Standard GBM910-4

GBM910-6, Figure 11-17 is a higher grade standard with a persistent negative bias of 2-3% throughout the time period of mine operation. However, the precision is reasonably good and the results are within acceptable limits.

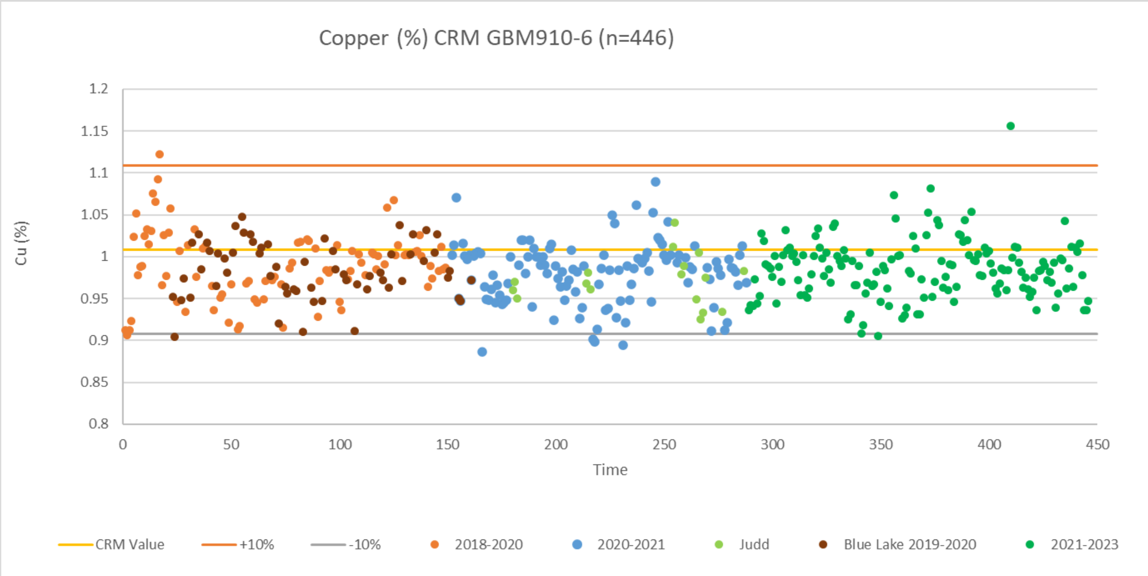


Figure 11-17 Copper Standard GBM910-6

GBM309-4, Figure 11-18 is a relatively high-grade copper standard and exhibits a positive bias of 2-3%, but precision is reasonably good. All results are within acceptable limits.

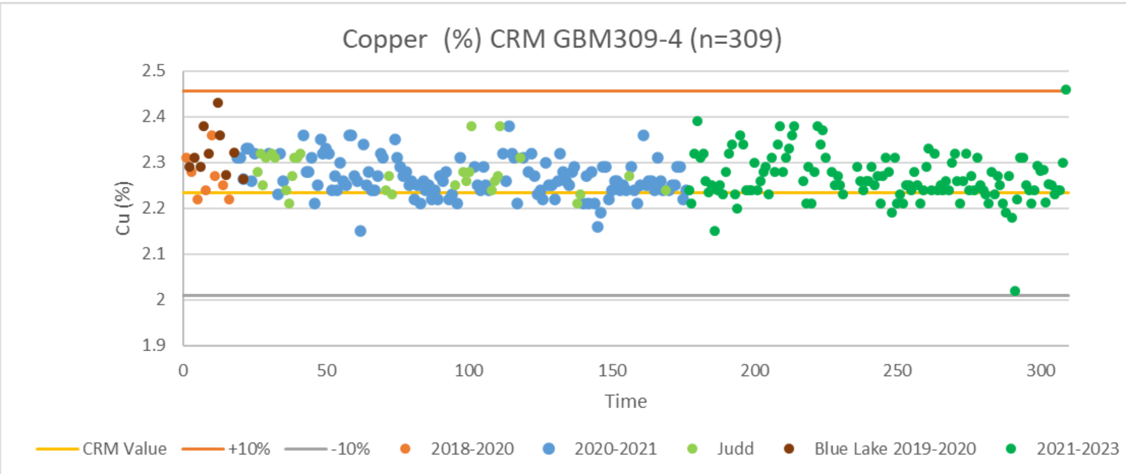


Figure 11-18 High Grade Copper Standard GBM309-4

11.5.4 Silver Standards

The silver method used by InterTek at the on-site laboratory has accuracy limitations at low silver levels with a range of ±3 ppm (g/t) for samples containing less than 10 ppm (g/t) silver. For silver standards under 10 ppm (g/t), upper and lower bound lines of ±1.5 ppm (g/t) have been plotted on the graphs. To date the cost of the silver analysis method required for increased accuracy, has not been entertained as the revenue generated by the Mine for silver is minor, compared to the benefit of improved silver assay accuracy and has not been supported by K92ML's economic analysis of the matter.

Silver standard GBM309-4 is a high grade standard which demonstrates a modest and reasonably consistent negative bias of 2-3% over the 2018 to 2023 time period with reasonable precision as shown in Figure 11-19.

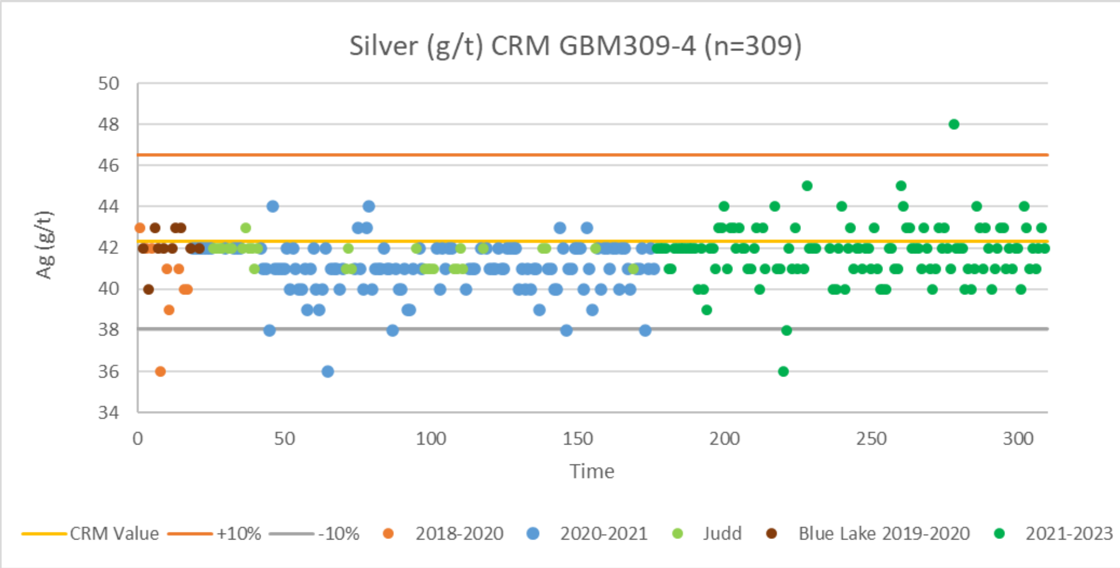


Figure 11-19 Silver Standard GBM309-4

11.5.5 Blanks Results

Blank gravel standards were inserted into the sample sequence on the 21st, 41st sample etc. If a batch was less than 20 samples a blank was inserted and other QAQC samples after the mineral sample sequence. The blank material consisted initially of clean crushed phyllite which was then replaced by dacitic intrusives in 2021, collected several kilometres away from any mineralization. The blank material was crushed in isolation from other samples; several samples were submitted to the laboratory for analysis to demonstrate its suitability as a blank prior to its use.

The gold blank assay QAQC results are presented in (Figure 11-20). In summary gold assay results for the blanks are within acceptable limits of variation and are continually being monitored. In the 2018-2020 time period gold results for the blank assaying greater than 0.1 g/t came to approximately 2% of all blanks submitted and the reasons were investigated. Two of the elevated gold blank values > 0.1 g/t returned from the laboratory had mineral sample assays greater than 1 g/t before them in the sample sequence, 9 blank values had samples before them > 0.1 g/t and the remaining 8 samples had values <0.1 g/t before them. This suggests that the Lab procedures have infrequently allowed for minor contamination between samples. Improvements in the lab procedure occurred in the 2020-2021 period and into the 2021-2023 period and have reduced the contamination mainly due to tightening up on cleanliness.

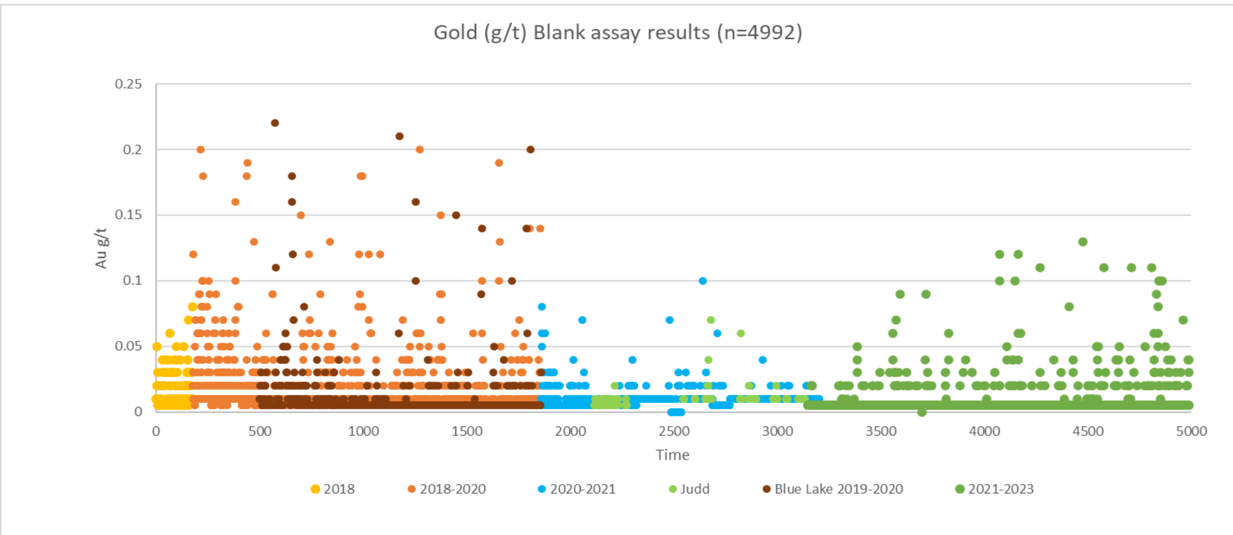


Figure 11-20 Gold Blank Results

The copper assay results for the blanks are presented in Figure 11-21. Overall, the copper blank results are within acceptable limits of variation. For the copper blank results, there was a minor contamination in the early drilling period that improved in the 2018-2020 drilling period. Results for the 2020-2021 campaign have suggested some minor contamination however, this coincided with a change in the blank standard material, which possibly accounts for the change in the copper results. In the 2021-2023 period copper blank values continue to be within acceptable limits of variation

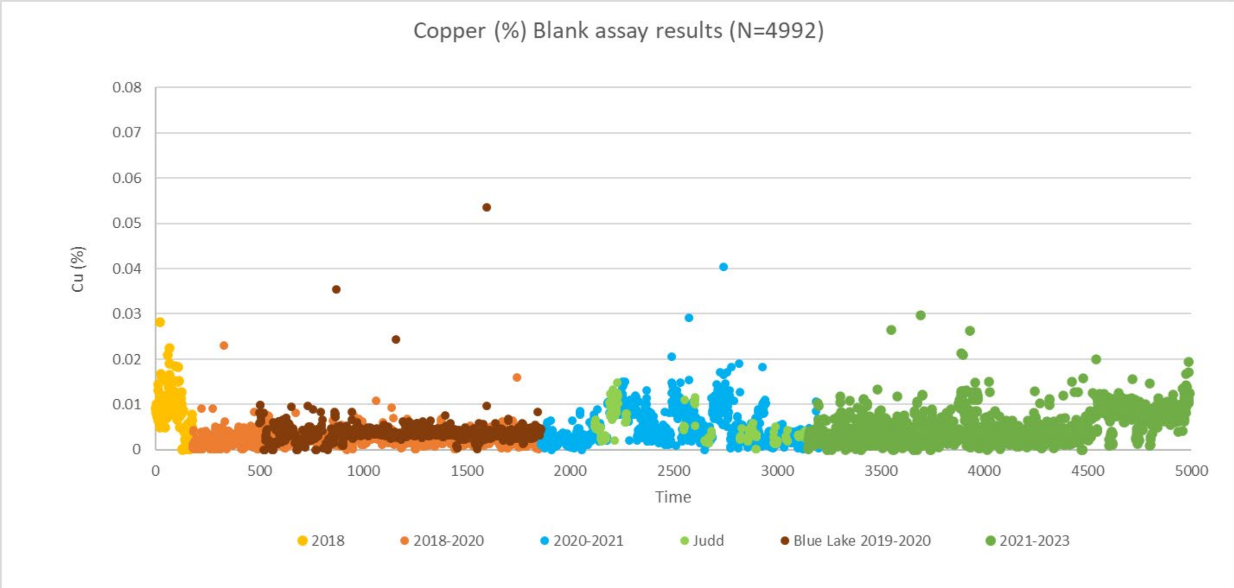


Figure 11-21 Copper Blank Results

Blank Values of silver were inconclusive because of the inaccuracy of the analytical method.

11.5.6 Laboratory Duplicates

Laboratory (or pulp) duplicates were inserted every 23rd sample in the sample submission sequence for the diamond core samples.

Laboratory duplicates for the face sampling were only completed for the 2018-2020 time period. The face sampling QAQC was discontinued after this time as the 2018 to 2020 results matched the core sample results.

The results of the gold analyses of the pulp duplicates for the diamond drilling showed a good match with the original sample with no evidence of any bias (Figure 11-22).

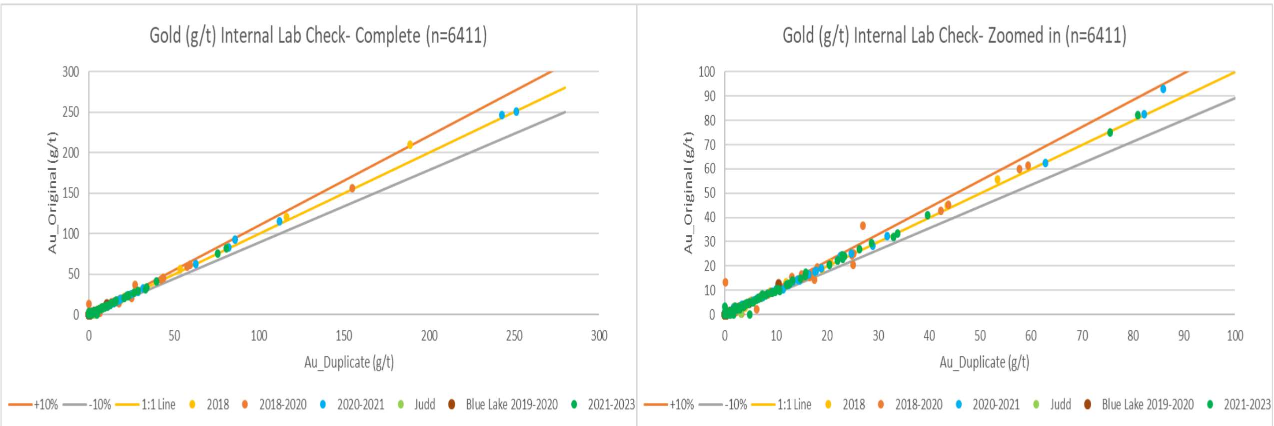


Figure 11-22 Laboratory Duplicates for Gold

The results of the copper pulp duplicates showed a good match with the original sample with no evidence of any bias (Figure 11-23).

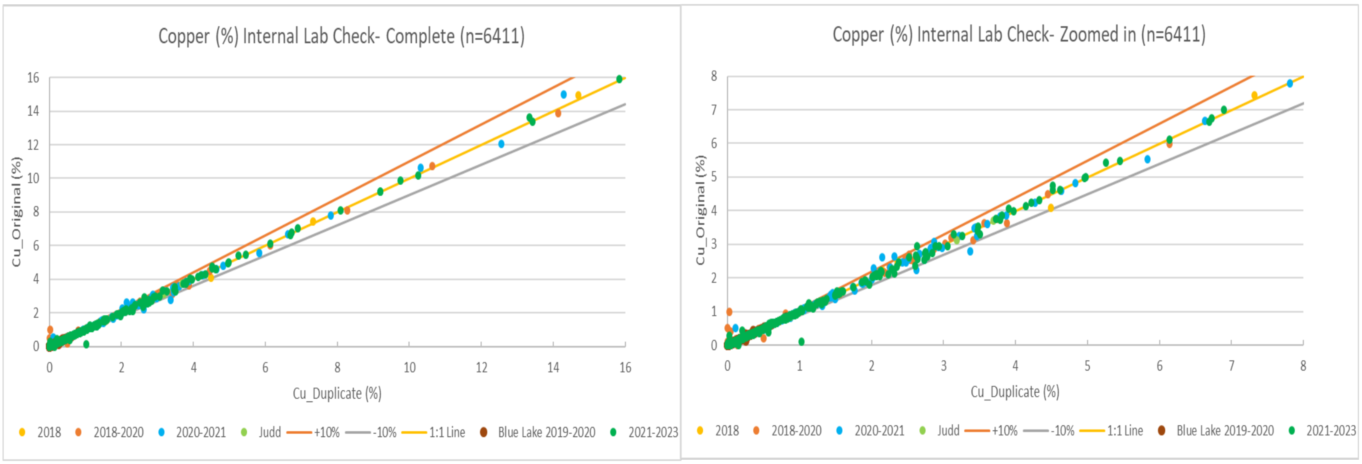


Figure 11-23 Laboratory Duplicates for Copper

The results of the silver pulp duplicates showed a reasonably good match with the original sample with no evidence of any bias (Figure 11-24). The bigger scatter with higher silver grades is noted, but has no obvious bias.

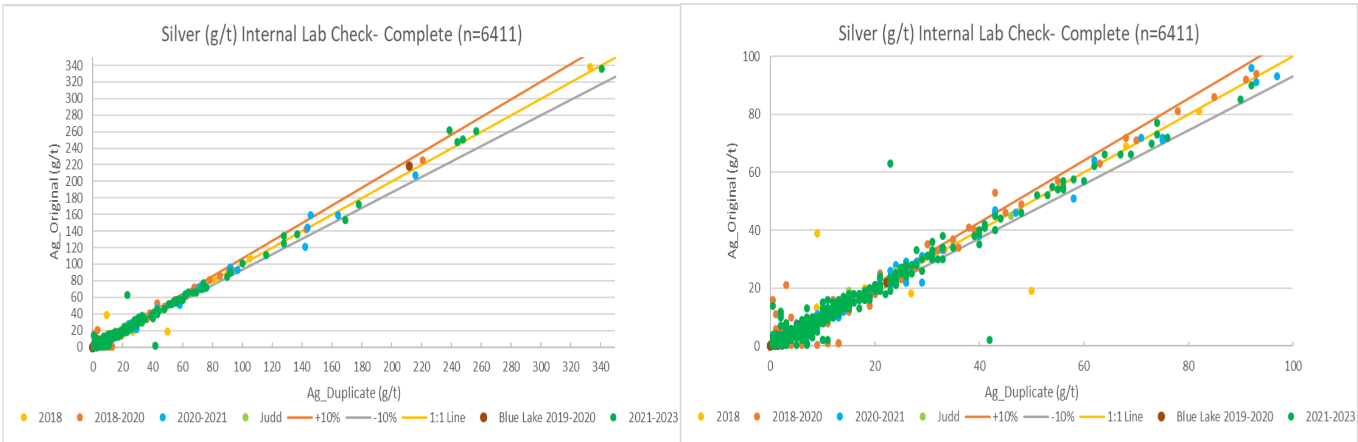


Figure 11-24 Laboratory duplicates for silver

11.5.7 Second Laboratory Checks

A program of check assaying was carried out by K92ML whereby a series of 100 to 200 mineralized pulp core samples including CRMs were submitted to a secondary laboratory (SGS, in Townsville Queensland). The SGS assaying used the same techniques as Intertek with the comparison of results between laboratories providing a check on the original laboratory's analytical accuracy. Second laboratory checks are a routine process completed approximately on a 6-month cycle.

The periods have been graphed according to K92ML's MREs.

The first graph (Figure 11-25) shows the gold results for the different periods. There is a slight positive bias to the original onsite Intertek laboratory for the 2020-2021 MRE period, ongoing monitoring improved the results in the 2021-2023 MRE period.

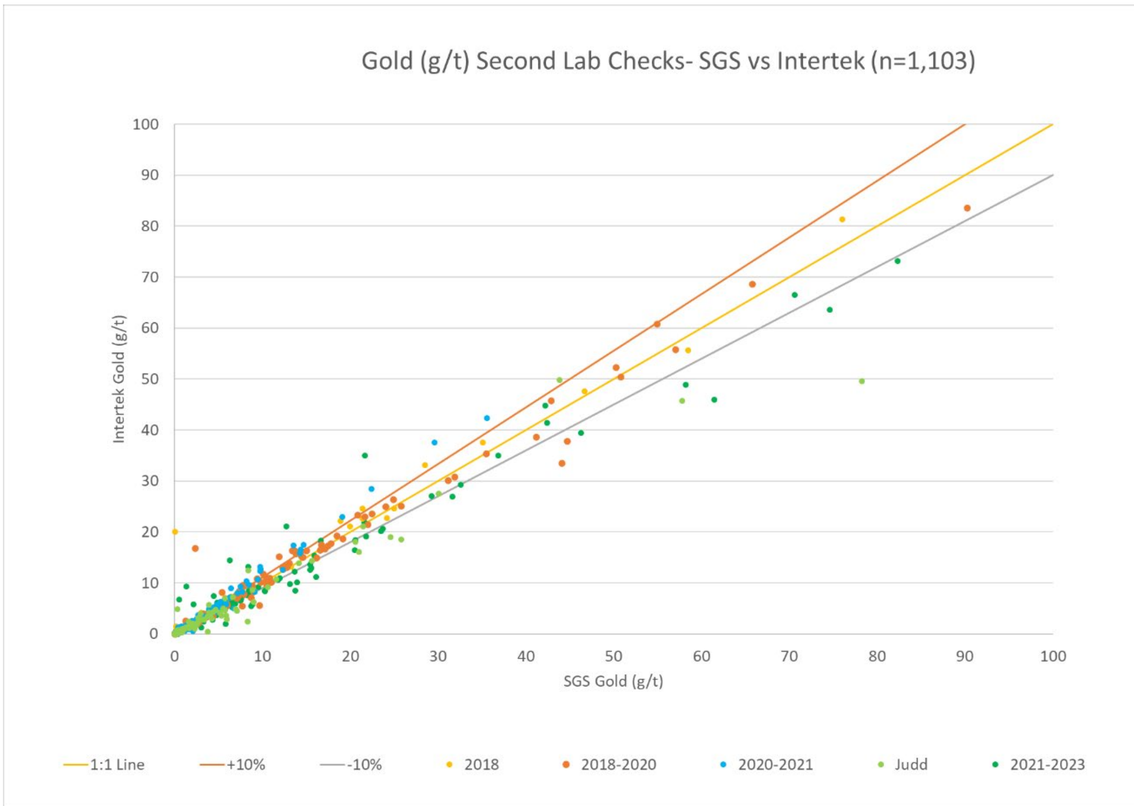


Figure 11-25 Gold- Second Laboratory Checks

The results for copper show no obvious bias (Figure 11-26). The extreme value pairs (3 sets at 12-15% Cu) are not included in the graph but present no issues.

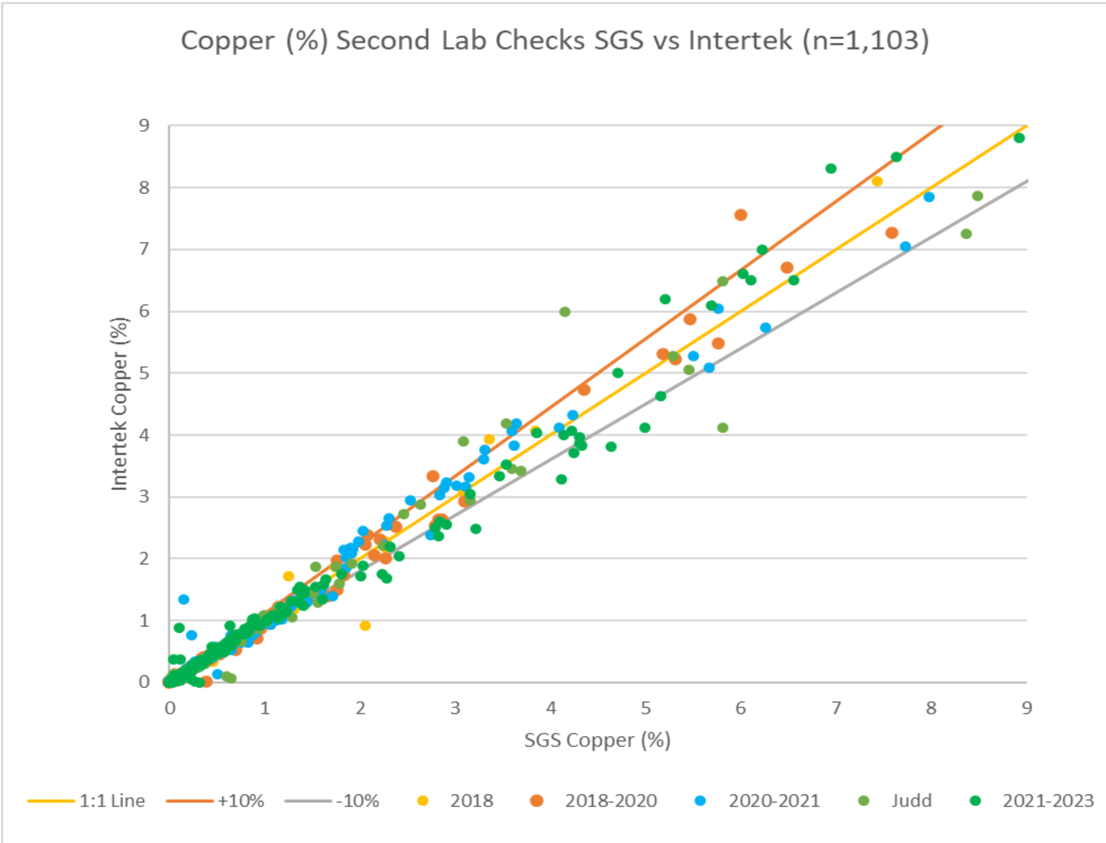


Figure 11-26 Copper- Second Laboratory Checks

The results for silver (Figure 11-27) show a bit more variability, particularly for the 2018 and 2018-20 drilling campaigns. There is a slight positive bias existing for medium to high grades associated with the second laboratory, SGS, but is not considered a significant issue. The bias does not continue for silver grades above 280 g/t.

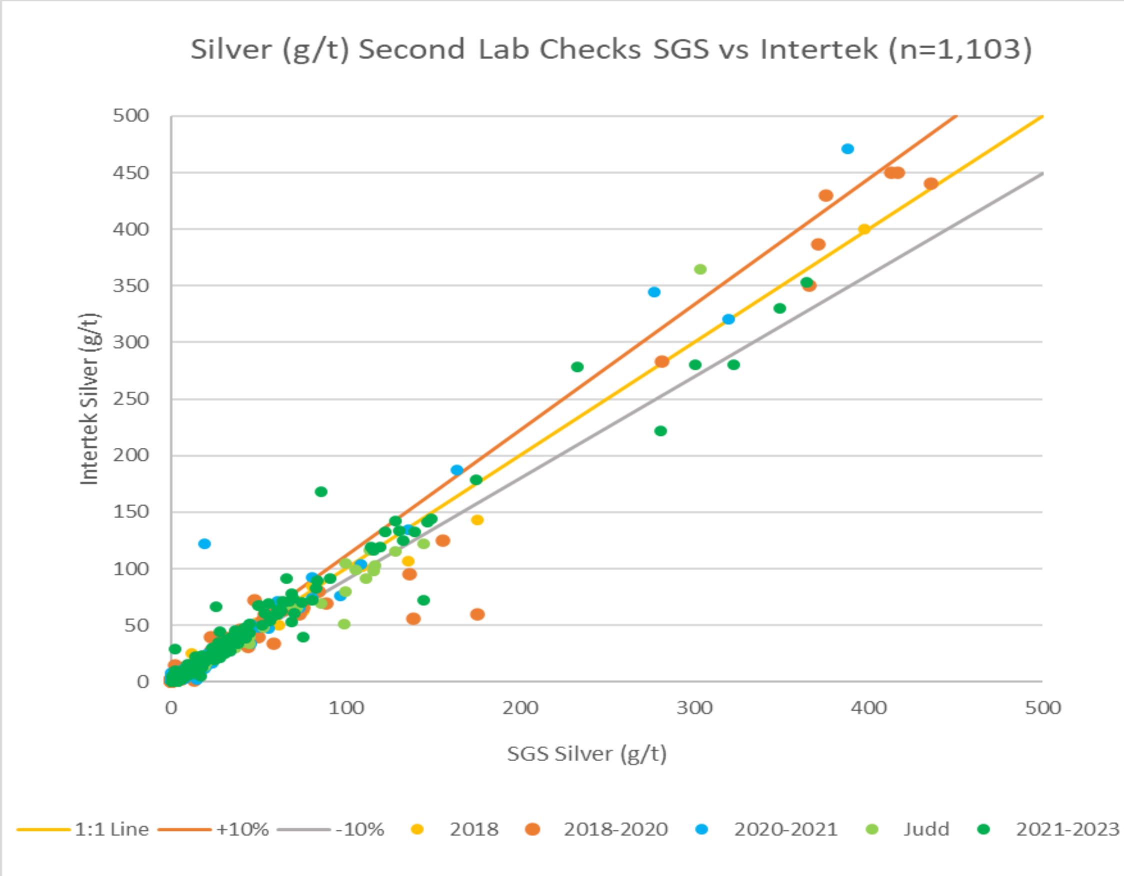


Figure 11-27 Silver- Second Laboratory Checks

11.5.8 Hole Twinning

A twin hole program was conducted by K92ML in the 2020-2022 period for checking any bias in the Barrick sample preparation and assaying. Four surface holes were twinned.

In summary the twinning shows no overall bias between the Barrick and K92ML data, however, the program highlighted the variable nature of mineralization and lack of short scale grade continuity, which is normal for this type of high-grade gold deposit, (Kohler, A., Tear, S., and Woodward, A., 2022).

11.5.9 Coarse Rejects

No coarse reject analysis was completed.

11.6 QAQC Summary

- The 2018-2023 K92ML QA/QC programme is of an acceptable level and conforms to common industry practice.
- The programme in the 2021-2023 period has included the use of a range of CRMs for gold, copper and silver, blank material, laboratory duplicates, and 2nd laboratory checks. The programme in the 2020-2021 period has included some twin hole drilling.
- The gold standards show a very small positive bias of 1 to 2% that seems to be slightly increasing with time. This needs to be continuously monitored but it may be a function of the standards themselves rather than any fundamental problem with the laboratory. However, all recent results are

acceptable and considered reasonably accurate and the low level of bias is not considered significant at this stage. The copper standard results indicate an under reporting by 2 to 3% for values under 1.5% copper and for higher copper values ~2.2% a slight positive bias has occurred. However, all results are within acceptable limits of variation. The silver standard results are inconclusive due to the relatively high lower detection limit of 1 ppm (g/t) and accuracy of analysis of ±3 ppm (g/t) for the K92ML laboratory and the relative low level of some of the standards (1 - 5 ppm). In the 2021-2023 period tightening up of procedures and follow up of failure of standards greatly improved and meets industry standards.

- Blanks comprise a locally sourced dacite or phyllite with fragments generally >5 cm in size. The assay results for the blanks indicate possible very minor contamination but the levels are low enough to be considered as not significant. The amount of gold contamination has reduced over time.
- Laboratory duplicates comprise a second pulverised pulp of the original sample analysed by the K92ML laboratory with the results showing no issue with the sample homogenisation and subsequent acid digest and analysis. A good range of assay values have been included in the sample selection.
- The second laboratory checks show no obvious bias. However, there is variation in the results, this is not considered significant at this stage. A good range of assay values has been included in the sample selection.
- No coarse reject samples have been tested.
- No field duplicates were taken for the drill core.
- QAQC samples for the face sampling were completed in the period 2018 to 2020. The programme was discontinued by K92ML subsequently with reliance for the sample prep and assaying transferred to the core sample QAQC results.
- Hole twinning carried out in the 2020-2021 period has comprised 4 pairs of an original Barrick diamond drillhole with a K92ML diamond twin drillhole. The results show some degree of short scale variability in gold grade between the twin hole intercepts for the K1 and K2 lodes, as might be expected. Comparison of the overall average gold, copper and silver grades for the relevant lode intercepts has indicated no systematic bias allowing for the conclusion that there are no issues with the Barrick analyses and that the data can be used in the resource estimation.
- The QAQC data for the Highlands drilling includes standards and duplicates and indicates no issues with the data.
- None of the above issues are considered critical but the cumulative effect of various minor issues could have a slight negative impact on the resource classification.
- It is concluded that there are no major issues with the sample preparation or assaying of the drill core and face samples for the K92ML exploration work. The 2018-2023 K92ML QA/QC programme is of an acceptable level and conforms to common industry practice.

11.7 QAQC Recommendations

During the compilation of the QA/QC section for this report K92ML and H&SC have been in constant communication with regards to improving the QA/QC program, which will flow through to a further improvement in the quality of any future resource estimates. The only recommendation is to maintain the procedures and continuously monitor the ongoing program.

12. DATA VERIFICATION

12.1 Site Visits

Andrew Kohler is a full-time employee of K92 Mining Limited. As Chief Geologist he visited the Kainantu site in February, April, June and August of 2023 and prior to that was regularly at the mine site as the Underground Mine Geology & Mine Exploration Manager, on a roster basis that was completed in March 2021. Mr Kohler regularly inspected the diamond drilling sites and core produced, and monitored the core handling, sampling and the assaying processes. Also, numerous visits were made underground to all the ore drives for Kora and for Judd. During these visits detailed verification of the drilling and face sampling processes took place.

Site visits to Kainantu were completed by Simon Tear of H&SC Consultants Pty Ltd in October 2018 and in June 2023. The visits included an inspection of underground workings including exploration drilling and geological assessment, an inspection of the processing plant including the on-site laboratory assay facilities, visual checks of randomly selected laboratory-issued assay sheets and a review of drill core.

The site visit completed by Simon Tear from the 24th to the 28th of June 2023. Comprised the following data validation:

1. Reviewing drill core from various drilling campaigns since the 2022 resource estimate for K1, K2, K Link and Judd Lodes.
2. Reviewing data entry procedures for the underground drilling.
3. Inspection of underground faces for mineralization at Judd, K1 and K2.
4. Helicopter-supported visual checks on Kora surface exploration drillholes.
5. Discussion with mine staff on the geological understanding of the Kora mineral lodes.

In addition, a series of database checks were completed on 11 drillholes randomly selected by H&SC, Table 12-1. List of Holes for Core Inspection. These are the same holes that were selected for core viewing i.e. 11 holes for 1,425 m of core reviewed covering both surface and underground drilling in different campaigns.

Table 12-1 List of Holes for Core Inspection

Hole	Type	From	To	K1	K-Link	K2	Kora South	Judd
JDD0120	UG	104	125					YES
		150	243.4					
KMDD0486	UG	76.8	148.03	YES	YES	YES		
KMDD0490	UG	33	60					
		69	137.2	YES	YES	YES		
KMDD0542	UG	0	66.41	YES	YES	YES		
KMDD0554	UG	99.61	155.4	YES		YES		
KMDD0565	UG	40.36	131.6			YES		
KUDD0006	Surf	118.07	303.24	YES		YES	YES	
KUDD0014	Surf	137.5	173.1	YES			YES	
		341.3	394.88	YES		YES		
		490.9	505.28					YES
KUDD0035	Surf	78.8	101				YES	YES
		248.77	423.17	YES		YES		
KODD0007	Surf	178.27	340.28	YES	YES	YES		
KODD0013	Surf	54.7	142.3	YES		YES		
		436.82	633.02					YES

Validation checks included:

1. Comparing logged geology as per the database with the actual core.
2. Checking all mineralized zones from the assays with the actual core.
3. Checking original assay sheets with database entries for gold, copper and silver.
4. Checking original downhole surveys with database entries.

Generally, the logged geology matched the visual observations and elements of mineralization e.g. sulphides, quartz veining etc could be recognized in the core to match the mineral zones identified in the assay table. Only one or two minor discrepancies were recognized, from the surface holes, between the assays and the geology which were pointed out to K92ML personnel with the recommendation of some check sampling and assaying of quarter core. Many of the issues identified have been dealt with by K92ML personnel.

An underground inspection of selected faces was undertaken providing insights into the geology of the main mineral lodes. Faces were checked for Judd (1325 level Sth), K1 (1305 level and 1130 level), K2 (1170 level). Actual access to the faces was limited due to unbolted areas close to the face.

12.2 Limitations

No independent samples were collected for analysis during the site visits. Industry standard procedures appear to have been used.

12.3 Verification Opinion

Based on the data verification performed, it is the QP's opinion that the collar coordinates, downhole surveys, lithologies, and assay results are considered suitable to support the MRE.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

13.1.1 Overview

This section covers processing information derived from the 2021 testwork and from plant operating records.

The 2021 testwork evaluated samples from K92ML plant operations, taken during separate campaigns on K1, K2, and Judd ores. Ore for these campaigns was supplied from a combination of mining and development activities. Additional testwork was performed on a series of variability samples from exploration drilling. This current report gives a brief review of the testwork results, focussing on important aspects of the treatment response of K92ML ores. Reference should be made to the 2022 IDP document for comprehensive details and interpretation of the testwork results. This document is available from https://k92mining.com/documents/technical-reports/.

For the 2022 IDP, plant performance data from the period January 2021 to March 2022 was used in conjunction with the testwork results to predict future plant performance. The current report considers plant performance over the period from January 2018 to June 2024 for the performance prediction. This longer interval covers changes and additions to the processing plant flowsheet during the stage 2 (400k tpa) and 2A (500k tpa) expansions and their effect on performance and throughput capability.

13.1.2 Current IDP Development Activities

On July 24, 2023. K92ML announced awarding the contract for the 1.2M tpa Stage 3 Expansion Process Plant to GR Engineering Services Ltd (GRES). By mid-2024, the plant design had been finalised, long lead items were on order and construction work on the new plant was in progress. Similar progress was being made on underground mining developments and surface infrastructure packages.

The 2022 IDP

On September 12, 2022, K92ML announced the results of its Integrated Development Plan for the Kainantu Gold Mine Project in PNG. The updated IDP comprised two scenarios: 1) the Kainantu Stage 3 Expansion DFS Case; and 2) the Kainantu Stage 4 Expansion PEA. Both cases involved the construction of a new 1.2 million tonnes per annum process plant to accept ore from the Kora and Judd deposits.

Lycopodium Minerals Pty Ltd prepared a detailed design for the new processing plant located adjacent to the existing plant. The design was based on a combination of ore processing characteristics from the testwork, current plant operating experience, established best practice for gold/coper concentrator design, and the specifics of the Kainantu plant site. Metals recovery and concentrate grade performance estimates were based on existing plant operations and results from the testwork program.

13.2 The 2021 Testwork Program

13.2.1 Testwork Schedule

A series of tests were conducted on samples from K92ML plant operations (during campaigns on ore from mining and development activities), and on core samples from exploration drilling. The sequence of testwork was as follows:

Sample receiving, preparation and preliminary compositing:

- Belt cuts of from campaigns on individual ores, K1, K2 and Judd.
- Stream samples (final concentrate, cleaner tailings, and final tailings) from plant operations whilst processing individual ores.
- Diamond drill core samples from exploration drilling.

Mineralogical evaluation of stream samples from plant operations:

- QEMSCAN Automated scanning electron microscopy and image analysis to determines the nature and mode of occurrence of gold, sulphides and gangue minerals.
- Laser Ablation ICP-MS analysis of individual sulphide grains for gold and individual silicate grains for fluorine.
- Bulk mineralogy, including size by size analysis, screen fire assays, and sequential leach determination of gold occurrence.
- Comminution testwork on the belt cut samples including "The JKTech SMC Laboratory Comminution Test", plus standard Bond tests: Crushing Work Index, Rod Mill Work Index, Ball Mill Work Index, and Abrasion Index.
- Bead milling, thickening, pressure filtration and rheology testwork on a sample of concentrate from the K92ML plant.
- Sample preparation for flotation testwork: recombining fragments from the Comminution Tests to three individual ore type samples and a "Master Composite" sample.
- Gravity recoverable gold tests, on the three individual ore type samples, using the industry standard three-stage procedure.

Batch flotation tests on samples of the three individual ore types and Master Composite:

- Rougher rate tests at P_{80} 0.075 mm using the same reagent suite as the operating plant to investigate required dosage rates.
- Rougher rate tests at grind P_{80}'s from 0.053 to 0.150 mm to investigate primary grind requirements.
- A rougher cleaner, recleaner test incorporating bulk sulphide roughing followed by selective cleaning.
- Rougher rate tests at a selected grind P_{80} using four alternative reagent suites.
- Rougher, cleaner recleaner tests using the selected reagents and with three levels of concentrate regrind.
- Locked cycle rougher, cleaner and recleaner tests on the Master Composite to simulate two flowsheets, one incorporating concentrate regrind and the other without regrind.
- Variability testwork on eight composites of diamond drill core that were selected to represent mineralized zones within the Mineral Resource. Each variability sample was subject to the following testwork:
  - JKTech SMC Laboratory Comminution Test.
  - Standard Bond Ball Mill Work Index Test.
  - A single rougher rate flotation test using standard conditions.

All flotation testwork samples were analysed for the following components: Au (duplicate assay), Cu, Fe, Zn, Ag, S, F, SiO_2, Al_2O_3, CaO. More detailed analytical work for a wider range of elements was included in the mineralogy work.

Samples Tested

A full list of the samples used in testwork, with head analyses for the main elements of interest, is shown below in Table 13-1. There is a wide range of variability of gold and copper grades in the ore samples. It is also interesting to note that the fluorine levels in the three 'final concentrate' plant samples are above penalty levels. A full list of stream samples from plant operations that were sent in for mineralogical evaluation is shown below in Table 13-2.

Table 13-1 Samples Used in the 2021 Testwork

Analysis Composite Name	Au g/t	Cu %	S %	F ppm	Zn %
Belt Cut Samples					
Fresh K1 Ore Belt Cuts	7.582	0.400	8.765	2400	0.032
Fresh K2 Ore Belt Cuts	28.43	2.260	6.350	2600	0.032
Fresh Judd Ore Belt Cuts	3.700	0.525	4.745	1600	0.134
Variability Samples from Diamond Drill Core					
J1A	5.574	1.975	8.24	800	0.065
J1B	17.16	1.175	8.8	800	0.182
K1A	25.45	3.285	15.64	800	0.012
K1B	18.43	0.255	11.10	1400	0.044
KLA	9.154	0.675	9.885	1000	0.144
K2A	7.242	1.015	11.57	800	2.193
K2B	3.226	0.99	9.79	1000	0.076
K2C	8.516	2.19	17.06	800	0.057
Plant Sample of Flotation Concentrate for Bead Milling, Thickening and Filtration Testwork (taken Q4 2021)					
20 kg Concentrate Composite	279	8.41	23.3	1000	1.01

Table 13-2 Samples Sent for Mineralogical Examination

Analysis Composite Sample Name	Au g/t	Cu %	S %	F ppm	Zn %
Plant Stream Samples for Mineralogy					
K1 Ore Final Tailings	2.775	0.030	6.700	3000	0.023
K2 Ore Final Tailings	5.156	1.565	7.695	2000	0.029
Judd Ore Final Tailings	1.140	0.105	4.410	2000	0.056
K1 Ore Final Concentrate	549.1	10.690	22.39	1200	0.975
K2 Ore Final Concentrate	75.61	25.315	25.22	1000	0.530
Judd Ore Final Concentrate	66.47	12.130	16.72	1200	1.523
K1 Ore Cleaner Tailings	23.27	0.355	6.225	2800	0.091
K2 Ore Cleaner Tailings	10.91	3.140	9.115	2200	0.061
Judd Ore Cleaner Tail	5.596	0.775	5.155	2200	0.187

13.2.2 Physical Location Origins of Samples Tested

The belt cut samples were taken in the first quarter of 2021 and were representative of ore mined at that time. The variability samples were assembled from exploration drill core. Physical locations of where ore was mined in Q1 2021, and of the drill core for the variability samples were recorded.

13.3 Mineralogical Evaluation of Processing Plant Samples

13.3.1 Mineralogical Examination Methods Used

Plant samples have been evaluated by both microscopy and bulk mineralogy methods. The microscopic methods included optical microscopy, QEMSCAN, and Laser Ablation ICP-MS spot analyses. Bulk mineralogical evaluations included size by size analyses, mineral abundance tabulation, mineral particle size tabulation, and screen fire assays. All work except the screen fire assays was handled by ALS Laboratories, Perth.

Optical microscopy was used to identify coarse gold grains. QEMSCAN, with its electron microprobe analysis capability and image analysis software was used for identifying minerals and quantifying parameters such as grain size and degree of locking of mineral particles.

Additional information on gold and fluorine mineralogy was provided by (LA-ICP-MS) of selected sulphide and gangue mineral grains. QEMSCAN has some limitations regarding gold mineralogy due to the relatively low concentrations of gold in all samples. Furthermore, light elements such as fluorine cannot be determined by electron microprobe analysis.

13.3.2 Gold Mineralogy

In the plant, a combination of gravity concentration and froth flotation has been achieving high gold recoveries. Well liberated gold mineralization, as both free gold and telluride, is strongly recovered.

Grains of native gold and gold tellurides were observed in all samples. The number and mean size of particles observed in each sample was much in line with the gold grade of the sample. Native gold and telluride grains in the cleaner tails and final tails samples were fewer in number, finer and less well liberated than in the final concentrate.

Nevertheless, a significant number of gold bearing particles were observed in the final tailings. Gold locking data for tailings samples showed significant losses of liberated gold telluride particles in the minus 20-micron size fraction of the final tailings plus some pyrite locked native gold and gold tellurides in the coarser size fractions. This sample received by ALS could be considered unusual in that the gold value is much higher than typically reported by the plant.

In the K1 samples, eight coarse gold grains, ranging in size between 100 µm to 200 µm, were observed in optical microscopy. No coarse gold grains were identified in K2 or Judd samples. QEMSCAN identified large numbers of native gold grains and gold telluride grains in all samples. The native gold grains could all be classified as “electrum”, a gold/silver alloy. Telluride grains were gold/silver/copper telluride.

13.3.3 Laser Ablation ICP-MS Gold Analysis of Sulphide Grains

Results for the three ore types were as shown below in Table 13-3. Each analysis value is the average of gold levels measured in pyrite or chalcopyrite grains in the final concentrate, cleaner tails, and final tailings for an individual ore type. The analyses are for gold in solid solution and in sub-micron particles; they do not include native gold or telluride grains that are locked in pyrite or chalcopyrite.

Table 13-3 Average Gold Contents of Pyrite and Chalcopyrite Grains

Ore Type	Pyrite Grains g/t	Chalcopyrite Grains g/t
K1	1.7	0.57
K2	0.19	0.22
Judd	3.8	0.27

LA-ICP-MS results showed very low levels of solid solution gold in chalcopyrite and pyrite. Calculations showed that in the final tailings samples, the gold hosted by pyrite was less than 10% of the total.

Copper Mineralogy

Chalcopyrite is the main copper bearing mineral in all samples. Minor amounts of copper minerals bornite, chalcocite, covellite and tennantite-tetrahedrite are also present.

Chalcopyrite and pyrite are the dominant sulphides. Selective flotation with high recoveries of chalcopyrite whilst rejecting most of the pyrite to final tailings was achieved for all three ore types.

Pyrite is less abundant in all three Judd products compared with K1 and K2 products. About 50% of the pyrite recovered to final concentrate was in composite particles with some chalcopyrite. Pyrite liberation in the cleaner tailings and final tailings is high at between 75% and 80%.

Mode of Occurrence of Copper in the Plant Samples

The main findings for the variations in mode of occurrence of copper are summarized below in Table 13-4. General observations are as follows:

- For copper, the agreement between chemical analyses and analyses calculated from QEMSCAN results was generally good.
- The proportion of copper hosted by chalcopyrite is highest in the final concentrate samples. However, the proportion of copper hosted by chalcopyrite varies widely between ore types, being highest in Judd and K1 ores and lowest in the K2 ore.
- The proportion of copper hosted by bornite is highest in the K2 samples, with 22.7% in the final concentrate, 39.3% in the cleaner tails, and 48.2% in the final tails. Lower but still significant proportions of the copper are hosted by bornite in the K1 and Judd samples.
- A small proportion of the copper is hosted by the secondary sulphides, chalcocite and covellite. The proportion of copper hosted by these minerals is highest in the cleaner tails and tails samples.

These findings are much as expected from the known behaviour of these copper minerals in flotation circuits. Chalcopyrite is strongly floating and efficiently recovered to concentrate. Bornite has variable flotation characteristics and is prone to poor recovery if subject to oxidation or weathering. The secondary sulphide minerals, chalcocite and covellite, can require the use of a sulphurising agent for efficient recovery.

Table 13-4 Copper Deportment in Plant Flotation Products

Copper Deportment									
Mineral Group	K1 Ore			K2 Ore			Judd Ore		
	Final Con	Cleaner Tails	Final Tails	Final Con	Cleaner Tails	Final Tails	Final Con	Cleaner Tails	Final Tails
	Cu mass in product %			Cu mass in product %			Cu mass in product %		
Chalcopyrite	95.4	88.7	87.4	73.0	53.9	44.4	96.8	94.7	94.8
Bornite	2.60	3.48	2.76	22.7	39.3	48.2	0.58	0.47	0.85
Chalcocite/covellite	1.24	6.92	8.37	3.52	5.90	6.68	2.49	4.73	4.24
Tennantite-tetrahedrite	0.26	0.73	1.24	0.05	0.28	0.33	0.03	0.01	0.01
Bi-(Cu,Pb)-S	0.48	0.12	0.00	0.74	0.66	0.44	0.09	0.05	0.04
Fe-(Mn)-oxyhydroxides/carbonates	0.02	0.02	0.19	0.00	0.01	0.01	0.01	0.01	0.04
Copper Accountability									
Cu (QEMSCAN) (%)	10.8	0.49	0.03	27.9	1.54	1.84	5.99	0.41	0.14
Cu (Chemical) (%)	10.2	0.39	0.03	24.6	3.10	1.61	5.07	0.78	0.14

Gangue Mineralogy

Quartz and micas (mainly muscovite) were the dominant silicates detected, followed by minor amounts of feldspars and chlorite. Trace to minor amounts of apatite, Fe-(Mn)-oxyhydroxides / carbonates and fluorite were also present.

Chlorite and feldspars were more abundant in all three Judd products compared with K1 and K2 products.

Most of the fluorine content was associated with muscovite. However, in the K2 final concentrate sample over 20% of the fluorine content occurred in fluorite and andalusite type minerals.

13.3.4 Bulk Mineralogy

Grain Size Distributions of the Various Minerals in the Plant Flotation Products

The main findings for grain size distribution, are summarized below in Table 13-5. The principal observations are as follows:

The 80% passing sizes of pyrite and chalcopyrite are significantly larger in the K1 ore than in the other two ore types.

Chalcocite and covellite grain sizes were very small compared to the other sulphides. However, Table 13-6 (mineral abundance) shows that these two minerals are relatively very minor contributors to the overall copper content of samples.

Compared to K1 ore, the K2 and Judd ores have significantly smaller sulphide particles.

Table 13-5 Particle Sizes of Various Minerals in Plant Flotation Products

Mineral Grain Type	K1 Ore			K2 Ore			Judd Ore		
	Final Con	Cleaner Tails	Final Tails	Final Con	Cleaner Tails	Final Tails	Final Con	Cleaner Tails	Final Tails
	P_{80} µm			P_{80} µm			P_{80} µm		
All Particles	97	49	94	20	65	89	37	60	85
Chalcopyrite	126	35	56	20	49	62	38	43	45
Bornite	103	113	10	15	100	62	22	18	9
Chalcocite/covellite	< 5	< 5	7	< 5	7	29	< 5	< 5	< 5
Combined Cu minerals	126	34	55	19	47	63	38	41	44
Pyrite	96	39	54	20	50	59	31	42	55
Combined sulphides	123	40	54	20	52	64	35	42	56
Combined silicates	26	49	96	21	67	92	37	60	82

Table 13-6 Mineral Abundance in Samples of Plant Flotation Products

Mineral Group	K1 Ore			K2 Ore			Judd Ore		
	Final Con	Cleaner Tails	Final Tails	Final Con	Cleaner Tails	Final Tails	Final Con	Cleaner Tails	Final Tails
	Mass in product %			Mass in product %			Mass in product %		
Pyrite	22.9	13.5	14.2	8.03	16.8	16.0	9.37	9.82	9.88
Chalcopyrite	30.0	1.36	0.09	63.0	5.90	2.54	17.0	2.50	0.38
Bornite	0.45	0.03	0.00	7.91	1.99	1.29	0.06	0.01	0.00
Chalcocite/covellite	0.40	0.12	0.01	2.64	0.53	0.26	0.54	0.16	0.02
Galena	1.00	0.05	0.00	0.07	0.01	0.01	0.42	0.06	0.01
Sphalerite	1.63	0.13	0.06	1.01	0.11	0.04	1.73	0.33	0.11
Other sulphides	0.08	0.02	0.00	0.02	0.04	0.04	0.02	0.01	0.00
Quartz	22.6	42.3	45.8	6.96	35.5	41.1	31.5	40.9	44.4
Muscovite	16.1	33.7	32.1	5.48	26.1	26.4	21.5	25.2	24.5
Feldspars	2.13	4.26	3.83	1.68	6.11	5.66	8.63	10.2	10.2
Chlorite	0.30	1.04	0.94	0.75	2.55	2.37	5.21	6.45	6.28
Andalusite or similar	0.06	0.08	0.04	0.05	0.08	0.06	0.01	0.02	0.02
Apatite	0.31	0.44	0.48	0.19	0.32	0.28	0.45	0.48	0.40
Ti minerals and intergrowths	0.72	1.07	1.07	0.33	0.79	0.78	0.97	1.17	1.16
Fe-(Mn)-oxyhydroxides/carbonates	0.24	0.37	0.31	0.33	0.69	0.65	0.91	1.05	0.92
Calcite	0.02	0.04	0.04	0.10	0.54	0.61	0.21	0.18	0.15
Fluorite	0.00	0.02	0.02	0.01	0.02	0.02	0.01	0.01	0.01

Size Distribution of Flotation Products

Sieve sizing results from the ALS mineralogy are shown below in Table 13-7.

Table 13-7 Size by Size Analysis of Flotation Products

Product	Size fraction µm	Weight %	Au ppm	Cu %	Fe %	S %	SiO2 %	Zn %	F ppm
K1 Final Conc.	+75 µm	31.8	532	19.1	32.6	38.2	5.37	0.79	200
	-75/+20 µm	23.5	868	8.93	19.2	22.4	37.3	1.15	1100
	-20 µm	44.8	320	4.55	8.56	9.34	51.0	0.93	2300
	Combined	100.0	516.0	10.20	18.7	21.6	33.3	0.94	1351
K1 Cleaner Tails	+75 µm	13.42	7.24	0.37	3.88	3.69	74.0	0.06	2200
	-75/+20 µm	30.0	13.2	0.32	7.09	7.36	68.0	0.08	2050
	-20 µm	56.6	32.8	0.44	6.72	6.09	56.9	0.11	3350
	Combined	100.0	23.5	0.39	6.45	6.15	62.5	0.09	2806
K1 Final Tails	+75 µm	29.3	1.39	0.03	2.83	2.50	76.2	0.02	2450
	-75/+20 µm	33.0	2.45	0.04	9.58	10.3	65.0	0.02	2050
	-20 µm	37.7	2.915	0.03	6.85	6.09	57.3	0.02	3600
	Combined	100.0	2.31	0.03	6.57	6.43	65.4	0.02	2752
K2 Final Conc.	+20 µm	17.6	52.4	25.2	24.3	28.3	15.7	0.43	500
	-20 µm	82.4	68.35	24.5	19.6	23.0	18.7	0.54	1400
	Combined	100.0	65.5	24.6	20.4	24	18.2	0.52	1242
K2 Cleaner Tails	+75 µm	17.70	8.21	1.51	5.85	5.26	71.5	0.03	1800
	-75/+20 µm	31.0	11.9	4.39	11.1	11.8	58.1	0.08	1500
	-20 µm	51.3	10.9	2.87	9.28	8.35	51.7	0.06	3050
	Combined	100.0	10.7	3.10	9.24	8.87	57.2	0.06	2349
K2 Final Tails	+75 µm	27.8	5.10	1.05	4.93	4.47	73.5	0.02	1800
	-75/+20 µm	30.9	7.71	2.53	10.5	11.1	61.6	0.04	1450
	-20 µm	41.3	4.51	1.30	8.34	7.27	53.6	0.02	3200
	Combined	100.0	5.66	1.61	8.06	7.68	61.6	0.03	2271
Judd Final Conc.	+75 µm	7.2	NA	2.98	5.71	4.43	68.70	0.15	1550
	-75/+20 µm	28.4	18.7	6.55	10.80	11.10	55.7	0.74	1400
	-20 µm	64.3	30.2	4.65	9.84	8.75	49.1	0.94	2650
	Combined	100.0	26.7	5.07	9.81	9.10	52.4	0.83	2214
Judd Cleaner Tails	+75 µm	16.78	1.30	0.48	3.79	2.43	74.50	0.07	1550
	-75/+20 µm	31.1	4.06	0.94	7.08	6.28	68.3	0.20	1400
	-20 µm	52.1	5.96	0.79	7.15	4.82	56.8	0.21	2550
	Combined	100.0	4.59	0.78	6.56	4.87	63.3	0.18	2025
Judd Final Tails	+75 µm	25.7	0.73	0.12	3.94	2.59	74.30	0.05	1350
	-75/+20 µm	32.8	1.76	0.16	6.71	5.74	70.2	0.07	1200
	-20 µm	41.4	2.03	0.14	6.98	4.27	58.0	0.05	2100
	Combined	100.0	1.61	0.14	6.11	4.32	66.2	0.06	1611

Sizing of Flotation Product Samples

For size distributions of the final concentrates from Table 13-7, the P_{20}, P_{50}, and P_{80} results were obtained by interpolation assuming linear size distributions on Rosin Rammler (RR) axes. Minerals processing size distributions usually give approximately linear plots on RR axes. In the R plot, the Y axis values are (LN-LN(R)) where R is the percent retained at a screen size. The X axis values are (LN(S) where S is the screen size in µm.

Size by Size Analysis of Flotation Products

The results of size-by-size analysis of flotation products from plant sampling for individual ore types is shown below in Table 13-7. The results are interesting in several respects:

In all samples fluorine levels are highest in the finest fraction, and in the Judd samples, the highest fluorine levels are in the final concentrate.

All three cleaner tails samples have greater than 50% by mass in the minus 20 micron fraction, with this fraction having the highest gold content.

A high degree of variation in all analyses can be seen between the three ore types.

All three final concentrate samples are out of marketing specification regarding fluorine content.

Screen Fire Assay/Cyanide Leach Evaluation

Samples of the flotation products were subject to a combined screen fire assay and cyanide soluble gold test. In this procedure, a weighed portion of about 800 grams of sample is ring mill pulverized and then screened at 106 µm. The mass yield to screen oversize was between 0.7% and 2.7% by weight. Coarse metallic gold particles flattened during pulverization reported to the screen oversize that was sent to fire assay. A 500 gram portion of screen undersize was cyanide leached and soluble gold was determined. A 50 gram portion of the washed residue was then subject to fire assay.

This test procedure originated in the gold industry where it can give useful information regarding the bulk mineralogy of a gold ore. The results can be classed as 'Coarse Gold', 'Cyanide Soluble Gold' and 'Refractory Gold'.

When applied to plant products from gold/copper flotation, the test is more qualitative in nature. Nevertheless, the results can be informative. The "Coarse Gold" class consists of large native gold particles whilst the "Cyanide Soluble" class represents well liberated gold minerals. The "Refractory Gold" class represent fine gold minerals that are locked mainly in sulfides but also in silicates.

Table 13-8 Screen Fire Asay Results

Sample	Coarse Gold		Cyanide Soluble Gold		Refractory Gold		Calculated Sample Grade
	g/t	%	g/t	%	g/t	%	g/t
K1 Concentrate	8.24	1.33%	176.57	28.60%	432	70.07%	617.45
K1 Cleaner Tails	0.19	1.06%	15.43	86.88%	2.14	12.06%	17.76
K1 Final Tails	0.11	1.38%	3.73	47.03%	4.09	51.59%	7.93
K2 Concentrate	1.08	1.48%	17.03	23.37%	54.7	75.15%	72.88
K2 Cleaner Tails	0.18	1.83%	4.07	41.88%	5.48	56.29%	9.73
K2 Final Tails	0.33	6.34%	2.50	47.83%	2.40	45.82%	5.23
Judd Concentrate	0.56	1.01%	27.18	49.38%	27.30	49.61%	55.04
Judd Cleaner Tails	0.20	3.51%	4.57	81.33%	0.85	15.16%	5.62
Judd Final Tails	0.07	5.64%	0.87	72.10%	0.27	22.26%	1.21

The most noteworthy features of the results shown in Table 13-8 are the well liberated gold minerals in the cleaner tails samples, and the refractory gold in the final concentrate.

There are significant differences between the three ore types although this could be largely due to ore grade differences. Relatively low levels of "coarse gold" were found in any samples.

The high levels of refractory gold in final concentrates can be interpreted as gold locked in sulfides. It follows that most of the gold recovered to final concentrate is locked in sulfides. The high degree of gold liberation in the cleaner tails probably indicates that liberated gold mineral particles are more difficult to recover by flotation than copper mineral particles. If this is the case then maximising the recovery of coarse grains of copper sulfide, e.g. by flash flotation, will be beneficial to overall gold recovery.

13.4 Comminution Testwork

JKTech SMC tests and traditional Bond Work Index Tests were conducted on all samples. The SMC test results were used for modelling of a range of alternative mill circuits. The Bond Work Index test results were used for check calculations.

Results of the SMC Test results are shown below Table 13-9 whilst parameters derived from the SMC results are shown in Table 13-10. Bond test results are shown in Table 13-11.

Table 13-9 SMC Test Results

Sample	Dwi kWh/m^3	Dwi %	MI Parameters			SG t/m^3
			Mia	Mih	Mic	
Belt Cut Samples						
K1	4.7	25.0	14.0	9.6	5.0	2.80
K2	4.4	23.0	13.7	9.3	4.8	2.76
Judd	4.9	27.0	15.2	10.6	5.5	2.68
Variability Samples (from drill core)						
J1A	3.8	16.0	11.1	7.3	3.8	2.99
J1B	3.9	18.0	11.6	7.7	4.0	2.96
K1A	5.4	34.0	13.0	9.2	4.8	3.42
K1B	5.7	37.0	14.3	10.2	5.3	3.22
KLA	3.2	12.0	9.7	6.2	3.2	3.01
K2A	3.9	17.0	12.1	8.0	4.2	2.82
K2B	4.0	18.0	10.2	6.8	3.5	3.38
K2C	3.7	15.0	10.6	7.0	3.6	3.07

Table 13-10 Parameters Derived from the SMC Results

Sample	A	b	A*b	ta	SCSE kWh/t
Belt Cut Samples					
K1	64.6	0.93	60.0	0.55	8.41
K2	63.8	0.97	61.9	0.58	8.25
Judd	53.8	1.02	54.9	0.53	8.56
Variability Samples (from drill core)					
J1A	54.8	1.44	78.9	0.68	7.74
J1B	58.7	1.29	75.7	0.66	7.84
K1A	72.5	0.87	63.1	0.48	8.74
K1B	63.0	0.90	56.7	0.46	9.18
KLA	70.3	1.34	94.2	0.81	7.23
K2A	59.1	1.22	72.1	0.66	7.82
K2B	61.7	1.39	85.1	0.66	7.63
K2C	62.7	1.33	83.4	0.70	7.64

Table 13-11 Bond Test Results

Sample	CWI kWh/t	RMWI kWh/t	BMWI kWh/t	AI g	CWI (Max) kWh/t	CWI STD Dev	Specific Gravity
Belt Cut Samples							
K1	10.4	16.7	16.33	0.1269	26.6	5.8	3.04
K2	17.7	17.7	15.93	0.1284	51.4	10.2	2.96
Judd	13.2	17.3	17.04	0.1305	31	6.6	2.87
Variability Samples (from drill core)							
J1A			15.91				
J1B			16.43				
K1A			14.23				
K1B			15.37				
KLA			13.57				
K2A			15.35				
K2B			15.94				
K2C			14.97				

13.4.1 Bead Milling, Thickening, Filtration and Rheology Testwork on a Plant Sample of Flotation Concentrate

Overview and Rationale of the Testwork

A 20 kg portion of final concentrate was obtained from the K92ML plant in November 2021. The concentrate was sent to Keramos Laboratory where a small portion was used investigate the grinding energy requirements to achieve a range of product sizes. The remainder was split into two equal samples and one of these was bead milled to simulate final concentrate produced from a flotation circuit incorporating rougher concentrate regrinding.

The two samples of concentrate ("as received" and "bead milled") then sent to the MOG laboratory for thickener and pressure filtration testwork. When this had been completed, the two samples were sent to ALS laboratory in Perth for rheology tests.

Bead Milling Testwork

Overview

The Keramos laboratory first determined the particle size distribution of the concentrate using both sieve sizing and laser sizing. The Metso 'Signature Plot' procedure was then used to evaluate the grinding energy required to achieve a range of concentrate sizing. The test uses a laboratory scale Metso Stirred Media Detritor (SMD) to determine sizing parameters (Signature Plots) for SMD mills. This is a batch test requiring 1.5 kg of sample.

The remaining sample was then split into two portions. The Keramos lab used the laboratory scale SMD in continuous feed mode to grind one portion of sample to a P_{80} of approximately 30 µm. The size distribution of the ground sample was measured by a laser particle sizer.

Evaluation of Different Grinding Bead Sizes

The Keramos lab did Signature Plot testwork with two different grinding bead top sizes, 5 mm and 3 mm. Keramos recommended the use of 5 mm beads for a K92ML concentrate regrind application. The 3 mm bead top size was less able than the 5 mm top size to break the coarse particles in the concentrate.

The Relationship Between Specific Energy and Grind Size

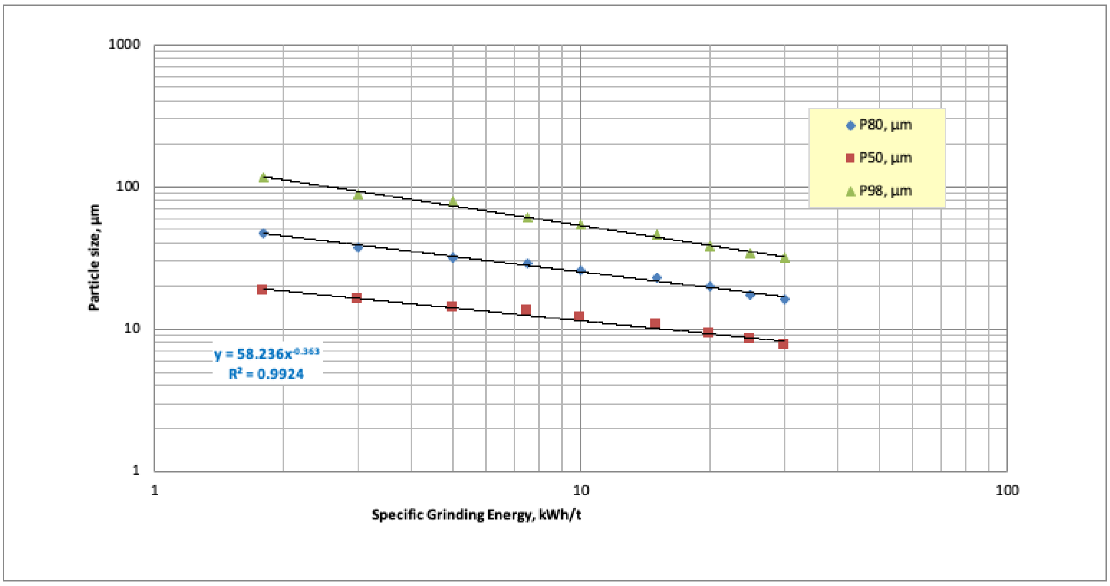
Results of the Signature Plot are presented in Figure 13-1, which shows data for grind sizes produced by a range of specific grinding energy input. The calculation shown in Figure 13-1 indicates that a Specific Grinding Energy of 6.2 kWh/t is required to achieve a grind P_{80} of 30 µm.

Figure 13-1 Metso SMD Signature Plot Results Showing Grind Size Data at a Range of Specific Energy

SMD Signature Plot - Keramos UFG Grinding Laboratory - Perth	
Media Product	MetPro AX280
Media SG (g/cm^3)	2.80
Media Charge Top-Size (mm)	5.0
Feed Sample	K92

P(80) Target	30	µm	
Specific Grinding Energy	6.20	kWh/t	
=LINEST(LN(Y_2),LN(X_2))	-0.363	4.065	58.236

SGE, kWh/t	1.8	3.0	5.0	7.5	10	15	20	25	30
P50, µm	18.46	16.06	13.97	13.29	11.89	10.68	9.28	8.34	7.71
P80, µm	47.60	37.32	32.01	28.82	25.67	22.81	20.04	17.55	16.27
P98, µm	116.90	88.08	78.87	60.32	54.59	45.72	38.50	33.99	31.77
CSI	2.46	2.36	2.46	2.09	2.13	2.00	1.92	1.94	1.95



13.4.2 Concentrate Thickening Testwork

Recommended Concentrate Thickener Design

From the testwork results, a high-rate thickener with a Vane feedwell was recommended. Design parameters for the thickener were as shown below in Table 13-12.

Table 13-12 Recommended Thickener Design Parameters

Process Stream	Unground Concentrate P_{80} 76 µm	As Received Concentrate P_{80} 34 µm
Solids Feed Rate (t/h)	18	18
Solids Loading (t/(m^2*h))	0.25	0.25
Feed slurry density (% solids (w/w))	30	30
Diluted feedwell slurry density (% solids (w/w))	18	18
Slurry pH	11	11
Flocculant Dosage (g/t)	10	20
Flocculant Type	BASF Magnafloc 155	BASF Magnafloc 155
Underflow Density (% solids (w/w))	69.5	63.8
Yield Stress (Pa)	77	65
Overflow Clarity (mg/L)	<100	<100
Required Thickener Diameter (m)	10	10

Thickening Testwork

Concentrate thickening testwork was performed by the MOG laboratory in Perth. The objectives were to establish design parameters for thickening the "bead milled" and "as received" concentrate samples to a high underflow density suitable for feed to pressure filtration.

The test work showed that both samples can be successfully thickened to high densities. In dynamic thickening tests, with 0.25 t/(m^2*h) solids loading, the as received concentrate sample gave underflow densities of 66.7% to 69.5% solids (w/w) with yield stresses ranging from 39 Pa to 77 Pa. At the same solids loading, the bead milled concentrate sample gave underflow densities of 61.8% to 63.8% solids (w/w) with yield stresses ranging from 38 Pa to 69 Pa.

Plant Operating Results

The maximum quarterly concentrate exported was 6,707 dry tons in Q3, 2023. This corresponds to about 26,828 tons per year, or an average hourly rate of about 3.35 tons per hour, based on 8,000 hours per year running time.

The existing plant has a 4 m diameter concentrate thickener with a settling area of 12.56 m^2. The average hourly feed rate corresponds to a thickener duty of 0.267 tons per square meter per hour. The new plant was designed for a duty of 0.25 tons per square meter per hour.

13.4.3 Pressure Filtration Testwork

Overview

Pressure filtration testwork was performed by the MOG laboratory in Perth on thickened samples from the preceding thickener tests. The results were used to establish design parameters for filtering 18 t/h of concentrate and producing filter cake with a maximum of 10% moisture content. It was assumed that the equipment used would be of the same type as in the existing plant: a vertical plate membrane filter press with air blow drying.

Recommended Pressure Filtration Design Basis

From the testwork results, MOG recommended a Larox Pressure Filter with 33 mm chambers, diaphragm cake pressing and air-blow cake drying.

Table 13-13 Recommended Pressure Filtration Parameters

Filter Feed	As Received Concentrate	Bead Milled Concentrate
Feed Slurry Density (% solids (w/w))	68	64
Filtration Rate (kg of dry solids per m^2 per hour)	390	392
Cake Moisture (% moisture (w/w))	10.5	13
Air Drying	High Pressure	High Pressure
Chamber Thickness (mm)	33	33
Cake Thickness (mm)	24	25

The feed pressure to the filter, the pressure for diaphragm pressing and the quantity and pressure of air required for cake drying were not specified in the MOG report.

Outline Details of the Test Program

A chamber thickness of 45 mm was initially tried with the as received concentrate. However, it was clear from the results that the target moisture content of 10% would not be achieved. All further testwork on both samples utilized plates with a 33 mm chamber.

Moisture content versus cycle time curves for the unground (as received) and ground (bead milled) samples are shown below in Figure 13-2 and Figure 13-3 respectively. These graphs have been copied from the MOG report.

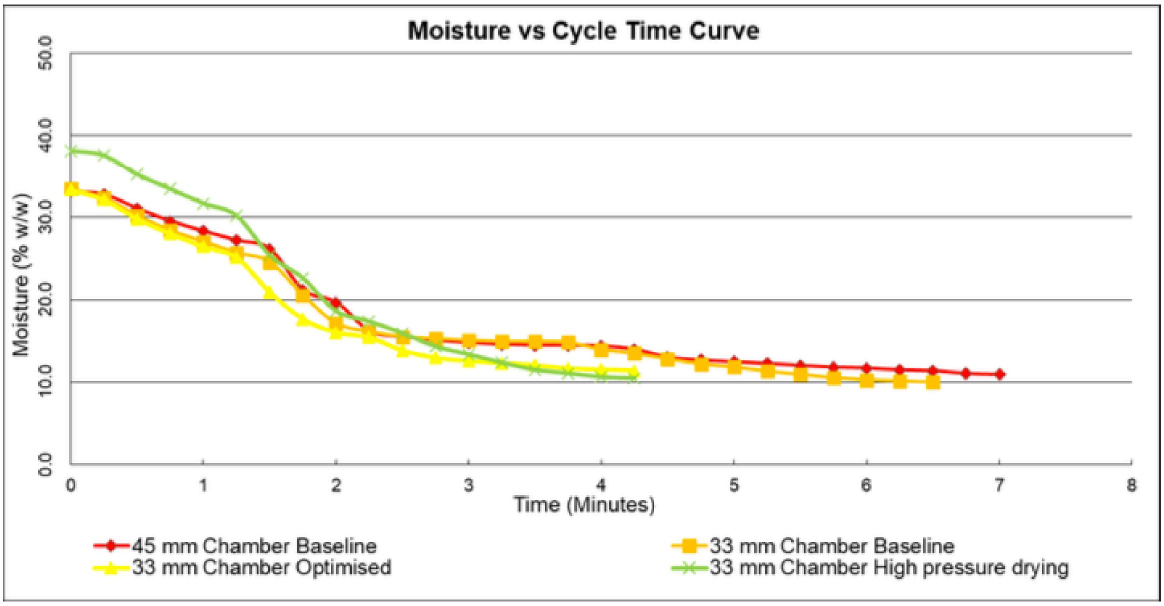


Figure 13-2 Filtration Results on the Unground Sample

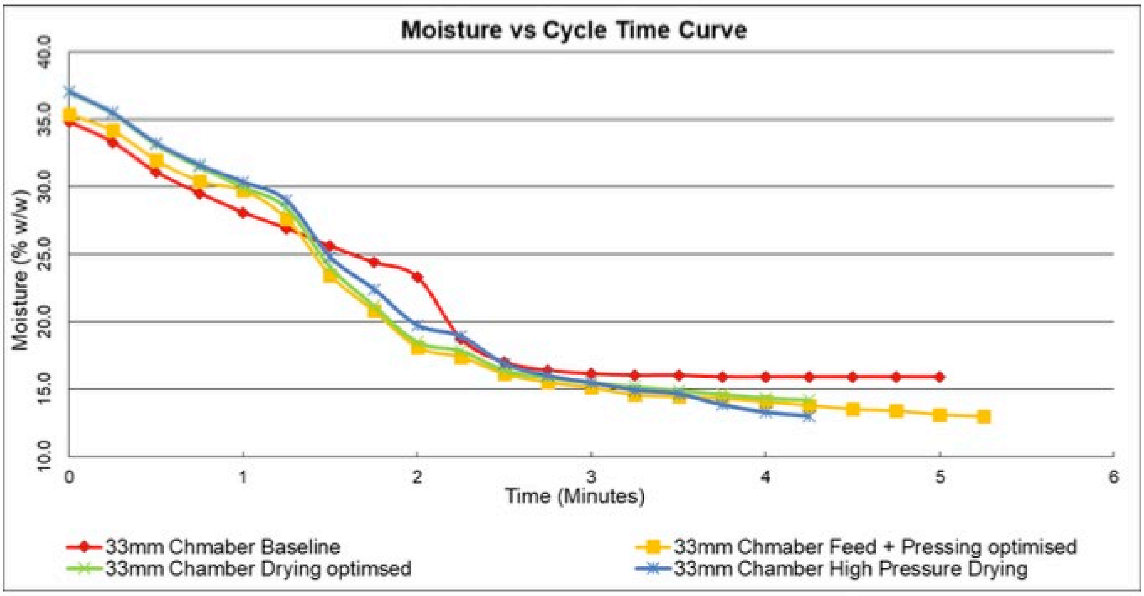


Figure 13-3 Filtration Results on the Ground Sample

The advantages of high-pressure drying air can be seen in both cases.

Plant Operating Results

The original filter press in the existing plant is an Andritz vertical plate diaphragm plate unit with air blow dry of concentrate. The plates have dimensions of 1.2 meters square with 45 mm cavity thickness. There were 16 plates (so 15 cavities), a total filtration are of 35 m^2, and a filter chamber volume of 800 litres.

A second filter press was installed in 2020 as part of the Stage 2 Plant Expansion but never brought into routine operation. However, in 2022, an additional five plates were added to the original Andritz filter press. With 21 plates (20 cavities), it now has a total filtration area of 46.7 m^2, and a chamber volume of 1,067 litres.

The maximum quarterly concentrate exported was 6,707 dry tons in Q3, 2023. This corresponds to about 26,828 tons per year, or an average hourly rate of about 3.35 tons per hour, based on 8,000 hours per year running time. For a filtration area of 46.7 m^2, this corresponds to a filtration rate of about 72 kg per square metre per hour.

Concentrate Filter in 1.2M tpa Plant

The new plant has a Jord Filter press with 51 Plates (50 chambers), and 115 m^2 of filter area. The plates have dimensions of 1.2 meters square with 35 mm cavity thickness. The total chamber volume is 1,550 litres. The design criteria is for a maximum capacity of 11.2 dry tons per hour at 100% utilisation.

In practice, filtration capacity is likely to be proportional to filter area. Scaling from the existing plant, at 72 kg per square meter per hour, with a filtration area of 115 m^2, the capacity will be 8.26 tons per hour. Over an 8,000 hour year, this would correspond to about 66,000 dry tons of concentrate.

13.4.4 Rheology Testwork at ALS Laboratory Perth

Overview

Rheology testing at ALS Metallurgy Services (ALS), Perth was conducted under the direction of Lycopodium Engineers, Brisbane. The testwork included the Thixotropy Curve and Vane Yield Stress rheology tests. Objectives were to determine the maximum slurry density for pumping concentrate slurries using conventional centrifugal pumps.

Rheology measurements over a range of slurry densities indicated that both samples had a yield stress point, consistent with the Bingham fluid model. Shearing of the pulp before testing resulted in a significant yield stress reduction for the bead milled sample but not for the as received sample. Plots of yield stress versus slurry density are presented in the figures below.

In Figure 13-4. ALS Results for the As Received Sample, Measured Yield Stress versus Solids Concentration, for the as received sample, the measured yield stress increases rapidly beyond about 76% solids (w/w). In Figure 13-5. ALS Results for the Ground Sample, Measured Yield Stress versus Solids Concentration, for the ground sample, the measured yield stress increases rapidly beyond about 62% solids (w/w). In practice, centrifugal pumping of concentrate slurry would be possible up to a slurry density just below where the yield stress increases rapidly. From the graphs, this point could be taken as 74% solids for the as received sample and 58% solids for the bead milled sample.

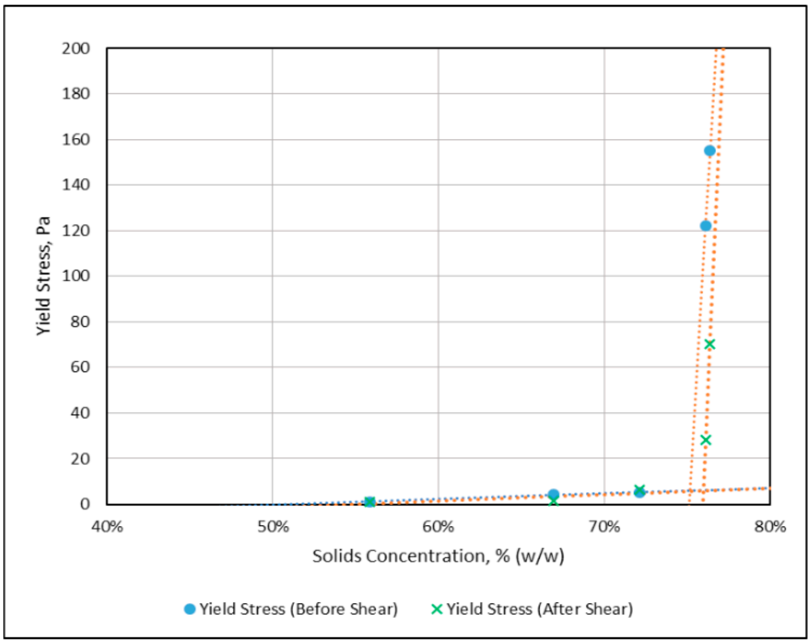


Figure 13-4 ALS Results for the As Received Sample, Measured Yield Stress versus Solids Concentration

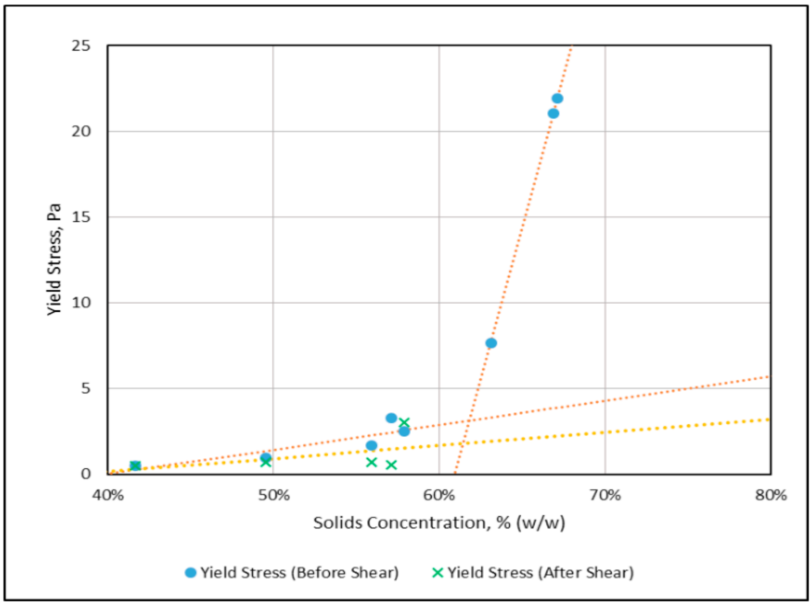


Figure 13-5 ALS Results for the Ground Sample, Measured Yield Stress versus Solids Concentration

13.5 Sample Preparation for Gravity Recoverable Gold and Flotation Testwork

The K1 and Judd ore samples were delivered to Nagrom Laboratory during March 2021 whilst the K2 sample was delivered in May 2021. Comminution testwork was completed and samples were recombined in preparation for float tests. The Bulk composite sample was made up from 40 kg of K1 ore and 40 kg K2 ore. Grind calibration runs were completed on all four samples in preparation for recovery testwork.

13.6 Gravity Recoverable Gold Tests on the Three Individual Ore Type Samples

13.6.1 Overview

Results of the GRG tests are summarised in Table 13-14. The results for K1 ore are quite different to those for K2 and Judd. The gravity recoverable gold is much finer, and the recovery is significantly lower. The size class shown is based on the Amira project P420B.

Table 13-14 Results of GRG Tests

	K1	K2	Judd
GRG Recovery %	38%	58%	62%
Size Class	Fine	V Coarse	Coarse
P_{80} (µm)	106	260	227
P_{50} (µm)	62	150	91
P_{20} (µm)	36	60	45

13.6.2 Processing Considerations

Portions of the three ore types were subject to a standard three-stage Laplante type test procedure as described in reference: (Evolution and Optimization of the Gravity Recoverable Gold Test, Liming Huang, and Sunil Koppalkar, 39th Annual Meeting of the Canadian Mineral Processors, January 23 to 25, 2007 Ottawa, Ontario, Canada). Results of these standard tests can be used to predict the gravity gold recovery in a full-scale plant with various configurations of gravity gold recovery circuit.

Recovery of liberated gold particles based on particle density is commonly referred to as 'gravity concentration', with recovered gold being termed gravity recoverable gold (GRG). In the existing K92ML processing plant, a portion of the mill discharge goes to a centrifugal gravity concentrator and a portion of the cyclone underflow goes to a flash flotation cell. Both the flash float cell and the centrifugal concentrator will recover GRG. Centrifugal concentrators can efficiently recover gold particles that are too large for flash flotation recovery. To maximize gold recovery over a wide particle size range, it is not unusual to have both a flash float cell and a centrifugal concentrator in the mill circuit.

The capability of the existing K92ML flash flotation cell to recover gold particles of various sizes has not been evaluated. However, testwork on other installations has shown excellent recoveries of gold particles up to a size of about 100 microns. Recovery progressively decreases for larger gold particles, and the maximum recoverable size is between 300 µm and 500 µm. This work is described in the technical paper: A Decade of Gravity Gold Recovery, G Wardell-Johnson, A Bax, W P Staunton, J McGrath and J Eksteen, World Gold Conference / Brisbane, Qld, 26 - 29 September 2013.

Conventional float cells are less capable than flash flotation for coarse particle recovery. Gold recovery in copper gold flotation is discussed in the technical paper: Size dependent gold deportment in the products of copper flotation and methods to increase gold recovery, G. Small, A. Michelmore, and S. Grano, The Journal of The South African Institute of Mining and Metallurgy, November 2003. The paper notes that gold recovery in conventional rougher flotation cells decreases rapidly at particle sizes greater than 75 µm.

13.6.3 Details of the Gravity Gold Recovery Testwork

The tests used a laboratory size Knelson centrifugal concentrator (three inch diameter) with rotational speed set to generate 60 times gravitational force. The conditions shown in Table 13-15 were used in sequential passes through the Knelson.

The samples were stage crushed to 99% passing 850 µm and the plus 850 µm was separated and sent for gold analysis. The minus 850 µm fraction was fed through the Knelson and the first pass concentrate was collected. The tailings were then ground to 50% passing 75 µm before the second pass through the Knelson. Second stage tailings were then reground to 80% passing 75 µm before the third pass through the Knelson. The three stage concentrates were then subject to size-by-size analysis for gold.

Table 13-15 Summary of Knelson Operating Parameters for the Three-Stage Gravity Recoverable Gold Tests

Target Parameter	Primary Pass	Secondary Pass	Tertiary Pass
Feed Slurry Density (% solids (w/w))	40	40	40
Target Grind Size (µm)	P_{99} 850	P_{50} 75	P_{80} 75
Target Feed Rate (kg/min)	0.80	0.50	0.30
Fluidization Water Pressure (kPa)	21 - 34	21 - 34	14 - 20

13.6.4 Results of the Three Stage Gravity Gold Recovery Tests

Mass yields and gold recoveries to the various products from the GRG tests are shown below in Table 13-16. The gold analysis of the +850 µm size fraction (approximately 1% by weight and removed before the first pass through the Knelson) gives first indication of the presence of coarse gold particles. The relative gold recoveries in the primary, secondary and tertiary passes through the Knelson correspond to the gravity gold liberated at the three grinds. The total gold recovery to concentrates approximates to the maximum recovery of GRG when the ore is processed in a full-scale plant with an optimum gravity concentration circuit.

The results of the size-by-size analyses of the Knelson concentrates are shown below in Table 13-17. Gold distributions by size for the K1, K2 and Judd ores are shown below in Figure 13-6, Figure 13-7 and Figure 13-8 respectively. Size-by-size distribution of gold in the cumulative concentrates from the three stages (primary, secondary and tertiary) of recovery are shown. The graphs show the cumulative gold recovery versus particle size for the following products:

- Primary Concentrate (feed grind P_{95} 850 µm).
- Primary and Secondary Concentrate (feed grind P_{50} 75 µm).
- Primary, Secondary and Tertiary Concentrate (feed grind P_{80} 75 µm).

The material passing the 25 µm screen is shown as having 12 µm size in the graphs.

Table 13-16 Summarized Results of Gravity Gold Tests

Sample			Recovery	
	Mass kg	Gold Analysis ppm	Mass Yield %	Gold Recovery %
K1				
Assay Head		7.582		
Calc. Head	141.000	7.833	100.00	100.00
+850 µm Size Fraction	1.537	2.368	1.09	0.33
Primary Concentrate	0.516	340.2	0.37	15.91
Secondary Concentrate	1.957	110.5	1.39	19.58
Tertiary Concentrate	3.714	39.30	2.63	13.21
Tertiary Tailing	133.275	4.224	94.52	50.97
K2				
Assay Head		28.43		
Calc. Head	95.480	7.468	100.00	100.00
+850 µm Size Fraction	0.971	132.3	1.02	18.02
Primary Concentrate	0.350	361.2	0.37	17.70
Secondary Concentrate	1.326	156.8	1.39	29.17
Tertiary Concentrate	2.438	32.71	2.55	11.18
Tertiary Tailing	90.394	1.888	94.67	23.93
Judd				
Assay Head		3.701		
Calc. Head	134.000	3.950	100.00	100.00
+850 µm Size Fraction	1.075	0.614	0.80	0.12
Primary Concentrate	0.501	247.2	0.37	23.39
Secondary Concentrate	1.818	74.35	1.36	25.54
Tertiary Concentrate	3.463	25.88	2.58	16.93
Tertiary Tailing	127.142	1.416	94.88	34.01

Table 13-17 Size by Size Gold Distribution Results

Sieve Size µm	Cumulative Gold Retained (% Recovery from Head Sample)								
	K1			K2			Judd		
	Primary P_{99} 850 µm	Primary and Secondary P_{50} 75 µm	Primary, Secondary and Tertiary P_{80} 75 µm	Primary P_{99} 850 µm	Primary and Secondary P_{50} 75 µm	Primary, Secondary and Tertiary P_{80} 75 µm	Primary P_{99} 850 µm	Primary and Secondary P_{50} 75 µm	Primary, Secondary and Tertiary P_{80} 75 µm
850	0.0%	0.0%	0.0%	0.1%	0.1%	0.1%	0.0%	0.0%	0.0%
500	0.1%	0.1%	0.1%	5.1%	5.1%	5.1%	2.9%	3.1%	3.1%
300	0.3%	0.4%	0.4%	7.2%	8.8%	8.8%	4.0%	7.4%	7.4%
212	0.8%	0.9%	0.9%	9.2%	18.5%	18.6%	7.1%	14.0%	14.0%
150	1.9%	3.2%	3.3%	11.2%	29.1%	29.2%	9.3%	20.4%	20.9%
106	4.1%	7.4%	7.5%	12.8%	34.6%	35.3%	11.7%	25.5%	27.4%
75	7.4%	13.5%	14.0%	14.3%	38.4%	42.1%	14.8%	31.5%	37.2%
45	13.3%	25.2%	26.4%	16.5%	43.3%	51.4%	19.9%	40.7%	50.3%
25	15.6%	33.9%	36.3%	17.5%	46.4%	57.3%	22.8%	47.4%	60.3%
0	15.9%	35.5%	38.2%	17.7%	46.9%	58.3%	23.4%	48.9%	62.3%

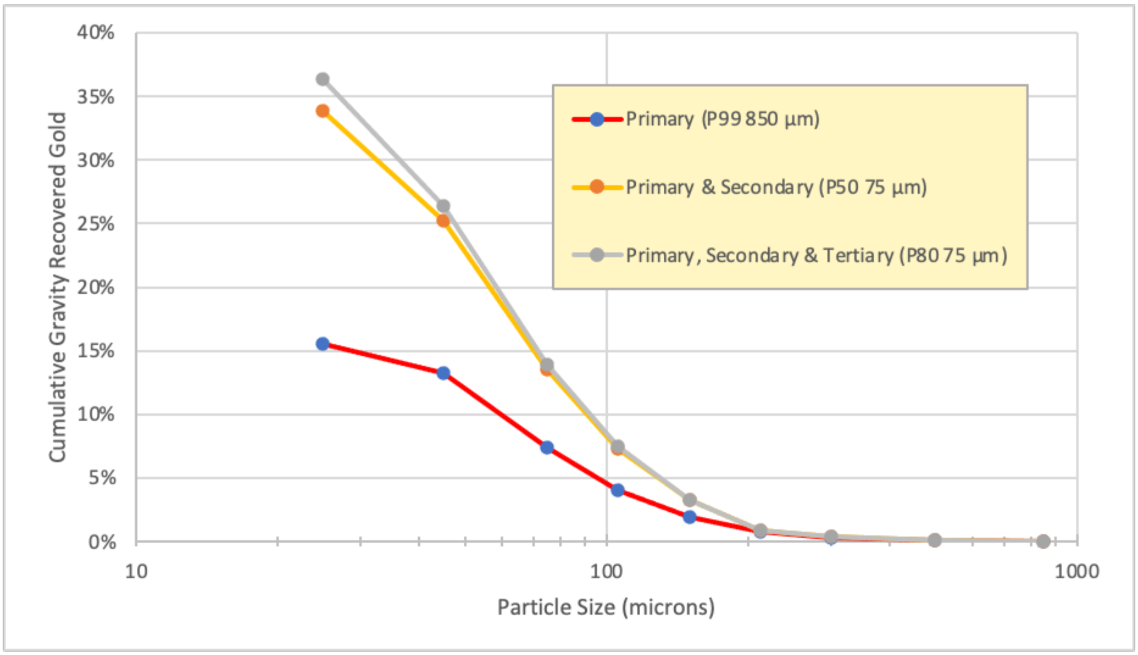


Figure 13-6 K1 Composite: Size Distribution of Gravity Recoverable Gold

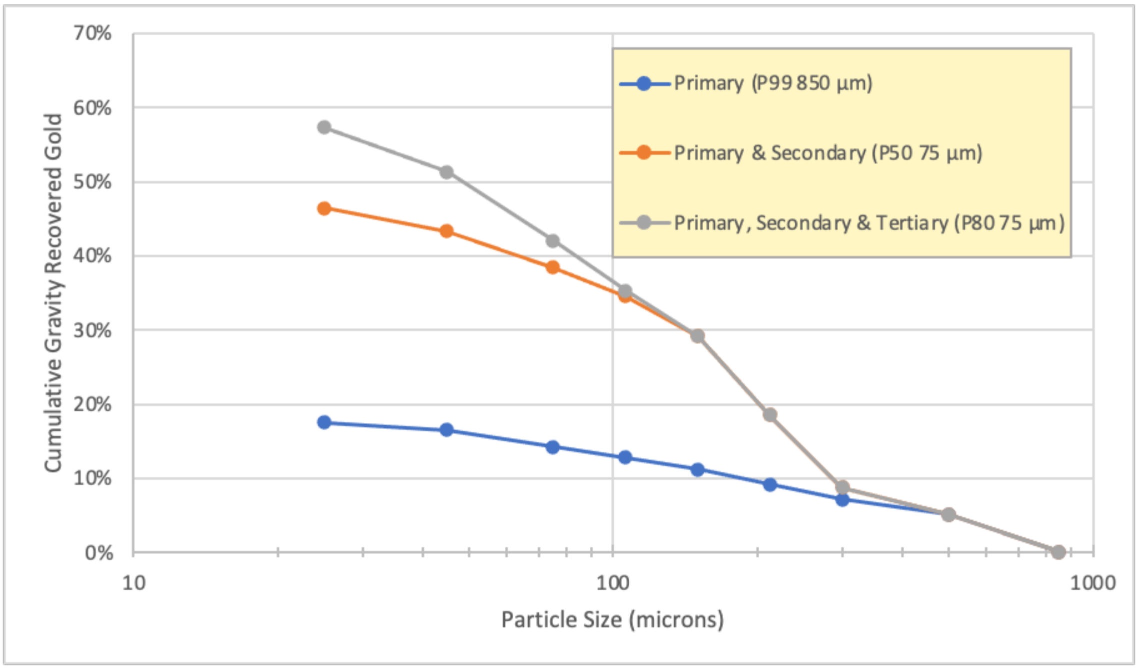


Figure 13-7 K2 Composite: Size Distribution of Gravity Recoverable Gold

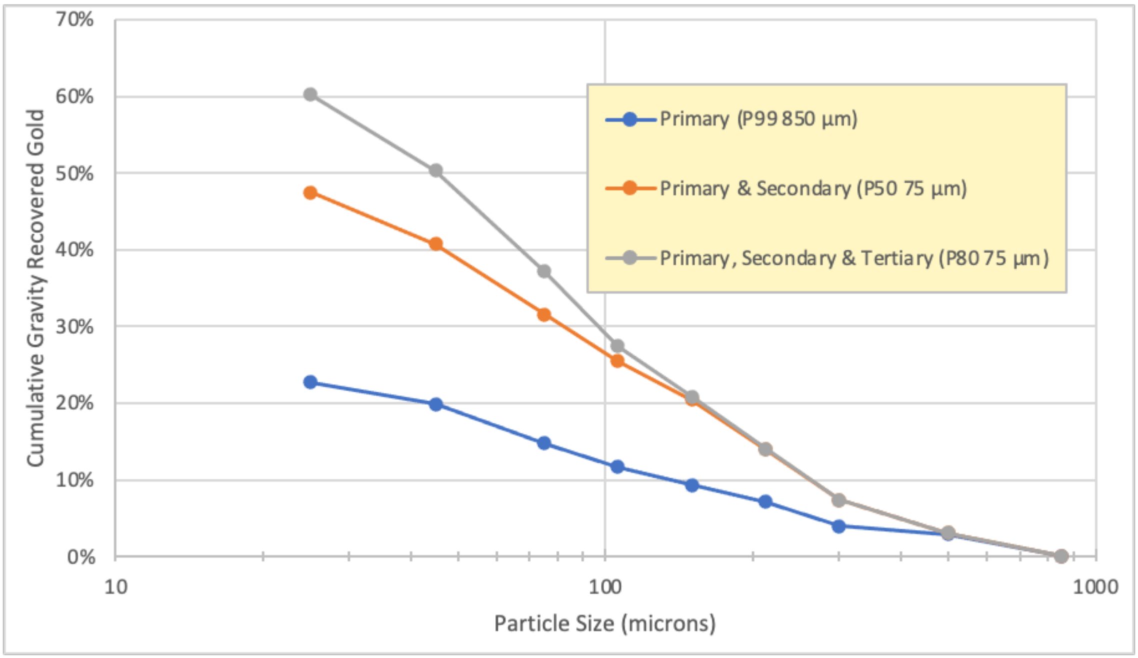


Figure 13-8 Judd Composite: Size Distribution of Gravity Recoverable Gold

13.6.5 Differences in Response from K1, K2 and Judd

For the K2 ore, about 18% of the gold content reported to the +850 µm size fraction that was screened out ahead of Knelson recovery. For the K1 and Judd ores well under 1% of the gold reported to this size fraction.

The response of the K1 ore to centrifugal concentration is quite different to K2 and Judd. For K1, 35% of the gold content was recovered to Knelson concentrate, and most of the gold was recovered in the second pass. For K1, about 7% of the gold was recovered to the +75 µm size fraction of the concentrate. K2 and Judd are similar with about 60% total recovered gold and with 42% and 37% respectively recovered to the +75 µm particle size fraction of the combined (primary, secondary and tertiary) Knelson concentrates.

The results can be interpreted as follows:

- For K1 ore, there is no GRG that is too coarse for flash flotation recovery.
- For K2 and Judd ores, some GRG is too coarse for flash flotation recovery.
- For K2 ore, this is the sum of the 10% of the gold content with a particle size greater than 300 µm plus the 18% of the gold content that reported to the plus 850 µm size fraction.
- For Judd, this is the 8% of the gold content with a particle size greater than 300 µm.
- For the K2 and Judd ores, maximising GRG recovery will require the installation of a centrifugal concentrator as well as a flash flotation cell.
- All the ore types will benefit from flash flotation recovery of coarse gold particles.

13.7 Batch Flotation Tests on the Three Individual Ore Types and the Master Composite

13.7.1 Results and Recommendations

Recommended Collector Reagent

Blend 1 Collector at an addition rate of 100 g/t gave the best overall flotation performance and was recommended for the new plant design.

Blend 1 - Aero3477, Aero6697 and Aero5100 with a mass ratio of 64:16:20 dosed at 100 g/t, pH 11.

Recommended Grind

A grind P_{80} of 106 µm gave the best gold and copper recoveries and was recommended for the new plant design.

Grind P_{80}'s of 75, 106 and 150 µm were evaluated in batch flotation tests on portions of K1, K2, Judd, and the Master Composite. The best overall performance including consideration of concentrate grade and gangue rejection was achieved at the finest grind. However, the mid grind was recommended for maximum recoveries

Recommendation Regarding Rougher Concentrate Regrinding

It was recommended that the new plant be designed with space provision for the retrofit of rougher concentrate regrinding.

The comparison of results of batch and locked cycle tests with and without rougher concentrate regrind was inconclusive. In the locked cycle tests, regrinding gave improved gangue rejection and higher concentrate grades but at the expense of gold and copper recoveries. Comparison of the results in terms of Recovery versus Concentration Ratio did not clearly demonstrate either flowsheet to be superior to the other.

Recommendation Regarding Selective or Bulk Rougher Flotation.

Selective rougher and cleaner flotation was recommended for the new plant.

Selective flotation gave better results than bulk roughing followed by selective cleaning.

13.7.2 Testwork Overview

A total of 28 batch rougher tests and 16 rougher-cleaner recleaner tests were conducted. This was followed by two locked cycle tests to investigate the flowsheet alternatives of closed circuit first cleaner tails with no concentrate regrind, and open circuit cleaner tails with rougher concentrate regrind.

Batch flotation Tests were conducted on the three individual ore composites and the Master Composite. The objectives of the testwork were as follows:

- To evaluate three alternative reagent suites in comparison with the reagents used in the current plant.
- To determine the best combination of reagent selection and dosage rate for selective rougher flotation.
- To evaluate three different primary grinds.
- To evaluate possible benefits of regrinding the rougher concentrate before cleaner flotation.
- To evaluate the alternative flowsheet concept of bulk rougher flotation followed by selective cleaner flotation.

Testwork Procedures

The work started with grind establishment to determine the laboratory rod milling time required to achieve a range of specified grind 80% passing sizes (P_{80}). A 2.5 litre laboratory float cell, with a rotor speed of 800 rpm was used for tests on one-kilogram portions of sample.

The following tests were performed on the four composite samples:

- Rougher tests at grind size P_{80} of 75 µm using the same reagent suite as the operating plant, with four stages of reagent addition and collection, to confirm required dosage rates and flotation times (total four batch tests).
- Rougher tests at a grind size P_{80} of 75 µm, with four stages of reagent addition and collection, to evaluate three alternative collector blends (total 12 batch tests).
- Rougher tests, using the selected collector and selected addition rate, at grind size P_{80}'s of 53 µm, 106 µm, and 150 µm to investigate the effect of primary grind (total 12 batch tests).
- Rougher, cleaner recleaner tests using the selected collector and primary grind to investigate three levels of concentrate regrind (12 rougher cleaner recleaner tests).

- A rougher cleaner, recleaner test incorporating bulk sulphide roughing followed by selective cleaning (four rougher cleaner recleaner tests).

This batch testwork was then followed up by locked cycle flotation tests on the master composite.

13.7.3 Batch Rougher Tests to Evaluate Collector Dosage Requirements and Compare Alternative Collectors with Current Plant Practice

Existing Plant Reagent Scheme

Prior to commencing testwork the average reagent additions during plant operations (period 07.09.2020 to 12.12.2020) were:

- Collector Aero 6697 at dosage rate 72 g/t of ore.
- Co-collector PAX at dosage rate 5 g/t of ore.
- An average rougher feed pH 11.2 by adding about 2 kg/t of lime.
- No frother or gangue depressant was used during this period. Discussions with process management concluded that a maximum combined collector addition of 100 g/t and pH 12 should be considered as the base reagent suite for testwork.

Selection of Alternative Collector Schemes

The reagent scheme used by the existing plant seems quite simple when compared with regimes used on other concentrators where gold recovery is the priority. Advice was sought from an independent reagent supplier with extensive experience in the supply of reagents to copper / gold and gold flotation plants.

The objectives of the alternative reagent selections were:

- To achieve higher recoveries of gold - by broadening the collector action.
- To achieve improved selectivity against pyrite by using a more selective collector at pH 11.
- To determine the minimum collector dosage required for acceptable recoveries of gold and copper.

Details of the various collector chemical used (alone or in blends) in the comparative tests are as follows:

- Cytec Aero 6697: sodium di-isobutyl mono-thiophosphate (SDIBMTP).
- PAX: potassium amyl xanthate.
- Aero 3477: Sodium di-isobutyl dithiophosphate (SDIBDTP).
- Aero 5100: Carbamothioic acid, N-2-propen-1-yl-, O-(2-methylpropyl) ester (CAS # 86329-09-1).

Batch Rougher Flotation Tests to Evaluate Alternative Collectors

The four comparative tests used the following total collector additions:

1. As used currently on the plant: PAX 6 g/t, Cytec Aero 6697 94 g/t, pH 12.
2. A simple alternative capable of selective flotation at lower pH: Aero 5100 dosed at 100 g/t, pH11.
3. Blend 1 - Aero3477, Aero6697 and Aero5100 with a mass ratio of 64:16:20 dosed at 100 g/t, pH 11.
4. Blend 2 made up of Aero3477 and Aero6697 with a mass ratio of 80:20 dosed at 94 g/t, plus 6 g/t PAX, pH11.

In each test, the total collector addition of 100 g/t was added in four increments of 25 g/t. Four corresponding increments of concentrate, plus the final tailings, were sent for analysis.

Standard Rougher Flotation Test Procedure

The rougher flotation test procedure consisted of the following steps:

1. Conditioning for 6 minutes with lime addition to selected pH.

2. Conditioning for a further 2 minutes with 25% of PAX addition (only in cases where PAX was used).
3. Conditioning for 2 minutes with 25% of collector addition, and MIBC frother added as necessary to provide a stable froth.
4. First stage rougher flotation for 5 minutes (this was very close to flotation to extinction, i.e. until no more mineralized froth was formed).
5. Addition of a further 25% of the collector addition and conditioning for 2 minutes.
6. Second stage rougher flotation for 4 or 5 minutes (effectively to near extinction).
7. Addition of a further 25% of the collector addition and conditioning for 2 minutes.
8. Third stage rougher flotation for 4 minutes (effectively to near extinction).
9. Addition of the final 25% of the collector addition and conditioning for 2 minutes.
10. Third stage rougher flotation for 4 minutes (effectively to near extinction).

In practice, in all tests only one drop of MIBC frother was added during the first conditioning. Furthermore, Aero 5100 was the only collector that required 5 minutes of flotation in second stage roughing. All other collectors required only 4 minutes of flotation. The slaked lime addition to reach pH 11 was about 1.0 kg/t whilst about 1.8 kg/t was required for pH 12. Perth tap water was used in all flotation tests.

This test procedure gives information regarding the relationship between collector addition, value metals recovery, concentrate grade and concentrate yield. It also gives some indication regarding the flotation kinetics.

Results of the Collector Comparison Tests

Analysing the results of the rougher tests to evaluate the alternative collector blends is complicated by:

- The widely differing gold and copper head-grades of the (K1, K2, Judd and Master Composite) samples.
- The differences in mineralogy between the samples and inherent differences in flotation response.

The results of all 16 tests are shown plotted on gold recovery (%) versus concentrate grade (Au g/t) axes in Figure 13-9. The results are coded by symbol for the four ore types and by colour for the 4 collectors. The lines on the graph represent cumulative gold recovery plotted against cumulative concentrate grade. Performance may be judged by comparing both recovery and concentrate grade.

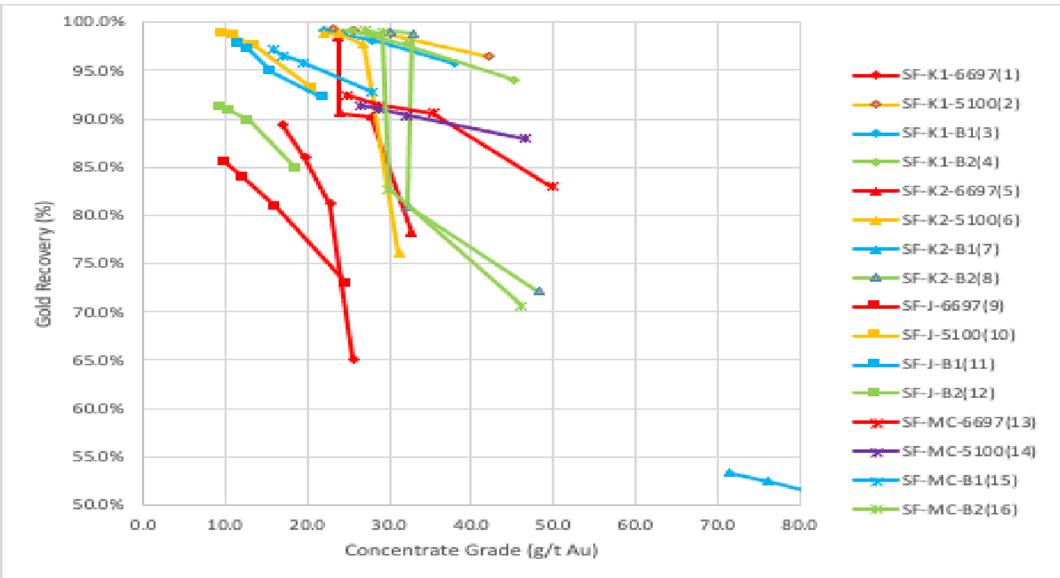


Figure 13-9 Results of the Batch Rougher Tests to Evaluate Alternative Collectors Gold Recovery versus Concentrate Grade

As expected, results are scattered, and there is no clear winner between the four collector suites. However, it does seem possible that one of the three alternative suites may offer improved grade recovery performance compared with the currently used collectors.

A more broadly based comparison was made by ranking of each increment of rougher flotation based on:

- Mass yield to concentrate - low mass yield and selective flotation are desirable.
- Gold recovery.
- Copper grade of rougher concentrate.
- Copper recovery to concentrate.
- Rejection of Al2O3 containing gangue and by inference fluorine containing minerals.

On each of the above criteria the performance of each rougher float increment was ranked 4 points for best performance ranging down to 1 point for the worst. Results are summarized below in Table 13-18.

Table 13-18 Comparison of Collectors - Total Performance Points

Collector Type	6697	5100	Blend 1	Blend 2
Performance Points	179	193	220	204

13.7.4 Batch Rougher Tests to Evaluate the Effect of Grind P_{80}

Collector Blend 1 was selected for the next stage of batch rougher flotation tests to evaluate the effects of grind P_{80}.

Rougher Flotation Test Procedure to Evaluate Primary Grind

A simplified test procedure was used to investigate rougher flotation at three different primary grinds. Based on the staged recoveries achieved in the collector comparison a total reagent dosage of 50 g/t of Blend 1, added in 4 equal increments, was selected. Each tests run commenced with conditioning and lime addition to pH 11. After 2 minutes of lime conditioning, the first dose of collector was added. Then after 2 minutes further conditioning, flotation was started, and the first concentrate increment was collected over 5 minutes. The second, third, and fourth collector additions were then made, each with a concentrate collection time of 4 minutes.

The instructions to the laboratory technician were to use MIBC frother as necessary to maintain stable froth. In practice, only one drop of MIBC was added to the rougher float.

Primary Grinds Evaluated

The simplified rougher flotation test procedure was used at grind P_{80}'s of 75, 106 and 150 µm on portions of K1, K2, Judd, and the Master Composite. Results, evaluated using the performance criteria as used for collectors, are summarized below in Table 13-19. The finest grind gave the best overall result in terms of performance criteria. However, the 0.106 mm grind gave the best copper and gold recoveries. This grind was therefore selected for rougher, cleaner testwork and for mill circuit modelling by external consultants, Orway Mineral Consultants.

Table 13-19 Grind Comparison - Total Performance Points

Grind P_{80} (µm)	75	106	150
Performance Points	170	162	153

13.7.5 Batch Rougher Cleaner Tests to Evaluate Concentrate Regrind

Collector Blend 1 with a 106 µm primary grind P_{80} was selected for batch rougher, cleaner flotation tests to evaluate the effects of rougher concentrate regrind.

Rougher Cleaner Test Procedure to Evaluate Rougher Concentrate Regrind

A rougher-cleaner test procedure was specified from consideration of results to date. The new procedure commenced with a 106 µm P_{80} primary grind and then conditioning at pH 11 with a starvation level (25 g/t Blend 1) of collector addition and then rougher flotation for 10 minutes. The instructions to the laboratory technician were to use MIBC frother as necessary to maintain stable froth. In practice, only one drop of MIBC was added to the rougher float.

Cleaner and recleaner flotation of the rougher concentrate were then evaluated with no regrinding and with two levels of regrind. Each cleaner float commenced with conditioning and lime addition to pH 12. After 2 minutes of lime conditioning, 25 g/t (dosage rate calculated on basis of rougher feed mass) of Blend 1 collector was added. Then after 2 minutes further conditioning, flotation was started, and the first cleaner concentrate was collected over 10 minutes. The concentrate was then returned to the float cell and second cleaner flotation was continued until flotation was complete.

A 2.5 litre laboratory float cell was used in all tests so the slurry density in cleaning was quite dilute. These conditions were selected to promote highly selective cleaner flotation.

Selection of Rougher Concentrate Regrind Intensity

A scouter regrind bead milling test was conducted on a batch of rougher concentrate (from a single float). Small samples were taken from the mill at time zero, 1 minute, 2 minutes and 3.8 minutes. These samples were then subject to laser sizing to determine the particle size distribution.

The results shown in Figure 13-10, give interesting perspective on the effects of bead milling on the size distribution. The size in µm is shown along the bottom axis of each histogram plot. The peak of the size distribution shifts down from about 100 µm, at time zero to about 30 µm after 1 minute, then to about 15 µm after 2 minutes and finally to about 8 µm after 3.8 minutes. As grinding progresses, the shape of the histogram changes: the bias towards larger sizes is lost and the distribution becomes narrower, and more bell shaped.

Based on these curves, regrind times of 2 minutes and 4 minutes were selected for evaluation against no regrind. It was judged that these regrind times resulted in a larger proportion of the concentrate being in the ideal size for flotation than for the non-reground concentrate.

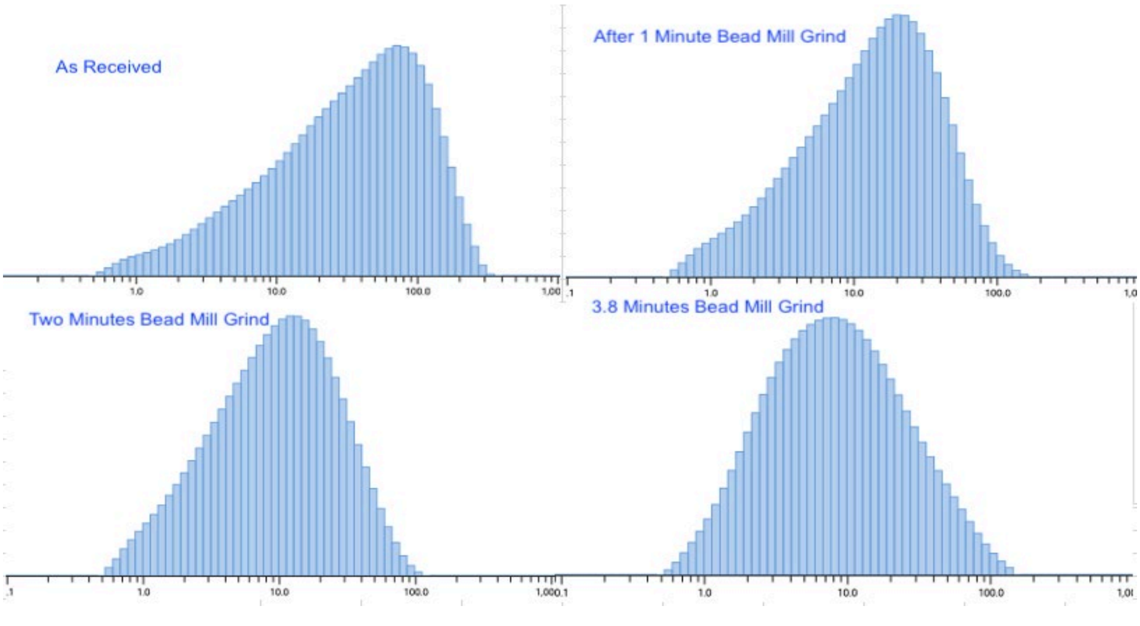


Figure 13-10 Size Distributions Measured During Scouter Regrind

Additional Tests to Evaluate Bulk Rougher Flotation

A fourth series of rougher-cleaner tests was performed on each of the belt cut samples to investigate bulk sulphide flotation in roughing followed by selective cleaning. For this series, a combination of copper sulphate activator, pH 9, and a sodium normal propyl xanthate co-collector at 15 g/t addition rate were used in the rougher float. Conditioning at pH 10 was used ahead of cleaner flotation. There was no regrind of rougher concentrate. These conditions were selected to achieve reduced selectivity against pyrite.

This bulk sulphide roughing test was added at the request of the K92ML Process Manager. It was postulated that having a greater recovery of sulphides to concentrate would boost gold recovery. Furthermore, it was expected that the additional sulphides in the final concentrate would displace fluorine bearing gangue minerals.

13.7.6 Recovery Versus Concentration Ratio Evaluation of Flotation Performance

The objectives of froth flotation are to recover and to concentrate value minerals. It follows that performance can be judged by plotting results on axes of Recovery % versus Concentration Ratio (CR). CR is defined as the weight of feed divided by the weight of concentrate; for example, a mass yield of 5% to concentrate would correspond to a CR of 20. This approach is particularly useful for comparing results where there is a wide range of head grade in the feed materials. In this respect, it is better than plotting recovery against concentrate grade as shown in Figure 13-9.

In practice, there is a trade-off between these two parameters and increased concentrate mass yields (i.e. lower CR) will be associated with increased recoveries. For any set of flotation conditions (grind, reagents, agitation, and aeration) there will be an operating relationship between recovery and CR. Flotation performance can be judged by the position of points on a graph of recovery vs CR; improved performance is demonstrated by combinations of higher recovery and increased CR.

13.8 Locked Cycle Rougher, Cleaner Recleaner Tests

Locked Cycle Flotation Testwork for Flowsheet Simulation

An estimate of the actual proportion of the cleaner tails that can be recovered to final concentrate can be determined by a locked cycle float test run. A locked cycle test run simulates a flowsheet with recycle streams in a sequence of batch tests. For example, to simulate a flowsheet with cleaner tails returning to the rougher feed, and second cleaner tails returning to first cleaner feed, batch tests would be conducted as follows:

The first cycle starts with rougher float using a portion of the ore sample. The rougher concentrate goes to cleaning and the cleaner concentrate goes to recleaning.

For the second cycle, the cleaner tail from the first cycle is combined with the rougher feed. Then the recleaner tails from the first cycle is combined with the rougher concentrate going to cleaner flotation. Similarly, the recleaner tails from the first cycle are combined with the cleaner concentrate that goes to recleaning.

This recombination of products from preceding cycles continues until steady circulating loads are established, simulating continuous operation of the flowsheet.

In practice, five or six cycles are used, and results are calculated from the weights and analyses of products from the last two cycles. Locked cycle testwork is relatively expensive due to the number of individual tests conducted in each campaign. However, doing two or more locked cycle tests allows a line to be plotted between points on a graph of recovery versus CR. This line gives a useful indication of the performance of the full-scale concentrator.

In feasibility studies, the recovery versus CR performance relationship can be used to estimate the effects on recovery of changing the concentrate grade target. Alternatively, this performance basis can be used to estimate value metals recovery with varying ore head-grade whilst maintaining a target concentrate grade.

For definitive feasibility studies, it is quite common to conduct mini plant testwork (treatment rate about 10 kg per hour), or pilot plant testwork (typical treatment rate >50 kg/h) for more detailed investigation of the treatment characteristics of various ore types and to further define the relationship between recovery and CR.

The K92ML Stage 3 Expansion Project will benefit from operating data from the existing plant. There are practical limits to the grind and other flotation parameters that can be investigated without adversely affecting operations. However, full-scale plant operating data has much greater authority than laboratory or pilot scale testwork.

(Reference: Flotation Mini Pilot Plant Experience at Falconbridge Limited, Dominic Fragomeni*, Maxine Hoffman, Andrew Kelly, Simon Yu and Norman O. Lotter ; MPMSC Conference, Sudbury, June 6-7, 2006.)

Flowsheets Selected of Locked Cycle Test Evaluation

Two simple flowsheets were chosen for evaluation by the locked cycle test. The first, shown in Figure 13-11, is similar to current plant operations, it has recirculation of cleaner tailings to rougher feed and does not have concentrate regrind. In tabulated results, this circuit will be referred to as 'LC No-Regrind'.

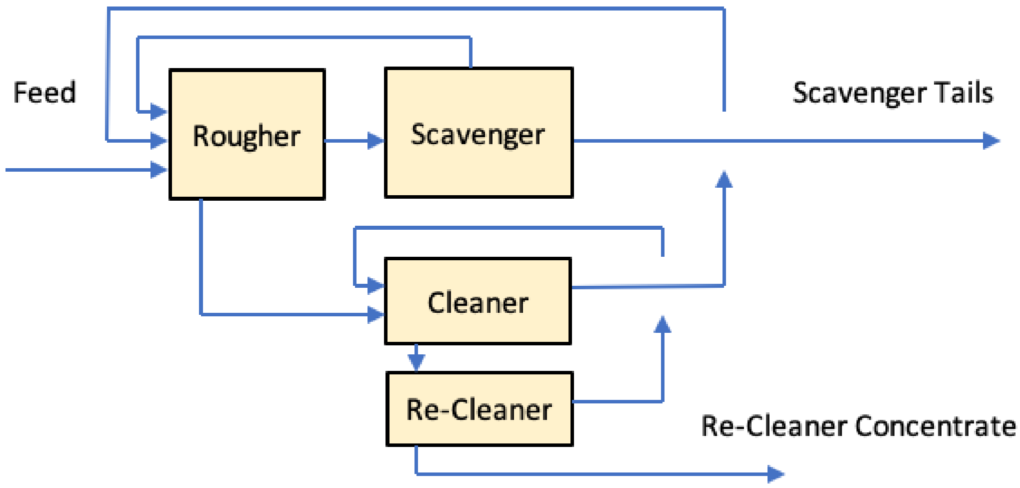


Figure 13-11 Flowsheet for the Closed-Circuit Cleaner Tails, No Regrind LCT Test

The second circuit incorporates regrind of rougher concentrates and has open circuit cleaner tails as shown in Figure 13-12. In tabulated results, this circuit will be referred to as 'LC With Regrind'.

With rougher concentrate regrind, it is generally considered poor practice to return the cleaner tailings to the rougher feed. The cleaner tails will have a finer particle size distribution than the rougher feed and the additional fines could adversely affect rougher recovery.

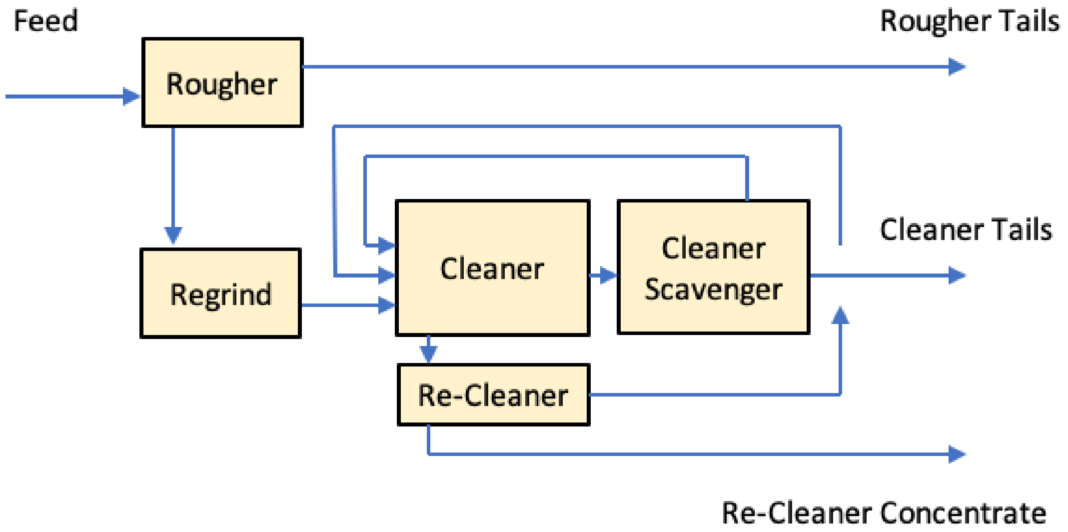


Figure 13-12 Flowsheet for the Open-Circuit Cleaner Tails, Rougher Concentrate Regrind LCT Test

Locked Cycle Test Procedures

Details of the 'LC No-Regrind' and 'LC With-Regrind' procedures are shown below in Table 13-20 and Table 13-21. The procedures are very similar to the rougher cleaner recleaner batch test.

Table 13-20 Details of the 'LC No-Regrind' Test Procedure

Parameter	Cycles 1 to 5			
	Ro	Cl	ReCl	Scav
Target Pulp Density (% solids (w/w))	~40			
Condition (rpm)	800			
Flotation (rpm)	800	800	800	800
Cell Size (L)	2.5	2.5	2.5	2.5
Float Cell Brand	ESSA	ESSA	ESSA	ESSA
Scrape Intervals (sec)	10	10	10	10
Target pH	11.0	12.0	12.0	11.0
pH Modifier	Lime	Lime	Lime	Lime
Collector Blend 1 (g/t)	25	25	-	15
Lime (10%) (ml) (shown for first cycle)	6.00	16.00	16.00	0.40
DOWFROTH 250-A (drops)	1	-	-	-
Total Conditioning Time (min)	**4.0**	**4.0**	**2.0**	**2.0**
Total Flotation Time (min)	**5.0**	**4.0**	**4.0**	**5.0**

Table 13-21 Details of the 'LC With-Regrind' Test Procedure

Parameter	Cycles 1 to 5 Use Same Procedure			
	Ro	Cl	ReCl	Cl Scav
Target Pulp Density (% solids (w/w))	~40			
Condition (rpm)	800			
Flotation (rpm)	800	800	800	800
Cell Size (L)	2.5	2.5	2.5	2.5
Float Cell	ESSA	ESSA	ESSA	ESSA
Scrape Intervals (sec)	10	10	10	10
Target pH	11.0	12.0	12.0	12.0
pH Modifier	Lime	Lime	Lime	Lime
Collector Blend 1* (g/t)	25	25	-	15
Lime (10%) (ml) (shown for first cycle)	6.20	16.00	16.00	-
DOWFROTH 250-A (drops)	2	-	-	-
Average Temperature (°C)	25.1	26.7	27.6	30.0
Total Conditioning Time (min)	**4.0**	**4.0**	**2.0**	**2.0**
Total Flotation Time (min)	**10.0**	**4.0**	**4.0**	**6.0**

In the LC test with regrind, each batch of rougher concentrate was ground for 2 minutes with 1.25 litres of 3 mm ceramic beads in a 3.2 litre vessel. The stirring speed was 1,450 RPM, corresponding to an agitator tip speed of 7.29 m/s. After some discussion with K92ML Operations, DOW 250 was selected at the frother rather than the MIBC used in previous tests. It had been noted that MIBC is unpopular as it is highly flammable.

In the tables, lime additions are shown for the first test cycles. With the recirculation of streams, the lime required to maintain selected pH levels decreased. For the 'LC No-Regrind' procedure, lime addition in the first cycle was 38.4 ml (of 10% $Ca(OH)2$) and this fell to 21 ml in the fifth cycle. Similarly, in the 'LC With-Regrind procedure, the lime addition in the first cycle was 32.0 ml, falling to 17.2 ml in the fifth cycle.

Results of the Locked Cycle and Regrind Cleaner Tests

Results of the locked cycle tests and the regrind cleaner tests are shown below in Table 13-22. Recoveries of gold and copper, as well as multi element recleaner concentrate analyses are included. The resolution of the analysis method used for fluorine was 200 ppm, so values have been rounded to the nearest 200 ppm interval.

Results for the same series of tests are also presented in Table 13-23 with gold, copper and silver built up heads, recoveries, and CR (concentration ratio) shown.

This body of results shows that regrinding generally results in improved concentrate grade but at the expense of gold recovery. A comparison of the fluorine levels in the two locked cycle tests shows significant benefits from regrinding.

Table 13-22 Results of the Locked Cycle and Regrind Cleaner Tests Gold and Copper Recoveries, Built-up Head Grade, and Concentrate Analyses

Test no.	Sample Tested	Test Details	Built Up Head Grade		Rougher Cleaner Recleaner Recovery		Concentrate Grade								
			Au g/t	Cu %	Au %	Cu %	Au g/t	Cu %	Ag g/t	Fe %	Zn %	S %	F ppm	SiO_2 %	Al_2O_3 %
31	K1	No Regrind	7.99	0.45	80.4%	89.9%	120.5	7.6	38.0	36.8	0.47	43.18	400	8.85	1.39
35	K1	Regrind 2 Minutes	7.80	0.40	69.2%	86.1%	239.1	15.4	76.0	27.6	0.95	31.17	800	16.03	3.31
39	K1	Regrind 4 Minutes	6.91	0.38	56.3%	79.1%	227.2	17.4	77.0	25.4	1.23	29.49	1000	19.73	4.34
43	K1	Bulk Sulphide	8.19	0.38	77.4%	88.5%	91.7	4.9	27.0	38.7	0.29	44.91	400	8.54	2.18
32	K2	No Regrind	4.66	2.32	76.2%	94.4%	31.1	19.2	304.0	32.7	0.23	39.86	0	6.65	1.17
36	K2	Regrind 2 Minutes	6.11	2.24	51.7%	84.5%	52.0	31.2	500.0	24.8	0.42	31.91	400	6.98	1.54
40	K2	Regrind 4 Minutes	6.89	2.23	43.6%	68.1%	62.1	31.4	499.0	24.7	0.55	31.26	400	6.37	1.49
44	K2	Bulk Sulphide	5.10	2.17	76.2%	91.8%	39.0	20.0	315.0	29.3	0.27	37.31	400	7.78	2.15
33	Judd	No Regrind	3.27	0.51	55.9%	89.6%	82.6	20.5	390.0	24.6	4.91	31.26	0	12.52	1.76
37	Judd	Regrind 2 Minutes	3.66	0.52	50.7%	89.4%	84.5	21.3	472.0	24.8	5.44	31.10	0	13.01	3.02
41	Judd	Regrind 4 Minutes	4.26	0.48	48.7%	87.5%	101.8	20.8	486.0	22.0	5.26	27.97	400	13.66	3.19
45	Judd	Bulk Sulphide	3.16	0.51	66.8%	78.9%	62.9	12.1	291.0	33.0	2.27	40.01	200	7.43	1.89
30	Master	No Regrind	5.48	1.20	76.3%	89.1%	93.0	23.9	323.0	29.5	0.47	35.84	800	7.43	1.22
34	Master	Regrind 2 Minutes	5.50	1.29	61.2%	73.8%	121.2	34.2	462.0	24.3	0.72	31.54	200	6.67	1.61
38	Master	Regrind 4 Minutes	5.36	1.23	58.1%	67.8%	120.8	32.5	461.0	23.6	0.77	30.29	200	5.83	1.42
42	Master	Bulk Sulphide	6.93	1.25	77.1%	90.1%	57.6	12.1	163.0	36.2	0.26	42.98	0	3.41	1.01
	Master	LC with Regrind	7.60	1.26	79.83%	94.53%	140.3	25.9	377.5	23.4	0.5	29.8	600	13.2	2.72
	Master	LC no Regrind	7.57	1.27	91.85%	97.98%	78.2	12.8	197.0	20.1	0.3	24.0	1800	29.1	6.867

Table 13-23 Results of the Locked Cycle and Regrind Cleaner Tests Rougher Cleaner Recleaner Recoveries and Concentration Ratios

Test Number	Sample Tested	Test Details	Built Up Head			Rougher Cleaner Recleaner Recovery			Rougher Cleaner Recleaner CR
			Au g/t	Cu %	Ag g/t	Au %	Cu %	Ag %	
30	Master	No Regrind	5.48	1.20	17.76	76.3%	89.1%	73.7%	22.3
34	Master	Regrind 2 Minutes	5.50	1.29	19.64	61.2%	73.8%	53.4%	36.0
38	Master	Regrind 4 Minutes	5.36	1.23	19.97	58.1%	67.8%	66.5%	38.8
42	Master	Bulk Sulphide Rougher	6.93	1.25	18.47	77.1%	90.1%	78.4%	10.8
31	K1	No Regrind	7.99	0.45	4.80	80.4%	89.9%	38.7%	18.8
35	K1	Regrind 2 Minutes	7.80	0.40	3.13	69.2%	86.1%	64.1%	44.3
39	K1	Regrind 4 Minutes	6.91	0.38	3.28	56.3%	79.1%	68.6%	58.3
43	K1	Bulk Sulphide Rougher	8.19	0.38	2.78	77.4%	88.5%	63.6%	14.5
32	K2	No Regrind	4.66	2.32	37.24	76.2%	94.4%	81.6%	8.8
36	K2	Regrind 2 Minutes	6.11	2.24	36.69	51.7%	84.5%	78.4%	16.5
40	K2	Regrind 4 Minutes	6.89	2.23	39.48	43.6%	68.1%	82.7%	20.7
44	K2	Bulk Sulphide Rougher	5.10	2.17	35.08	76.2%	91.8%	83.0%	10.0
33	Judd	No Regrind	3.27	0.51	13.42	55.9%	89.6%	76.0%	45.1
37	Judd	Regrind 2 Minutes	3.66	0.52	14.12	50.7%	89.4%	82.2%	45.5
41	Judd	Regrind 4 Minutes	4.26	0.48	14.01	48.7%	87.5%	70.5%	49.1
45	Judd	Bulk Sulphide Rougher	3.16	0.51	14.04	66.8%	78.9%	54.4%	29.8
LC-RG	Master	Locked Cycle with Regrind	8.08	1.26	1.26	79.83%	94.53%	77.94%	21.8
LC-NRG	Master	Locked Cycle no Regrind	9.32	1.43	1.43	91.85%	97.98%	Close to 100%	9.15

Recovery Versus Concentration Ratio Comparisons Between Results for the K1, K2, Judd, and Master Composite

Gold recovery versus Concentration Ratio results for the K1, K2, Judd and Master Composites are shown below in Figure 13-13. Similarly copper recovery results are shown in Figure 13-14. Results are shown for the rougher cleaner recleaner tests on the four composites and the locked cycle tests on the Master Composite.

For each composite, the rougher cleaner recleaner tests were repeated using four different procedures (three levels of rougher concentrate regrind and the bulk sulphide run). There is some scatter in the data points (for each composite) due to the different test procedures. Increased levels of rougher concentrate regrind generally give higher concentration ratio and lower recovery. The bulk sulphide runs give low concentration ratios and high recovery.

The copper recovery results shown in Figure 13-14 are slightly more scattered than the gold results. From the data shown in Figure 13-14, this is due to some low recoveries associated with the 4 minute regrind of rougher concentrate.

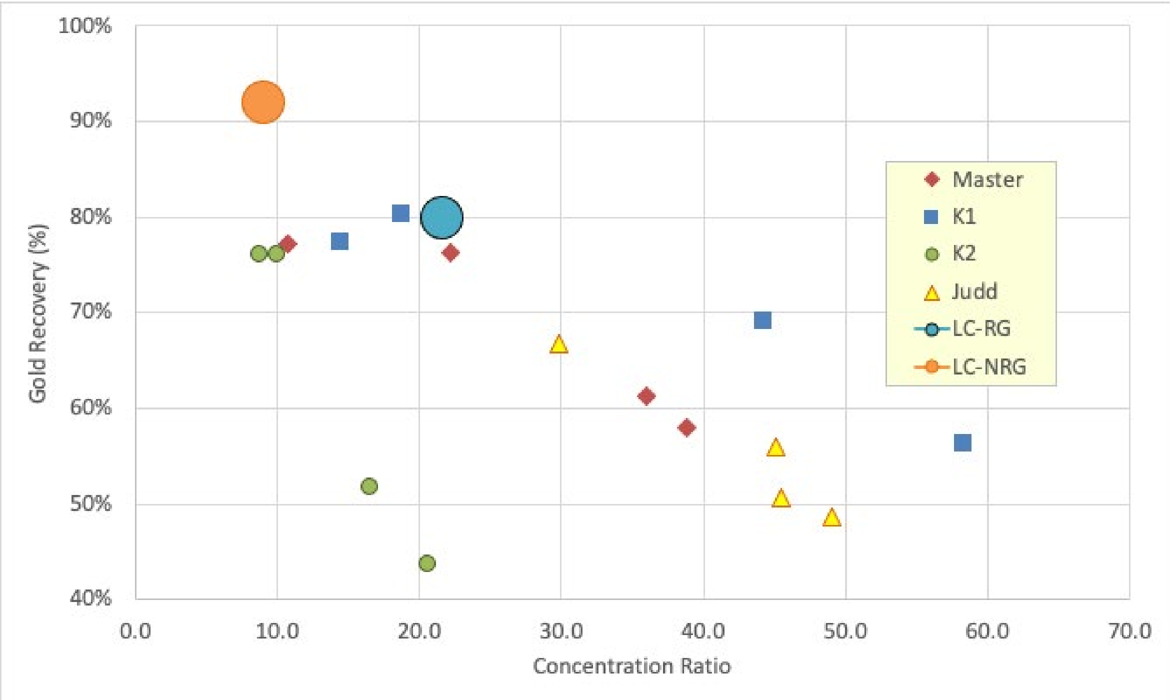


Figure 13-13 Gold Recovery versus Concentration Ratio Results for the K1, K2, Judd, and Master Composites (Locked Cycle and Rougher Cleaner Recleaner Tests)

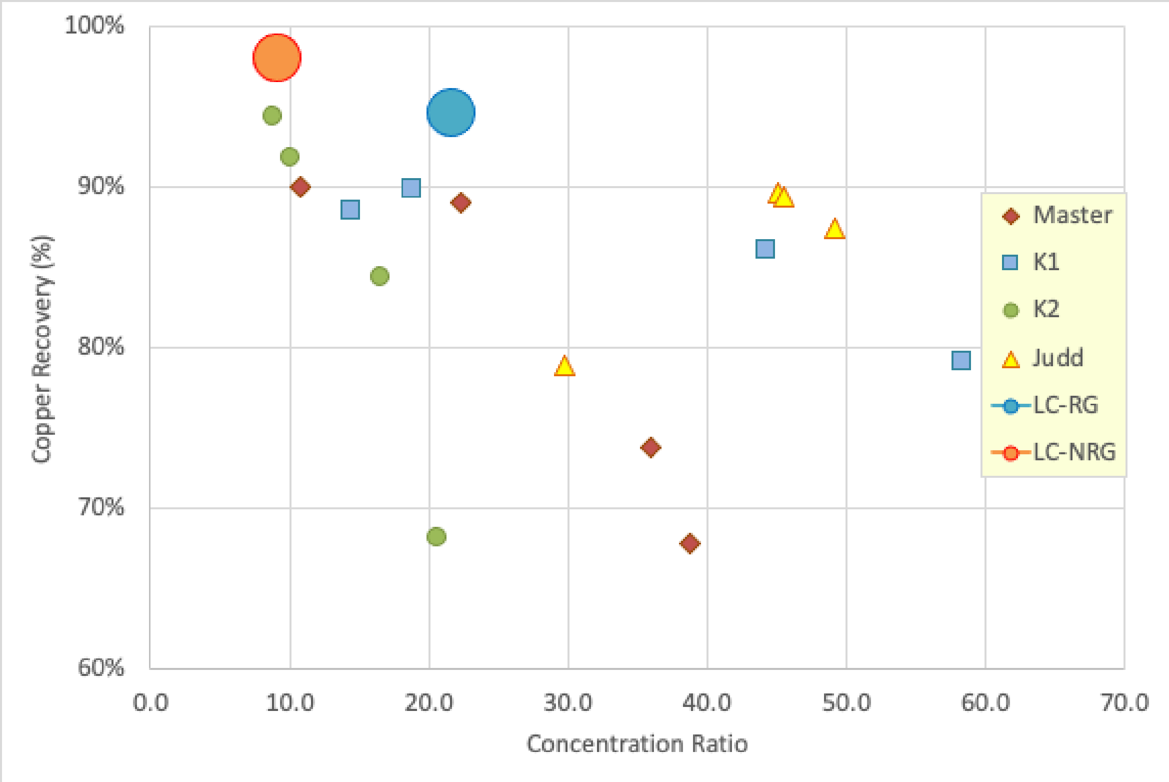


Figure 13-14 Copper Recovery versus Concentration Ratio Results for the K1, K2, Judd, and Master Composites (Locked Cycle and Rougher Cleaner Recleaner Tests)

However, when considered in terms of the relationships between gold and copper recoveries and concentration ratio, the response of the four composites is quite similar. This is an important conclusion.

Recovery versus Concentration Ratio Evaluation of the Rougher Regrind Cleaner Flotation Results

In Figure 13-15, gold recovery results from the two locked cycle (LC) tests are shown plotted on Recovery versus CR axes. The high recovery, low CR result is for the LC No-Regrind run, whilst the higher CR and lower recovery point is for the LC With-Regrind run. These LC results are close to the upper margin of the data points for the open circuit batch tests. This is to be expected as the LC recovery includes a contribution from recirculated tailings products.

A line between the two LC points would have the relationship:

Equation 1: Gold Recovery % = 100 * (-(0.0095 * CR) + 1.0057)

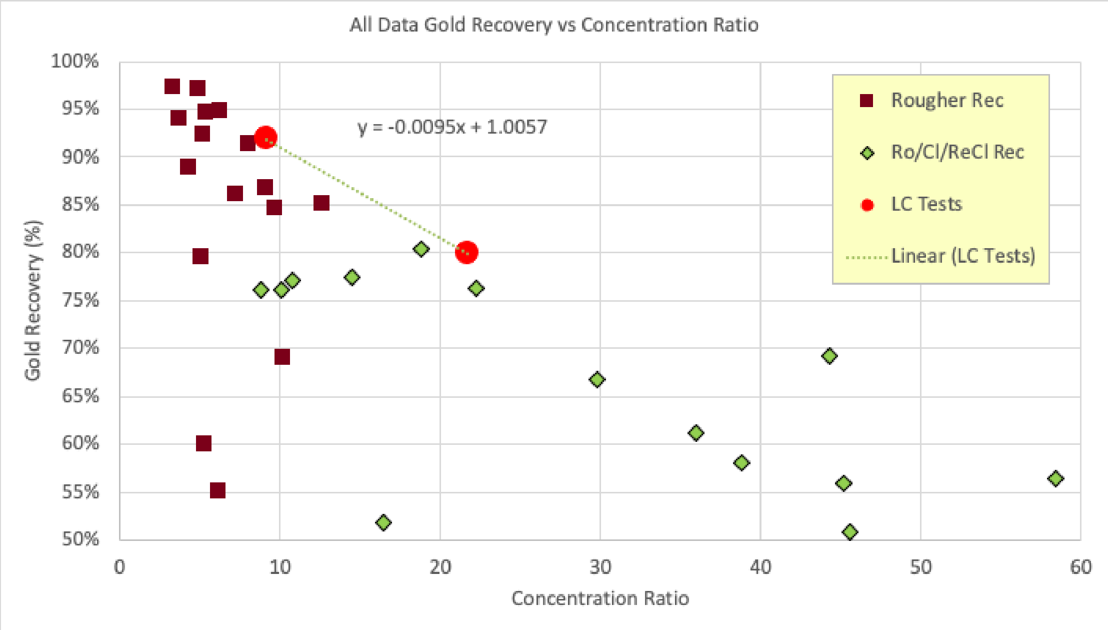


Figure 13-15 Gold Recovery versus Concentration Ratio for the Locked Cycle Tests. Shown with the Rougher, and the Rougher Cleaner Recleaner Results

This relationship could be used to predict gold recovery performance for a range of concentration ratios. For example, at a CR of 10, the gold recovery will be 91.07%. Therefore, if the head grade was 10 g/t Au, the concentrate would be 91.07 g/t Au. The three product formulas for recovery and concentration ratio (based on head grades, concentrate grades and tailings grades) can be used to estimate other combinations of performance data.

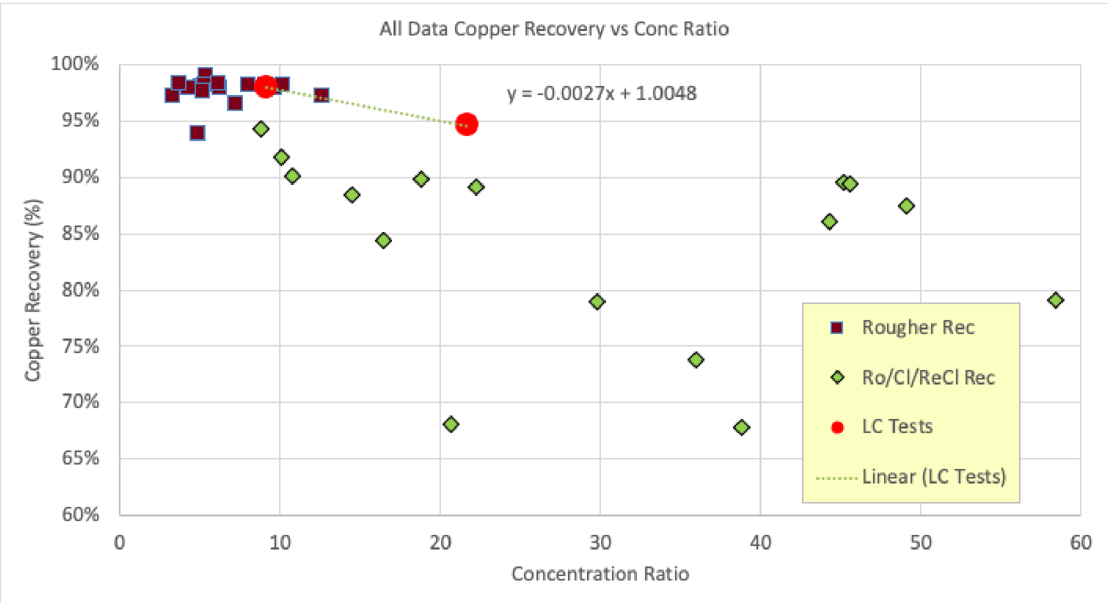


Figure 13-16 Copper Recovery Results from the Locked Cycle Tests Shown with the Rougher, Cleaner Recleaner Results

Similar comments apply to copper and silver recovery in the LC Tests. For copper, the line between the two LC results has the equation:

Equation 2: Copper recovery % = 100 * (1.0048 - (0.0027 * CR))

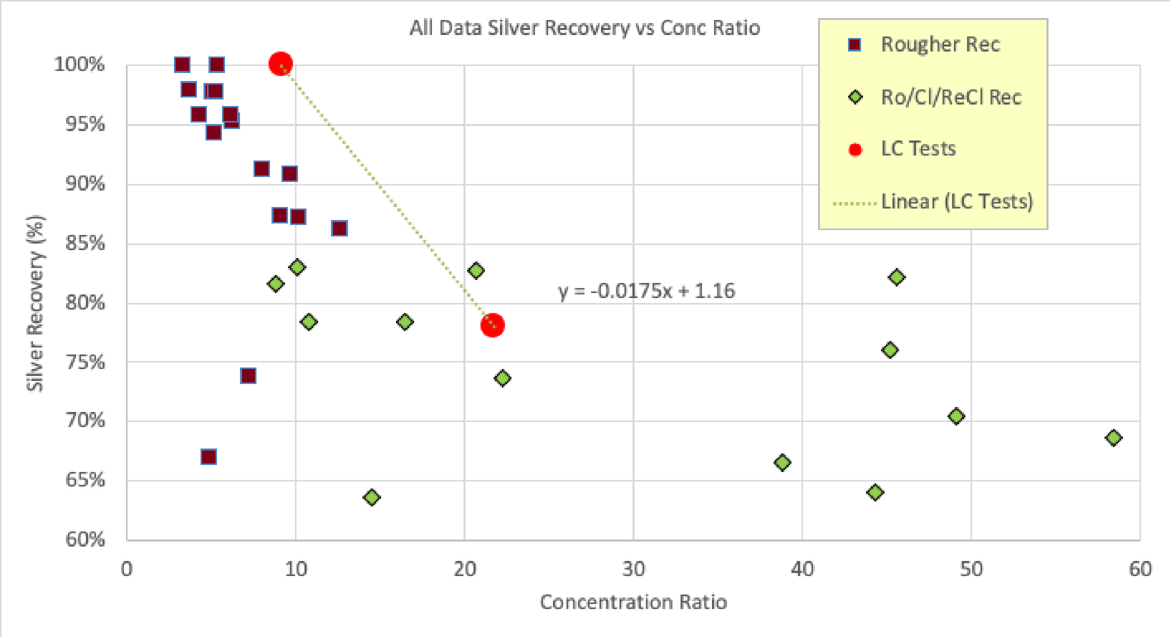


Figure 13-17 Silver Recovery Results from the Locked Cycle Tests Shown with the Rougher, Cleaner Recleaner Results

For silver, the line between the two LC results has the equation:

Equation 3: Silver recovery % = 100 * (1.16 - (0.0175 * CR))

Implications for Full Scale Operations

To produce a high-grade concentrate, the full-scale plant might operate at a CR of 20 (i.e. 5% mass pull to final concentrate). From Equation 2, the flotation gold recovery would be 81.5%, from Equation 2, the copper recovery would be 95.1% and from Equation 3, the silver recovery would be 81.0%.

These estimates of flotation recovery are more robust than estimates based on the batch testwork. The gold recovery estimates are for the master composite sample (a 50/50 blend of K1 and K2 ore) using conventional float cells. Actual total gold recoveries will be higher than these estimates. The new K92ML concentrator includes both flash flotation and a centrifugal concentrator for recovery of gold particles that are too large for conventional froth flotation recovery.

This additional gold recovery will depend in the mineralogy of the ore, and on the details of the concentrator design (with respect to coarse particle gold recovery). From the GRG test results, the Master Composite (as a 50/50 blend of K1 and K2 Ore) contains 16.2% of the gold as +150 µm particles and 21.3% of the gold as +106 µm particles.

13.9 Variability Testwork on Eight Composites of Diamond Drill Core

Overview and Conclusion

All variability samples showed excellent response to flotation recovery.

A single rougher rate test was conducted on each of the variability samples. Results were compared with the body of rougher tests conducted to date. Gold and copper recovery versus concentration ratio results were plotted. Recoveries and concentrate analyses were tabulated. The flotation rates of gold and copper values were compared with results for the belt cut samples. Selectivity of copper flotation against iron flotation was also compared.

Rougher Test Procedure to Evaluate Variability Samples

A simplified rougher rate test procedure was specified from consideration of the body of existing results. The procedure commenced with a 106 µm P_{80} primary grind and then conditioning at pH 11 for 2 minutes. This was followed by conditioning with a starvation level (25 g/t Blend 1) of collector for a further 2 minutes and then rougher flotation for 10 minutes. To provide information on flotation rate, concentrate was collected in increments at flotation times: 1, 2, 3, 5, and 10 minutes. The laboratory technician was instructed to use MIBC frother as necessary to maintain stable froth. In practice, only one drop of MIBC was added during collector conditioning.

Results of the Rougher Flotation Tests on Variability Samples

Results of the rougher flotation tests on the seven variability samples are shown at the top of Table 13-24. For comparison, results of all rougher flotation tests using Blend 1 collector are shown below the results for the variability samples. Each result is for the total rougher concentrate collected from a single test.

Gold and copper recoveries are shown plotted against concentration ratio in Figure 13-13 and Figure 13-14 respectively. The variability results show similar performance to the earlier tests (on samples K1, K2, Judd and Master Composite). The variability results were achieved with a starvation dose of collector (25 g/t Blend 1). As shown in the legends of the figures, some of the compared results utilized higher collector additions.

It is interesting to note that the fluorine contents of the concentrates from the variability tests are generally lower than achieved in the earlier tests.

Table 13-24 Rougher Flotation - All Test Results Using Blend 1 Collector

Sample	Conditions	CR	Recovery		Concentrate Grade									
			Au %	Cu %	Aug/t	Cu %	Fe %	Zn %	S %	F ppm	SiO_2%	Al_2O_3 %	CaO %
J1B-1	Variability, Grind 106 µm, 25 g/t Blend 1	11.1	72.3%	97.6%	140.0	12.5	17.1	1.5	19.9	690	33.2	8.3	0.7
K1A-1	Variability, Grind 106 µm, 25 g/t Blend 1	6.9	91.7%	96.5%	158.2	21.1	25.7	0.1	31.3	210	14.9	3.3	0.9
K1B-1	Variability, Grind 106 µm, 25 g/t Blend 1	17.1	87.6%	89.9%	243.4	3.6	23.0	0.6	25.9	1220	30.8	8.9	0.9
KLA-1	Variability, Grind 106 µm, 25 g/t Blend 1	5.3	88.4%	97.5%	36.3	3.3	33.2	0.6	38.2	470	15.6	4.3	0.9
K2A-1	Variability, Grind 106 µm, 25 g/t Blend 1	4.7	75.7%	96.3%	24.2	4.3	28.4	9.4	38.5	150	10.9	2.7	0.8
K2B-1	Variability, Grind 106 µm, 25 g/t Blend 1	4.9	91.4%	96.8%	11.8	4.7	33.1	0.3	39.8	470	15.5	4.3	0.9
K2C-1	Variability, Grind 106 µm, 25 g/t Blend 1	3.1	74.9%	98.4%	27.8	6.6	38.1	0.1	46.6	110	8.4	1.8	0.7
K1	Grind 75 µm, 100 g/t Blend 1	2.89	99.2%	99.2%	21.97	1.11	22.5	0.08	25.0	2790	34.6	10.5	0.5
K2	Grind 75 µm, 100 g/t Blend 1	4.05	53.4%	99.3%	71.48	8.83	21.0	0.11	26.0	1562	28.7	8.4	0.3
Judd	Grind 75 µm, 100 g/t Blend 1	4.04	97.8%	98.6%	11.58	2.11	18.3	0.55	19.1	1282	39.4	10.4	0.9
MC	Grind 75 µm, 100 g/t Blend 1	3.05	97.2%	99.5%	15.75	3.84	19.9	0.10	21.6	1719	36.9	10.1	0.4
K1	Grind 106 µm, 50 g/t Blend 1	4.00	95.1%	100.0%	29.17	1.63	31.0	0.11	34.5	1817	21.5	7.2	0.4
K2	Grind 106 µm, 50 g/t Blend 1	3.76	84.8%	99.4%	23.95	8.77	19.8	0.10	23.0	1733	31.3	9.2	0.5
Judd	Grind 106 µm, 50 g/t Blend 1	10.50	92.9%	98.4%	33.70	5.69	26.4	1.42	29.3	976	24.7	6.7	0.6
MC	Grind 106 µm, 50 g/t Blend 1	5.30	94.7%	99.1%	29.14	6.89	28.1	0.14	32.6	1303	22.0	6.2	0.2
MC	Grind 106 µm, 25 g/t Blend 1	6.27	94.9%	97.9%	32.55	7.38	31.3	0.15	36.1	2600	16.8	4.4	0.4
K1	Grind 106 µm, 25 g/t Blend 1	4.85	97.1%	93.9%	37.63	2.06	33.9	0.15	39.1	3000	16.6	4.5	0.3
K2	Grind 106 µm, 25 g/t Blend 1	5.09	79.5%	98.1%	18.85	11.58	27.2	0.14	32.5	No Data	19.3	5.7	0.4
Judd	Grind 106 µm, 25 g/t Blend 1	12.63	85.1%	97.3%	35.19	6.22	22.7	1.55	25.7	No Data	27.8	7.4	0.8
MC	Grind 106 µm, 25 g/t Blend 1	9.69	84.7%	97.9%	45.15	12.22	19.1	0.23	22.0	2600	28.7	9.0	0.9
K1	Grind 106 µm, 25 g/t Blend 1	5.37	94.7%	99.0%	39.69	2.146	32.3	0.14	36.4	2800	19.7	5.8	0.5
K2	Grind 106 µm, 25 g/t Blend 1	5.27	60.0%	98.2%	19.29	11.62	22.3	0.14	26.2	3000	25.6	7.3	0.5
Judd	Grind 106 µm, 25 g/t Blend 1	10.19	69.0%	98.3%	25.70	5.23	14.6	1.30	15.0	1600	40.5	11.0	1.3
MC	Grind 106 µm, 25 g/t Blend 1	8.01	91.3%	98.2%	39.27	9.717	24.3	0.19	28.1	2600	24.2	7.0	0.8
K1	Grind 106 µm, 25 g/t Blend 1	7.17	86.1%	96.6%	42.67	2.61	23.7	0.18	26.3	2600	30.9	9.2	0.6
K2	Grind 106 µm, 25 g/t Blend 1	6.09	55.0%	98.3%	23.10	13.38	21.4	0.18	25.6	2800	25.1	7.6	0.6
Judd	Grind 106 µm, 25 g/t Blend 1	9.13	86.9%	98.2%	33.74	4.345	10.5	1.18	9.8	1400	46.7	12.4	1.4
MC	Grind 106 µm, Bulk Sulfide Float	4.24	88.9%	97.9%	26.13	5.190	26.0	0.11	29.9	1250	26.0	6.6	0.3
K1	Grind 106 µm, Bulk Sulfide Float	3.32	97.3%	97.3%	26.46	1.234	24.0	0.09	27.1	1490	35.1	8.2	0.2
K2	Grind 106 µm, Bulk Sulfide Float	3.67	94.1%	98.3%	17.58	7.814	19.5	0.10	23.0	2060	33.7	9.1	0.4
Judd	Grind 106 µm, Bulk Sulfide Float	5.17	92.4%	97.6%	15.11	2.593	22.9	0.66	25.4	1120	32.8	8.2	0.5

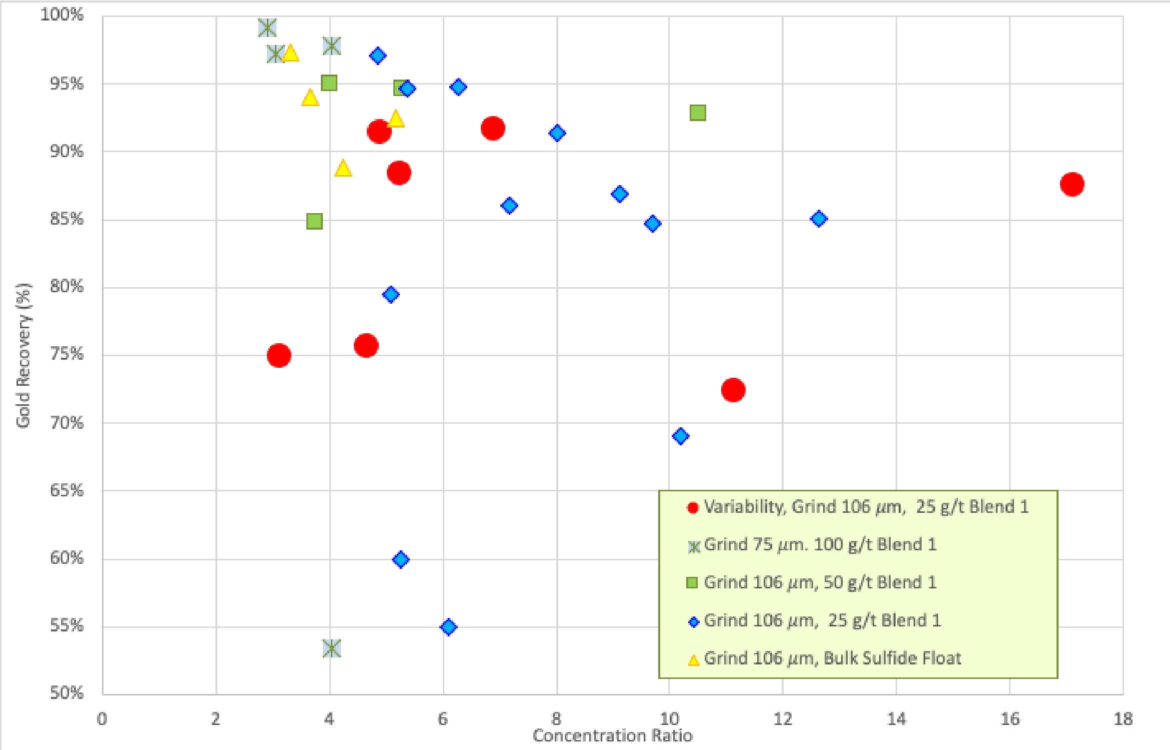


Figure 13-18 Gold Recovery versus CR for All Rougher Floats Using Blend 1 Collector

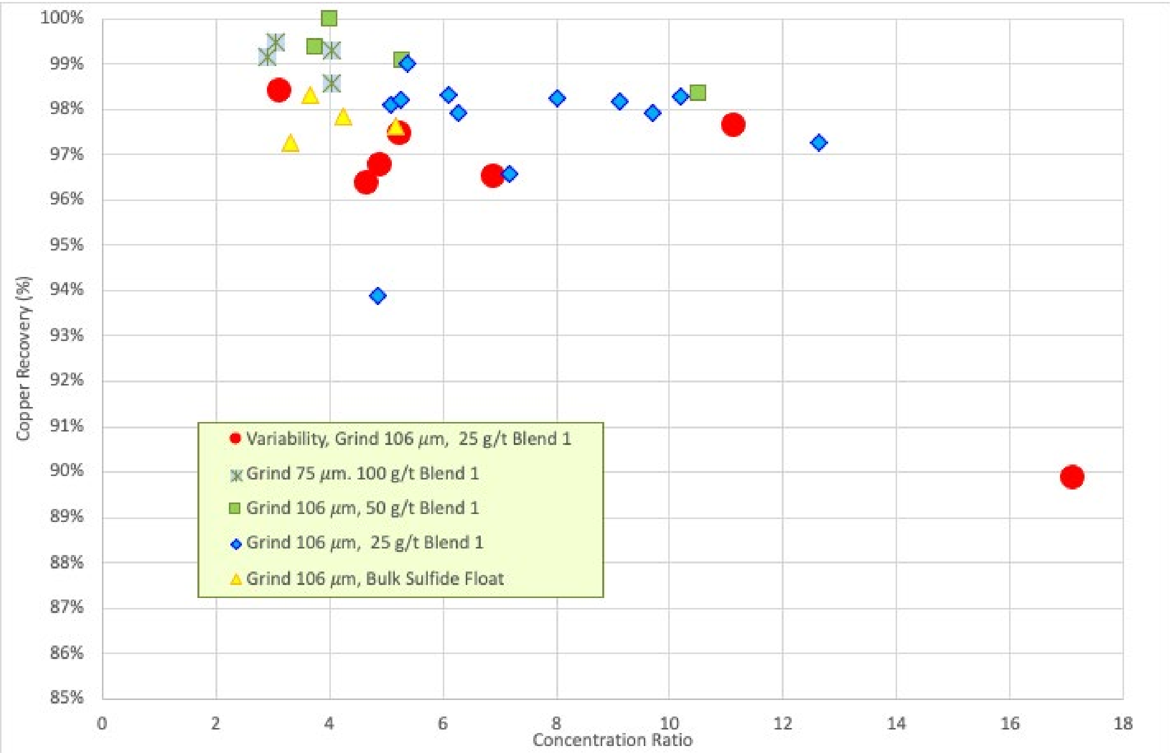


Figure 13-19 Copper Recovery versus CR for All Rougher Floats Using Blend 1 Collector

Rate of Flotation Comparisons

The flotation rates of gold and copper for the variability samples are shown in Figure 13-20 and Figure 13-21 respectively. Results for the belt cut samples (K1, K2, Judd, and the Master Composite) are also shown for comparison.

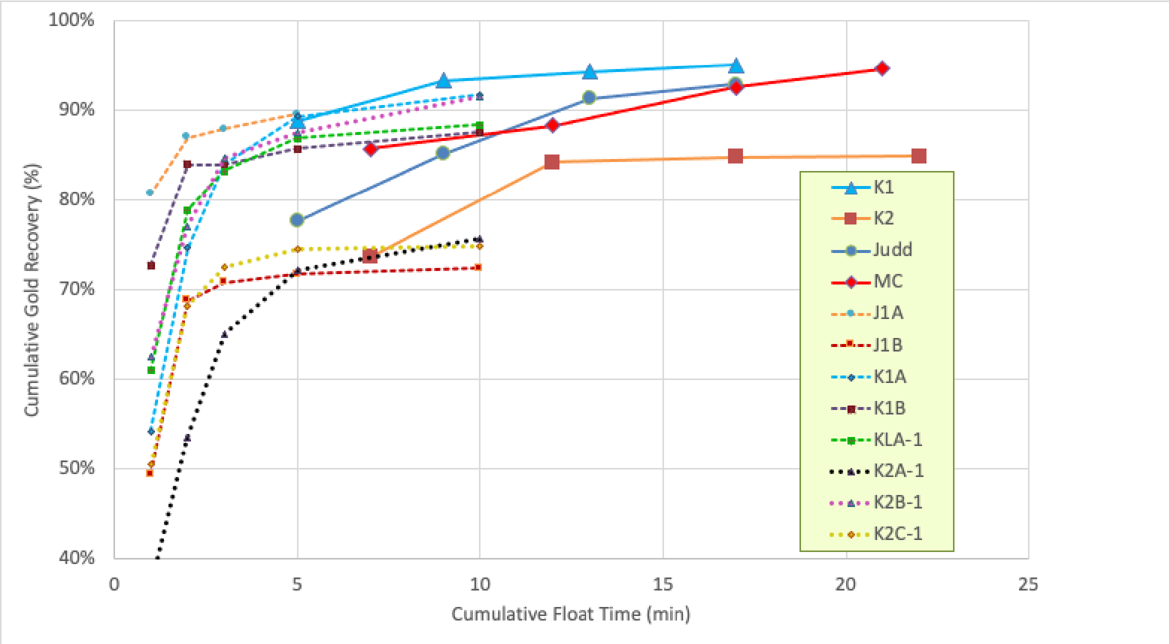


Figure 13-20 Gold Flotation Kinetics of the Variability Samples

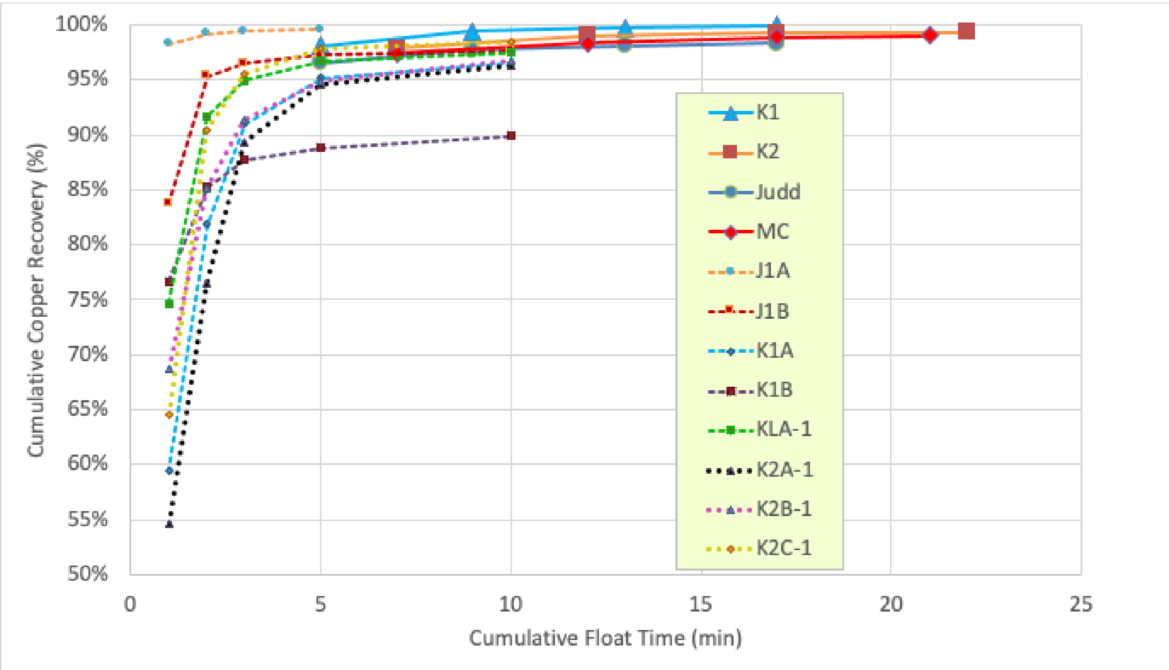


Figure 13-21 Copper Flotation Kinetics of the Variability Samples

The results for the two sets of samples ('belt cut' and 'variability') are not fully comparable as slightly different test procedures were used in the two cases. Both cases used a primary grind P_{80} of 106 µm. The reagent additions used for the belt cut samples was as described in Section 13.8. For these floatation tests, a total of 50 g/t of Blend 1 collector was added in 4 increments of 12.5 g/t. After the first increment, flotation was continued until near extinction. The next addition of collector was made and after two minutes of conditioning, flotation to was again continued to extinction. This was repeated for the third and fourth increments of collector.

For the variability floats, there was a single addition of 25 g/t of Blend 1 collector followed by four timed increments of flotation with no intermediate conditioning steps. This is a classic 'Rougher Rate' test procedure, and the results show that after 10 minutes, laboratory scale flotation of gold and copper values is substantially complete. The staged flotation procedure used for the belt cut samples uses longer total flotation times. However, in most cases substantially complete recoveries are achieved in about 10 minutes of flotation.

Despite the differences in test procedure, it can be concluded that there are no major differences in flotation kinetics between the 'Belt Cut' and 'Variability Samples'.

Selectivity for Copper versus Iron Flotation

Figure 13-22 shows copper recovery plotted against iron recovery with results for the variability samples and the belt cut samples shown for comparison. It is interesting to note that all belt-cut sample tests show a highly selective start to flotation with selectivity decreasing as more iron floats. Results for the variability samples follow a slightly different pattern with a lower level of selectivity against iron being maintained throughout the float.

Two of the variability samples, J1B and K1B show excellent selectivity whilst the other four samples still show good selectivity. It is interesting to note, from Table 13-24 that the two samples with the most selective flotation response have the highest fluorine in concentrate levels (of the variability samples).

The use of different test procedures is probably responsible for the different patterns of selective behaviour. However, all tests show selective flotation with copper recoveries remaining higher than iron recoveries.

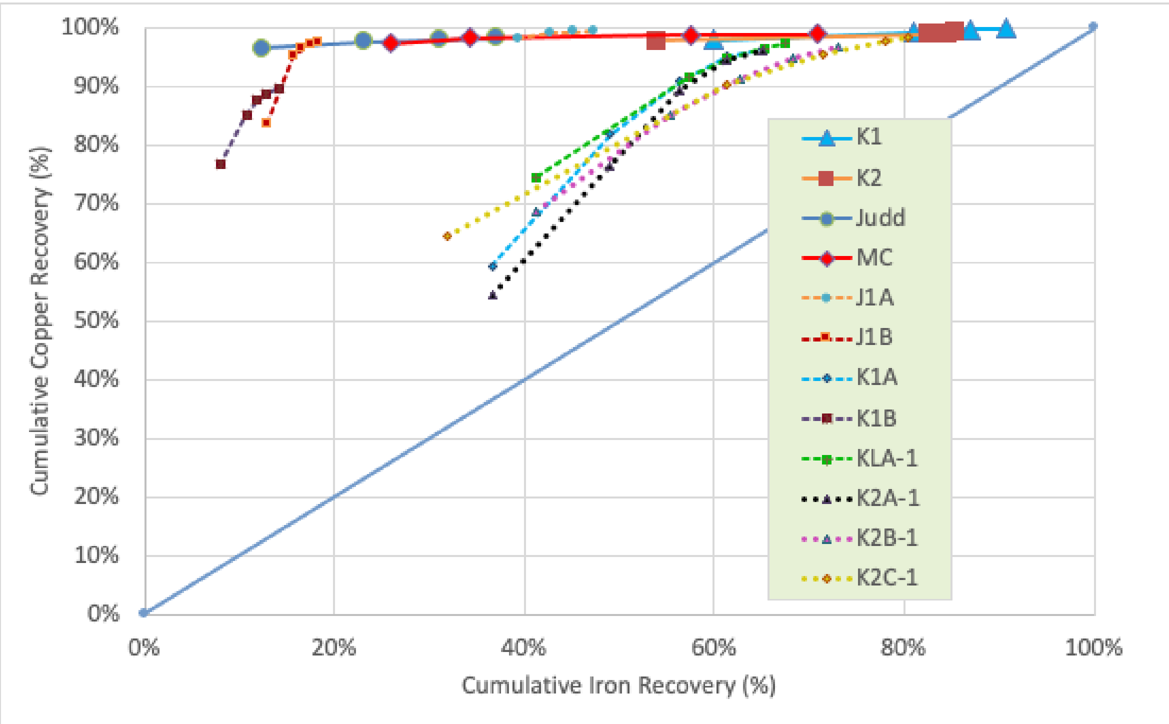


Figure 13-22 Copper versus Iron Flotation Selectivity for the Variability Samples

13.10 Evaluation of Current Plant Operating Results

13.10.1 Staged Improvements to the Existing Plant

The processing plant commenced operations under K92ML management during 2017. Essential modifications and additions to the plant were made and commercial operation was declared in 2018.

Subsequent revisions to the processing plant have allowed increased throughput whilst maintaining gold and copper recoveries. A history of these revisions is summarised below in Table 13-25.

Table 13-25 Staged Improvements to the Existing Plant

Period		Plant Revision	Details Announced
1	2018Q1	Rev 1	Commercial production with plant capacity 200k tpa. Stage 2 expansion to 400k tpa awarded to Mincore
6	2019Q2	Rev 2	Gravity Circuit completed but not commissioned. New mill circuit cyclones and float feed tank installed.
9	2020Q1	Rev 3	Larger secondary crusher installed
12	2020Q4	Rev 4	Stage 2 expansion commissioning completed. Five consecutive days at >1,200 tpd achieved.
15	2021Q3		Stage 2A expansion start announced. Additional TC1000 secondary crusher due Q4 2021, and additional rougher cells due Q3,2022.
16	2021Q4		Stage 2 expansion throughput rate (100 kt per quarter) achieved.
18	2022Q2		Second TC1000 crusher installed.
19	2022Q3	Rev 5	The first quarter with the gravity circuit fully operational.
20	2022Q4		Delivery of new rougher float cells.
21	2023Q1		Gravity circuit operating and doré metal shipment starts this quarter
22	2023Q2		Stage 2A expansion completed. New rougher float cells commissioned. Plant capacity 500k tpa (125k t per Quarter)
23	2023Q3	Rev 6	Stage 2A in operation with the capacity 600k tpa

13.10.2 Operating Data

Quarterly data for the existing K92ML Concentrator over the period January 2018 to June 2024 is shown below in Table 13-26. Plant throughput with gold and copper recovery is shown in Figure 13-23.

For the first two quarters of 2021 (periods 13, 14 in Figure 13-23), mining and concentrator operations suffered personnel shortages caused by the COVID pandemic. In the plant, the tons of ore treated was maintained but metals recovery was adversely affected. The situation started to ease in the third quarter of 2021 with plant operations improving. For the final quarter of 2021 and first quarter of 2022, the tonnage milled was close to the design rate of 400,000 tonnes per year (100,000 tonnes per quarter or 33,000 tons per month).

The Falcon concentrator was fully operational by 2022 Q3 (period 19 in Table 13-25). Parallel operation of the gravity gold recovery and flash flotation would have helped maintain gold recovery as the throughput increased.

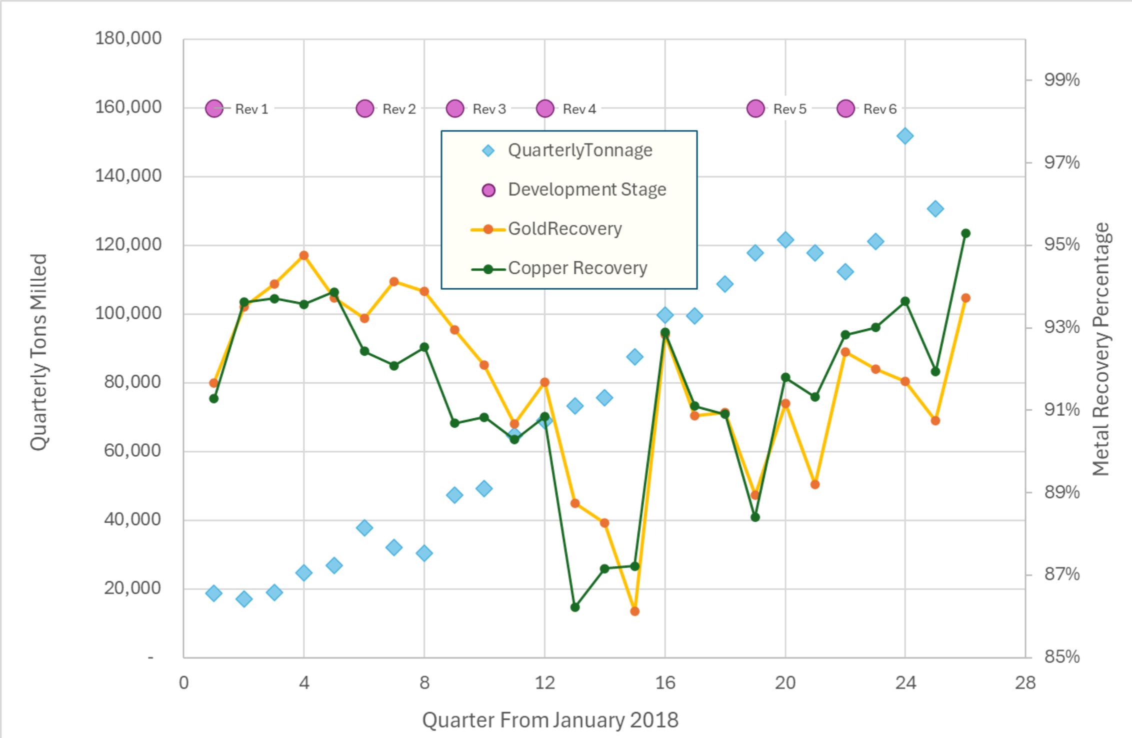


Figure 13-23 Plant Revisions, Throughput, and Metals Recovery

Table 13-26 K92 Existing Plant, Quarterly Operating Data from January 2018 to June 2024

Period			Tons Milled and Head-grade			Concentrate Tons and Metals Recovered				Recovery		CR
Number	Quarter	Circuit	Dry tons	Au g/t	Cu %	Dry tons	Au oz	Cu t	Ag oz	Au %	Cu %	
1	2018Q1	Rev 1	18,668	16.95	0.44	1067	9323	75	4743	91.7%	91.3%	17.5
2	2018Q2		17,105	20.39	0.36	915	10486	58	1704	93.5%	93.6%	18.7
3	2018Q3		18,907	16.70	0.37	1085	9549	66	2537	94.1%	93.7%	17.4
4	2018Q4		24,806	21.77	0.33	1266	16451	77	3088	94.8%	93.6%	19.6
5	2019Q1		26,846	23.64	0.48	1632	19125	120	5409	93.7%	93.9%	16.4
6	2019Q2	Rev 2	37,913	16.70	0.34	2192	18980	119	6932	93.2%	92.4%	17.3
7	2019Q3		32,094	19.19	0.32	2182	18636	95	5281	94.1%	92.1%	14.7
8	2019Q4		0,336	25.22	0.35	2480	23096	98	5382	93.9%	92.5%	12.2
9	2020Q1	Rev 3	47,421	13.58	0.32	2780	19241	137	8291	93.0%	90.7%	17.1
10	2020Q2		49,311	17.64	0.54	3667	25762	241	10721	92.1%	90.8%	13.4
11	2020Q3		64,702	11.29	0.38	3086	21298	221	7207	90.7%	90.3%	21.0
12	2020Q4	Rev 4	68,931	14.18	0.36	3792	28809	224	10572	91.7%	90.8%	18.2
13	2021Q1	Rev 4	73,221	8.51	0.31	3209	17774	193	7925	88.8%	86.2%	22.8
14	2021Q2		75,667	10.32	0.76	4228	22153	498	14914	88.3%	87.2%	17.9
15	2021Q3		87,621	9.03	0.48	3365	21908	364	19736	86.1%	87.2%	26.0
16	2021Q4		99,713	11.16	0.51	4009	33216	476	28215	92.8%	92.9%	24.9
17	2022Q1		99,611	8.30	0.76	4699	24152	692	28142	90.9%	72.8%	21.2
18	2022Q2		108,853	7.20	0.56	4679	22934	558	25224	91.0%	72.7%	23.3
19	2022Q3	Rev 5	117,938	8.67	0.72	5071	29256	756	32161	88.9%	70.7%	23.3
20	2022Q4		121,686	8.75	0.74	5872	31204	829	40517	91.2%	79.3%	20.7
21	2023Q1		117,903	5.20	0.70	4595	17592	749	29891	89.2%	91.3%	25.7
22	2023Q2	Rev 6	112,471	8.20	0.66	4987	27405	692	34001	92.4%	92.8%	22.6
23	2023Q3		121,201	6.20	0.72	5205	22227	809	40233	92.0%	93.0%	23.3
24	2023Q4		151,908	7.44	0.87	6350	33309	1238	56502	91.7%	93.6%	23.9
25	2024Q1		130,632	6.40	0.55	4872	24389	655	35650	90.7%	91.9%	26.8
26	2024Q2		95,582	7.52	0.62	4198	21661	565	26807	93.7%	95.3%	22.8

Ore Sources and Head Grade Changes

In 2017, the plant was being fed ore from the Irumafimpa orebody. By the beginning of 2018, the plant feed was mainly from the Kora K1 orebody with a small quantity of development ore from the K2 orebody. Since then, the ore supply from Kora has continued to increase. Judd started development in mid-2022 and by 2024 supplied about 30% of total ore deliveries.

As shown in Figure 13-24, the gold head grade has decreased with the change from Irumafimpa to Kora ores. At the same time, as shown in Figure 13-25, the copper head grade has increased. All ores from the various sources have responded well to processing with copper and gold recoveries generally over 90%. This is to be expected from the broad similarities between the nature and mode of occurrence of gold and copper values.

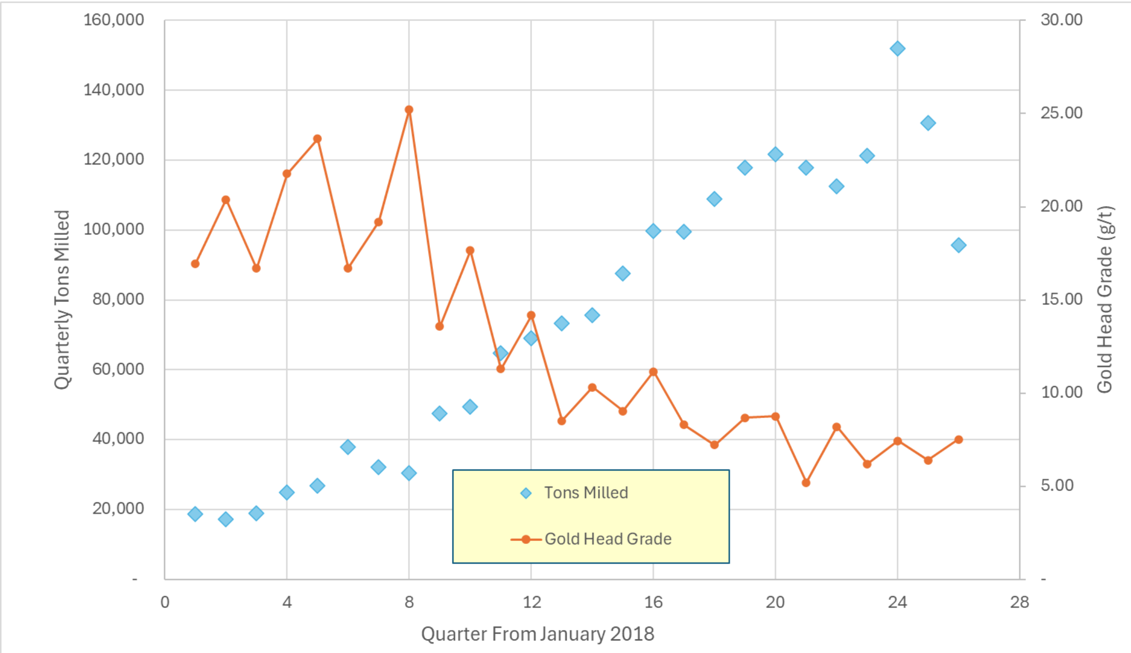


Figure 13-24 Tons Milled per Quarter and Gold Head Grade

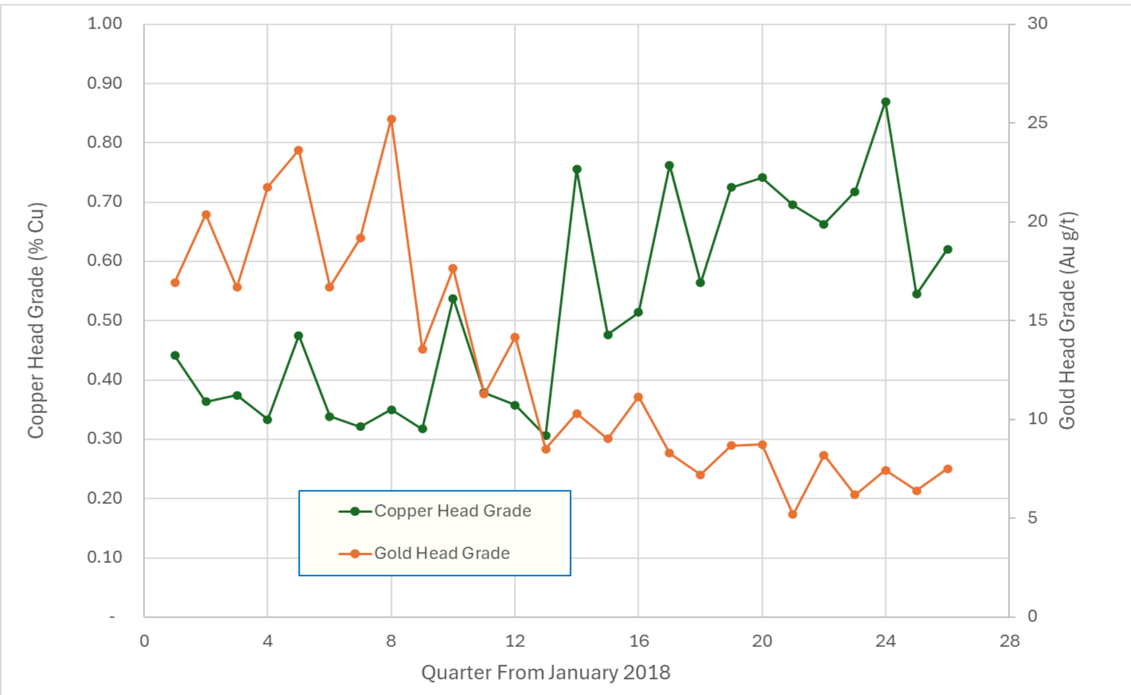


Figure 13-25 Quarterly Gold and Copper Head Grades

13.10.3 Performance Analysis: Recovery versus Concentration Ratio

Variable Operations and Recovery versus Concentration Ratio Considerations

The K92ML processing plant recovers metals values from the ore to a concentrate. Value recovery can be expressed as the percentage of the metals in concentrate compared to metals in the ore treated. The degree of concentration can be expressed as the concentration ratio (CR); a ratio between mass of ore treated and mass of concentrate produced.

Recovery of metals to concentrate becomes progressively more difficult as the concentration ratio is increased. A higher metals recovery at a given CR corresponds to improved performance. Similarly, lower recovery at that CR would correspond to reduced performance. It follows that plant performance must be judged by consideration of both metal recovery and concentration ratio.

In practice, for an operating concentrator, there are ongoing fluctuations in recovery and CR. These can be the result of changes in ore grade, and process management actions to achieve target concentrate grades or treat the required tonnage of ore. If there are no changes to the processing plant or ore type then when daily (or monthly, or quarterly) recovery points are plotted against CR, regression analysis can define an operating line.

There will be some scatter with points above and below the operating line. Points above or below the line respectively represent periods of higher or lower than average performance.

13.10.4 Effect of Plant Changes on Performance: Gold Recovery vs Concentration Ratio

At K92, the effect of staged modifications to the plant flowsheet on plant performance can be judged from Recovery vs CR plots for each revision.

In Figure 13-26, gold recovery data has been separately plotted for the plant revisions identified in Table 13-25. Results for the final revision, Rev 6, are amongst the best results over the life of the operation. This is a very positive achievement in view of the steadily increasing plant throughput.

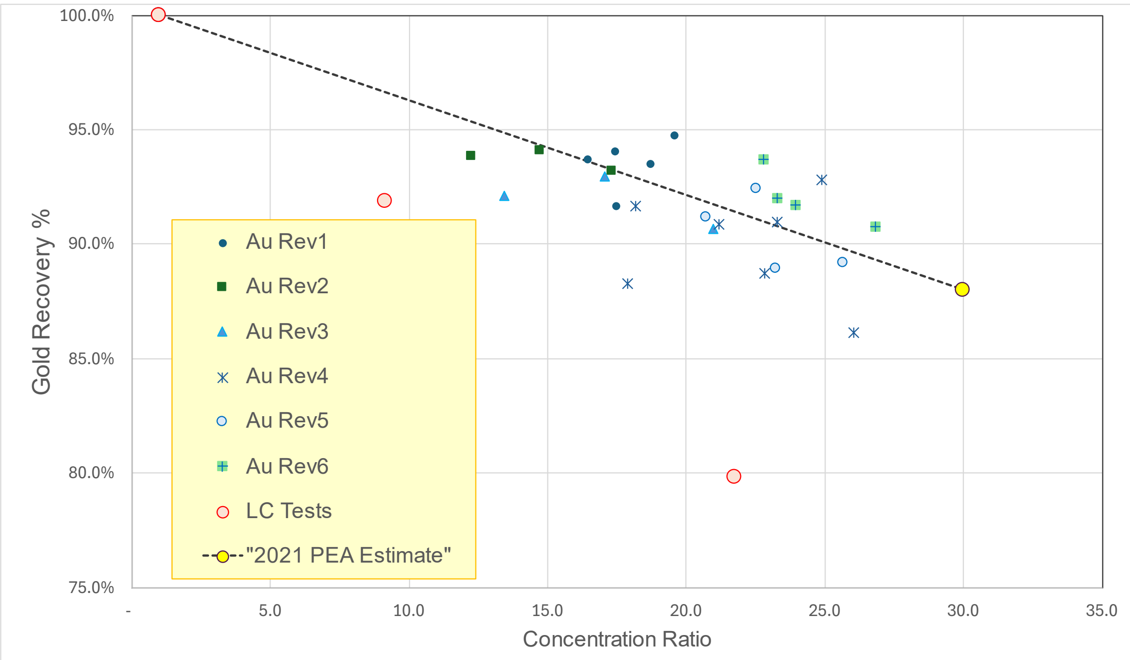


Figure 13-26 Gold Recovery versus Concentration Ratio; 2021 PEA Performance Recommendation, Plant Operations and the Locked Cycle Tests

Gold Recovery versus Concentration Ratio: Comparison of Quarterly Plant Operating Data with the Locked Cycle Testwork

Gold recoveries and concentration ratios from the quarterly plant data are shown with the results of the locked cycle tests in Figure 13-26. The two locked cycle test results have been extracted from Table 13-23. A third point, corresponding to 100% recovery when the concentration ratio is unity, has been added. A line drawn through these points would be an estimate of plant performance based on the locked cycle floats.

All quarterly gold recoveries plot above the locked cycle tests. This is to be expected as the plant incorporates both a flash float cell and a centrifugal concentrator for recovery of gold particles that are too coarse for conventional flotation. The improvement can be taken as coarse gold recovery by a combination of flash flotation and centrifugal concentration.

13.10.5 Gold Recovery Performance Prediction for the 2021 PEA

For the 2012 PEA, the processing plant was configured as "Rev4" and seven quarterly results were available. Performance predictions for future operations were based on the black dashed line (2021 PEA Estimate) in Figure 13-26. This line was from regression of the quarterly results with the intercept set so that the line passed through the point corresponding to 100% recovery at CR=1.

13.10.6 Effect of Plant Changes on Performance: Copper Recovery vs Concentration Ratio

The effect of staged modifications to the plant flowsheet on copper recovery can also be judged from Recovery vs CR plots for each revision. In Figure 13-27, copper recovery data has been separately plotted for the plant revisions identified in Table 13-25. In line with gold performance, results for the final revision, Rev 6, are amongst the best over the life of the operation.

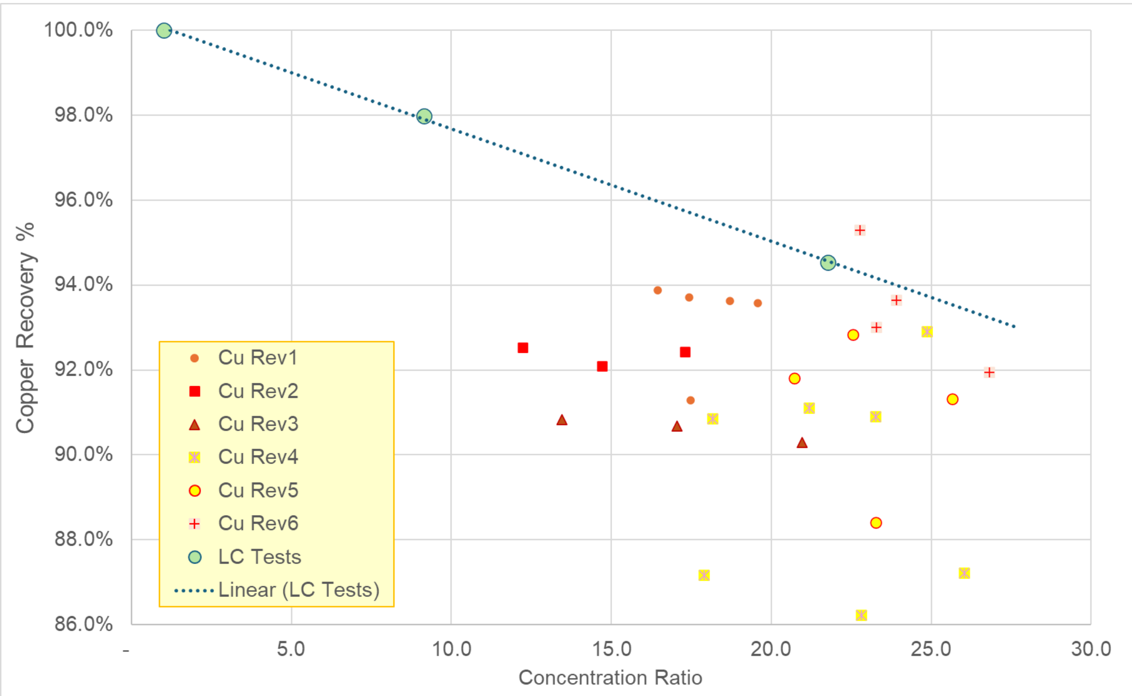


Figure 13-27 Copper Recovery, Quarterly Results and the Locked Cycle Tests

13.10.7 Copper Recovery Performance Prediction for the 2021 PEA

In Figure 13-27, the results of the two locked cycle tests are plotted with a point corresponding to 100% recovery at CR=1. The regression line (Linear LC Tests) is shown.

Quarterly operations results are also plotted. For the 2021 PEA, the processing plant was configured as "Rev4" and seven quarterly results were available. These are much closer to the regression line than was the case for gold results.

For the 2021 PEA Estimate, the locked cycle regression line was used to predict copper recovery performance.

13.11 Predicting Gold Recovery Performance for the 2024 IDP and DFS

13.11.1 Gold Recovery Performance Predictions Based on "Rev6" Plant Operating Data

A total of 400 days of daily data, covering the period 01 June 2023 to 18 August 2024, were available for the plant in "Rev6" configuration. The data included mill feed, flotation concentrate, and flotation tails analyses, plus the daily percentage (relative to mill feed) of gold recovered by gravity to doré metal bars.

The data was processed to calculate a float feed grade (i.e. not including gravity recovered gold) and an effective concentrate grade (i.e. including gravity recovered gold). The float feed, concentrate and tails values were used to calculate the concentration ratio. The effective concentrate grade, the mill feed grade, and the tailings grade were then used to calculate the overall gold recovery (float plus gravity).

Reconciled Quarterly data points (Recovery vs CR) for the same period were then plotted. This reconciled data is from the actual production figures declared by the Kainantu Gold Mine. The daily and quarterly data are directly comparable and are accurate reflections of plant performance. Both sets of data were then plotted as shown in Figure 13-28. A regression line was then plotted for the daily data.

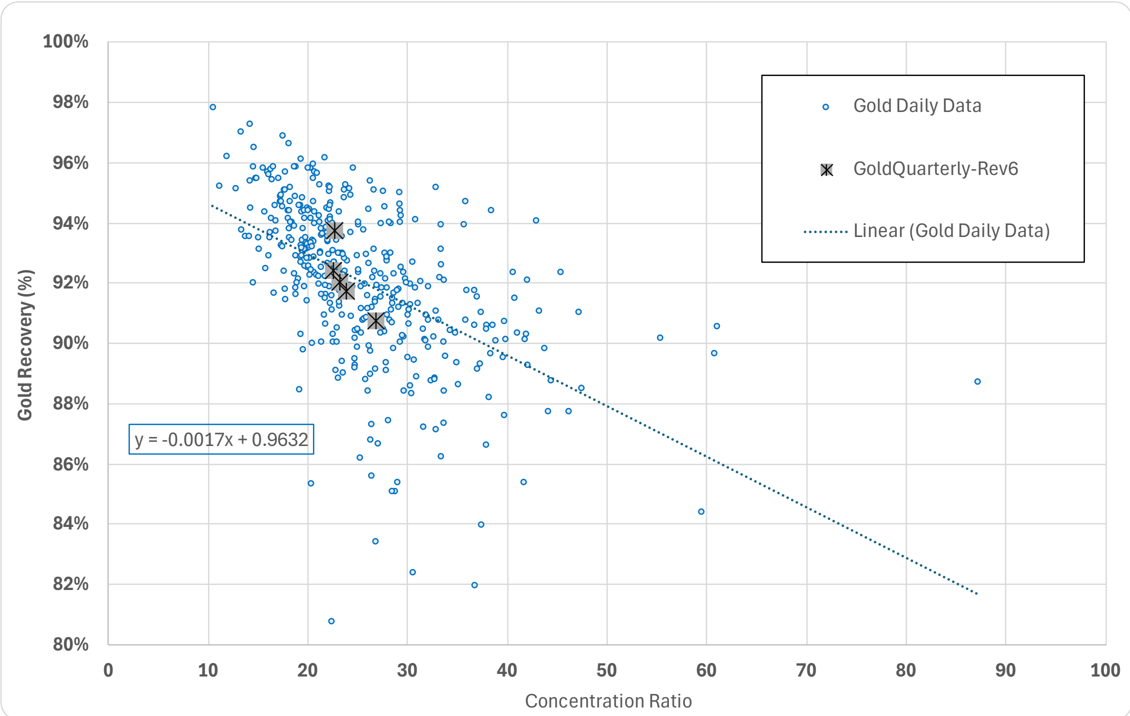


Figure 13-28 Gold Recovery, Daily and Quarterly Data for Plant Configured as "Rev6"

13.11.2 Recommended Relationship Between Gold Recovery and Concentration Ratio

For the 2024 updated IDP and DFS, MMS Pty Ltd recommends performance predictions based on the regression line, 'Linear (Gold Daily Data)" shown in Figure 13-28. This line is passes through the quarterly results and the slope of the line is a plausible estimate of plant performance. Whilst the slope is less than if the line had intersected 100% recovery at CR=1, this is likely to be realistic and could represent non-liberated gold values that do not respond to concentration. The daily data is somewhat scattered, but the regression line gives a better performance estimate than would be possible from the quarterly data.

This line corresponds to the equation:

Equation 4: Gold recovery (%) = 100 * ((-0.0017 * CR) + 0.9632)

At a concentration ratio of 20, the 2024 performance recommendation corresponds to a gold recovery of 92.92%.

Best estimates of future gold recovery performance are required for financial evaluation. The above recommendation is considered robust and valid for the existing plant. It is also valid, although possibly slightly conservative, for the new plant.

The new concentrator design will combine process knowledge from both the operation of the existing plant and the testwork program. It will also incorporate recent technological advances in process control and equipment design. Furthermore, the larger scale of the new concentrator, coupled with the high grade of the ore, has made it easier to justify on-line analysis of selected process streams. It follows that the new concentrator, whilst remaining an appropriate and cost-effective design, will at least match the best performance (in terms of gold recovery versus CR) of the existing concentrator.

With the plant controlled to achieve a target gold value in concentrate (e.g. target range 100 to 110 g/t), the predicted gold recovery will depend on the head grade of ore fed to the plant. Suitable calculations will translate the grade and tonnage of ore in the monthly mining schedule to gold contained in monthly concentrate production.

Comparison With the 2022 PEA Gold Recovery Performance Recommendation

In 2022, MMS Pty Ltd recommended performance predictions based on the dashed line, 'Linear (2021 PEA estimate)" shown in Figure 13-26. As noted, this was the regression line for the monthly gold recovery data for Q4 2021 and Q1 2022, that has been forced to go through to point for 100% recovery at CR =1.

The 2022 PEA Best Estimate line corresponded to the equation:

Equation 5: Gold recovery (%) = 100 * ((-0.0038 * CR) + 1.005)

At a concentration ratio of 20, the PEA performance estimate would be 92.90% gold recovery. This is quite close to the current estimate.

13.12 Predicting Copper Recovery Performance for the DFS

13.12.1 Copper Recovery Performance Predictions Based on "Rev6" Plant Operating Data

A total of 400 days of daily data, covering the period 01 June 2023 to 18 August 2024, were available for the plant in "Rev6" configuration. The data included mill feed, flotation concentrate, and flotation tails analyses. These analyses were used to calculate daily recoveries and concentration ratios.

Reconciled Quarterly data points (Recovery vs CR) for the same period were then plotted. This reconciled data is from the actual production figures declared by the Kainantu Gold Mine. The daily and quarterly data are directly comparable and are accurate reflections of plant performance.

Both sets of data are shown in Figure 13-29. A regression line was added for the daily data and MMS Pty Ltd recommends performance predictions based on this regression line.

This line corresponds to the equation:

Equation 6: Copper recovery (%) = 100 * ((-0.0021 * CR) + 0.9819)

With the plant concentration ratio (CR) controlled to achieve a target gold value in concentrate, both copper and gold recovery will depend on the CR, and hence on the gold head grade. Suitable calculations will translate the gold grade and tonnage of ore in the monthly mining schedule to copper contained in monthly concentrate production.

Taking the previous example; at a CR of 20, from Equation 6, the copper recovery will be 93.99%. If the copper head grade was 0.75% Cu, then the concentrate grade would be 14.10% Cu.

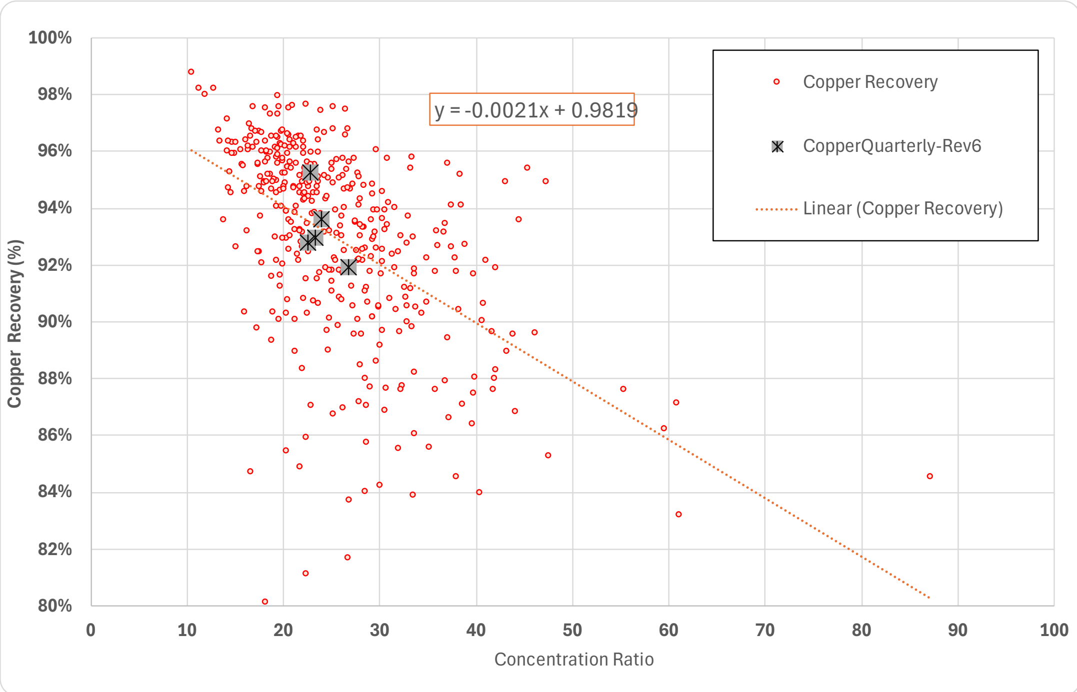


Figure 13-29 Copper Recovery, Daily and Quarterly Data for Plant Configured as "Rev6"

There may be some months where the ratio between copper head grade and gold head grade is significantly higher than average. At such times, the copper grade of the concentrate will rise to an achievable maximum whilst the concentrate gold grade is still below target. Owing to the predominantly chalcopyrite copper mineralization, the maximum grade of the concentrate will be about 25% Cu.

Copper Recovery versus Concentration Ratio: Comparisons with the 2021 PEA

Copper recoveries from quarterly operating data plus the results of the locked cycle tests are shown in Figure 13-27. The two locked cycle (cleaner flotation) results have been extracted from Table 13-23. A third point, corresponding to 100% recovery when the concentration ratio is unity, has been added. For the 2021 PEA, the line drawn through these points was used as an estimate of full-scale plant flotation performance based on the locked cycle test results.

The 2022 PEA Best Estimate line corresponded to the equation:

Equation 7: Copper recovery (%) = 100 * ((-0.0026 * CR) + 1.0031)

At a concentration ratio of 20, the PEA performance estimate would be 95.11% copper recovery. This is about a percentage point higher than the current estimate.

The copper recovery difference between plant and laboratory results is much smaller than for gold recovery. Copper recovery data has been separately plotted for the plant revisions identified in Table 13-25 and results for the most recent revision, Rev 6, are significantly better than older results. One of the Rev 6 results plots above this performance estimate and the other three points are just below the line.

13.13 Predicting Silver Recovery Performance for the DFS

In the testwork, silver results are very scattered and the relationship between concentration ratio and recovery is not clear. For financial evaluation, a silver recovery of 78% is recommended. This is the average silver recovery from January 2021 to June 2024.

13.14 Predicting Gold and Copper Recoveries Associated with the Mining Schedule

13.14.1 Plant Performance Prediction from Recovery vs CR Using the Three Product Formulas

The performance predictions for gold and copper recovery are equations relating recovery to concentration ratio. The equations have the form:

Equation 7: Recovery % = (X * CR) + Y, where X and Y are constants

In minerals processing, the recovery and CR can be calculated from the analysis values of the feed, concentrate and tailings using the well-known Three Product Formulas. These can be written:

Equation 8: Recovery = (c/f) * (f-t) / (c-t), and

Equation 9: CR = (c-t) / (f- t),

Where: f is the head-grade (e.g. g/t Au or % Cu), c is the effective concentrate grade and t is the tailings grade.

Substituting the values for Recovery and CR from Equations 8 and 9 into Equation 7 gives a quadratic equation.

Equation 10: (a * t2) + (b * t) + c = 0

Where, a, b, and c are constants and t is the tailings grade.

The quadratic equation can be solved using the quadratic formula from mathematics textbooks.

Equation 11: t = (-b ±(b2-4ac)) ^ 0.5

This gives two alternative values for the tailings grade, t, of which only one is plausible (the other is too high).

In the gold calculations the effective concentrate grade includes the gold recovered to gravity concentrate. If the gravity gold recovery is 15% then the actual flotation concentrate grade will be 85% of this effective grade. For copper calculations the effective concentrate grade and the actual concentrate grade are the same.

13.14.2 Concentrate Grade: Management Control and Maximum Copper Value

Management control of plant operations will aim to maximise the revenue from the terms of the offtake agreement.

The first factor to be considered will be the gold payability percentage that relates to the gold grade of the concentrate produced. The gold payability is constant for flotation concentrate grades above 120 g/t and decreases slightly for grades between 80 g/t and 120 g/t. However, payability decreases significantly when the concentrate grade falls below 80 g/t Au. Target minimum flotation concentrate grades down to as low as 80 g/t could be considered to maximise gold recovery.

Concentrate treatment and transport costs and their effects on net revenue must then be considered. Producing higher gold grades in concentrate adversely affects gold recoveries but may be necessary to maximise net revenue from ore with a low copper head-grade. Financial modelling suggests a target gold in concentrate grade of about 125 g/t Au and 6% copper in concentrate as the lower control limit. With low copper head-grades, the gold grade in the concentrate would be allowed to drift above the target grade of 125 g/t Au to maintain the minimum 6% Cu in concentrate.

The upper limit of the copper grade of the concentrate is dictated by the mineralogy of copper in the ore and the configuration of the plant. Most of the copper occurs as the mineral chalcopyrite (CuFeS2) which contains 34.6% Cu. Furthermore, chalcopyrite and pyrite are closely associated in the ore so some pyrite will be recovered with the chalcopyrite. Finally, with only two stages of cleaner flotation the concentrate will inevitably contain a small but significant proportion of siliceous gangue minerals. In view of these factors, a practical upper limit of the copper grade of the concentrate is likely to be around 25% Cu.

Depending on the ore head-grade for copper and gold, any one of these three factors could be controlling. When the copper head grade is high relative to the gold head-grade, the plant will produce concentrate with 25% Cu and less than 125 g/t Au. When the gold and copper head-grades permit, the plant will produce concentrate with around 125 g/t Au and copper content between 6% and 25% Cu. When the copper head grade is low relative to the gold head-grade then the plant will produce concentrate with 6% Cu and higher than 125 g/t Au.

13.15 Calculations Investigating the Effects of Varying Copper and Gold Head-Grades

An Excel spreadsheet was developed for recovery calculations based on Recovery versus Concentration Ratio relationships. The recommended gold and copper recovery relationships are given in Equation 4, and Equation 6 respectively.

A series of monthly periods with fixed copper head-grade 0.5% Cu and gold head-grade ranging from 4 g/t Au to 15 g/t Au was considered first. This was followed by consideration of a series of periods with fixed gold head grade of 6 g/t Au, and copper head-grade ranging from 0.2% Cu to 2.2% Cu. These ranges were selected based on the range of ore variability shown in the 2024 Mining Schedule.

For each month, three parallel calculations were done, each estimating metals recoveries and flotation concentrate grades for a specific plant control objective. The first calculation was based on controlling the flotation concentrate grade to 125 g/t Au. The second was based on controlling the concentrate grade to 6% Cu, and the third was based on controlling the concentrate grade to 25% Cu. For each month, one of the three calculations was then selected to limit the copper grade of the concentrate to the range of 6% Cu to 25% Cu.

The output of the calculations were the tailings grades, concentrate grades, metals recoveries and the Concentration Ratio. A monthly concentrate tonnage was estimated based on the existing plant operating at a throughput of 41,667 tons per month (500k tpa). The results of the calculations for constant copper head-grade are shown below in Table 13-27. Results for constant gold head-grade are shown in Table 13-28.

Table 13-27 Constant Copper Head Grade, Gold Head Grade Varies

Gold Head Grade	Copper Head Grade	Conc Gold Grade	Tailings Grade	Conc Copper Grade	Gold Recovery	Copper Recovery	Conc Ratio	Conc Produced
(g/t Au)	(% Cu)	(g/t Au)	(g/t Au)	(% Cu)	%	%		t/m
4	0.50%	125	0.440	18.44%	89.28%	89.54%	41.2	1,012
5	0.50%	125	0.476	14.80%	90.78%	91.39%	32.4	1,286
6	0.50%	125	0.515	12.37%	91.75%	92.58%	26.7	1,560
7	0.50%	125	0.555	10.62%	92.43%	93.42%	22.7	1,833
8	0.50%	125	0.596	9.30%	92.93%	94.04%	19.8	2,106
9	0.50%	125	0.638	8.27%	93.32%	94.51%	17.5	2,380
10	0.50%	125	0.681	7.45%	93.62%	94.89%	15.7	2,653
11	0.50%	125	0.725	6.78%	93.87%	95.20%	14.2	2,926
12	0.50%	125	0.770	6.22%	94.08%	95.45%	13.0	3,199
13	0.50%	131	0.822	6.00%	94.18%	95.53%	12.6	3,317
14	0.50%	141	0.885	6.00%	94.18%	95.53%	12.6	3,317
15	0.50%	151	0.949	6.00%	94.18%	95.53%	12.6	3,317

Table 13-28 Constant Gold Head Grade, Copper Head Grade Varies

Gold Head Grade	Copper Head Grade	Conc Gold Grade	Tailings Grade	Conc Copper Grade	Gold Recovery	Copper Recovery	Conc Ratio	Conc Produced
(g/t Au)	(% Cu)	(g/t Au)	(% Cu)	(% Cu)	%	%		t/m
6	0.20%	152	0.0181%	6.00%	90.73%	91.24%	32.9	1,267
6	0.25%	125	0.0193%	6.18%	91.75%	92.58%	26.7	1,560
6	0.40%	125	0.0308%	9.89%	91.75%	92.58%	26.7	1,560
6	0.60%	125	0.0463%	14.84%	91.75%	92.58%	26.7	1,560
6	0.80%	125	0.0617%	19.79%	91.75%	92.58%	26.7	1,560

Gold Head Grade	Copper Head Grade	Conc Gold Grade	Tailings Grade	Conc Copper Grade	Gold Recovery	Copper Recovery	Conc Ratio	Conc Produced
6	1.00%	125	0.0771%	24.73%	91.75%	92.58%	26.7	1,560
6	1.20%	105	0.0819%	25.00%	92.53%	93.48%	22.3	1,870
6	1.40%	90	0.0860%	25.00%	93.10%	94.18%	19.0	2,198
6	1.60%	79	0.0903%	25.00%	93.52%	94.70%	16.5	2,525
6	1.80%	70	0.0947%	25.00%	93.84%	95.10%	14.6	2,853
6	2.00%	63	0.0992%	25.00%	94.09%	95.42%	13.1	3,181
6	2.20%	57	0.1039%	25.00%	94.30%	95.67%	11.9	3,508

In the tabulated values, the tailings grade varies less than the head-grade. This is a result of increased metals recoveries associated with increased head-grade.

The variations in concentrate grades and metals recoveries as the gold and copper head-grades change are shown in Figure 13-30, and Figure 13-31 respectively. In both graphs, there is a range of the variable head-grade where the concentrate gold grade is constant. This interval is where the calculation based on controlling the gold grade in concentrate has been selected.

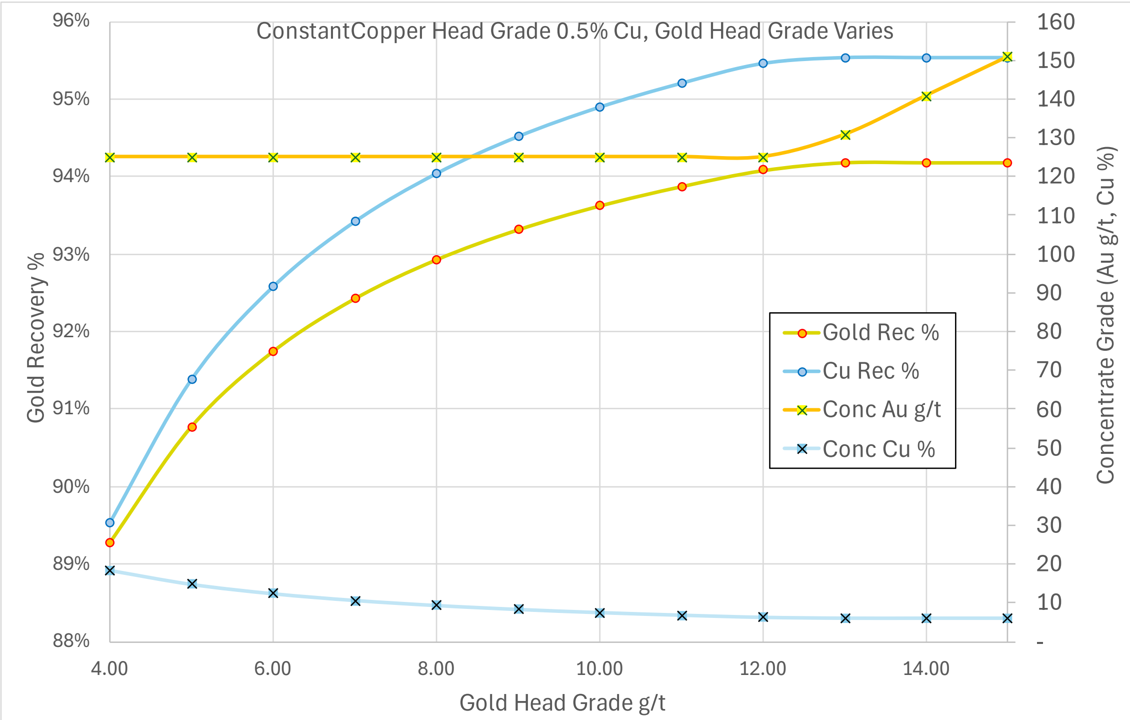


Figure 13-30 Metals Recoveries and Concentrate Grades for Constant Copper Head-grade

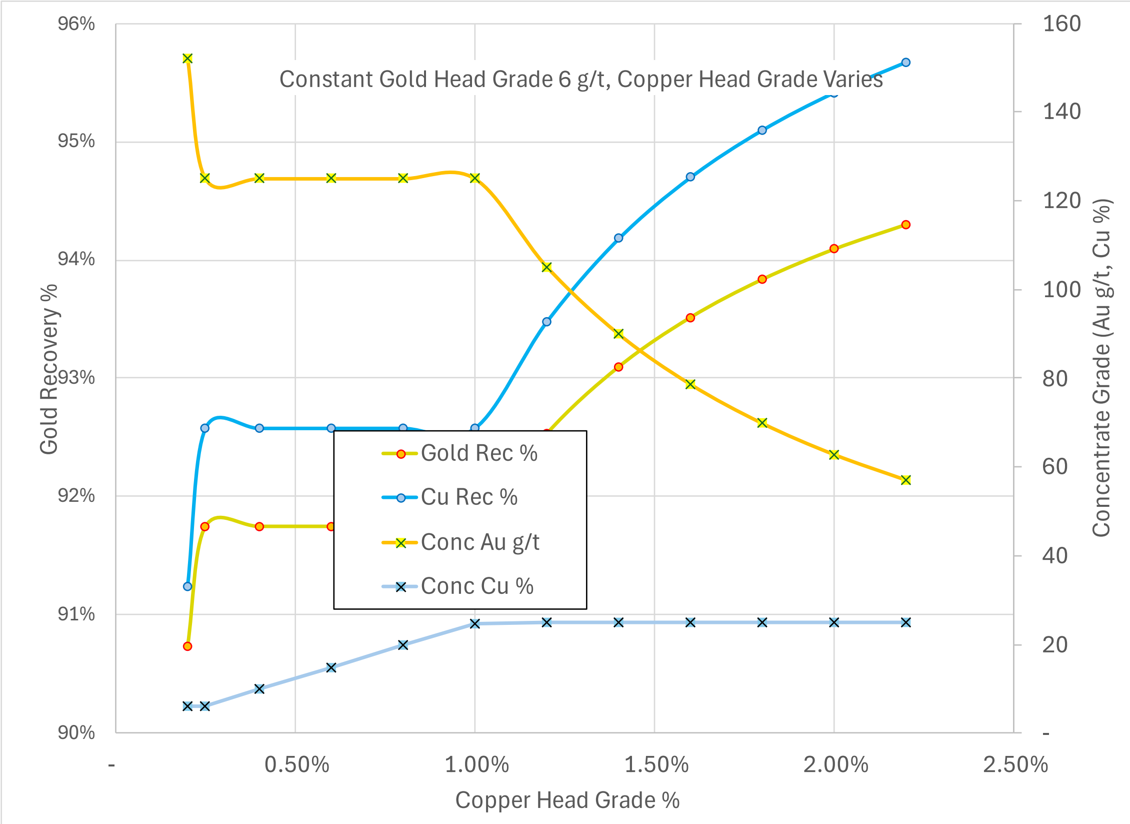


Figure 13-31 Metals Recoveries and Concentrate Grades for Constant Gold Head-grade

13.15.1 The Mining Schedule

The Mining Schedule is a monthly listing of the estimated tonnages of waste and ore mined. For the ore, estimates of the gold, copper and silver grades are also given. The 2024 Mining Schedule covers 167 months, from January 2024 to November 2037 (13 years and 11 months). An excerpt from the mining schedule with the first three months of data is shown below in Table 13-29. Total tonnages and average grades for the full 167 month period are also shown.

Table 13-29 An Excerpt from the 2024 Mining Schedule Showing Total and Average Data for 167 Months and the First Three Months of Data

Unit	Total	Description	Jan-24	Feb-24	Mar-24
t	8,516,461	Waste	49,962	45,543	19,068
t	3,858,368	Dev Ore (t)	10,698	11,298	2,489
t	16,537,644	Stope Ore (t)	28,646	35,241	22,358
t	20,396,012	Ore (t)	39,344	46,540	24,847
		Grade			
g/t	8.1	AuEq (g/t)	8.2	7.7	5.8
g/t	6.0	Au (g/t)	6.7	6.3	4.6
%	1.2	Cu (%)	0.8	0.7	0.6
g/t	23.1	Ag (g/t)	11.6	10.6	9.1
		Metal			
oz	3,908,952	Au (oz)	8,428	9,486	3,694
t	244,422	Cu (t)	326	317	154
oz	15,127,014	Ag (oz)	14,644	15,803	7,255

Over the 165 month period, there is a wide variation in head-grade as shown below in Table 13-30.

Table 13-30 Range of Head-Grade Variation over Mining Schedule

Grade	Average	Minimum	Maximum
AuEq (g/t)	8.21	4.66	15.14
Au (g/t)	6.20	1.69	15.78
Cu (%)	1.14	0.26	1.95
Ag (g/t)	22.11	7.15	49.98

13.15.2 Recovery and Concentration Ratio Estimates based on the Mining Schedule

As described above, the performance recommendations relating recovery to concentration ratio can be used to estimate gold and copper recovery in the plant. The gold and copper head-grade values in the mining schedule are the most important variables. Plant management control actions driven by the Offtake Agreement are then considered. The controlled parameters are the target minimum gold and copper grades in concentrate. The practical upper limit of copper grade in concentrate can be reached when treating ore with high copper head-grade.

Spreadsheet calculations have been applied to the monthly ore head-grades in the 2024 Mining Schedule. The recommended relationships between recovery and concentration ratio have been applied and monthly Recovery versus CR estimates are plotted in Figure 13-32. High head-grade months are at the upper left portion of the graph whilst low head-grade points are on the lower right.

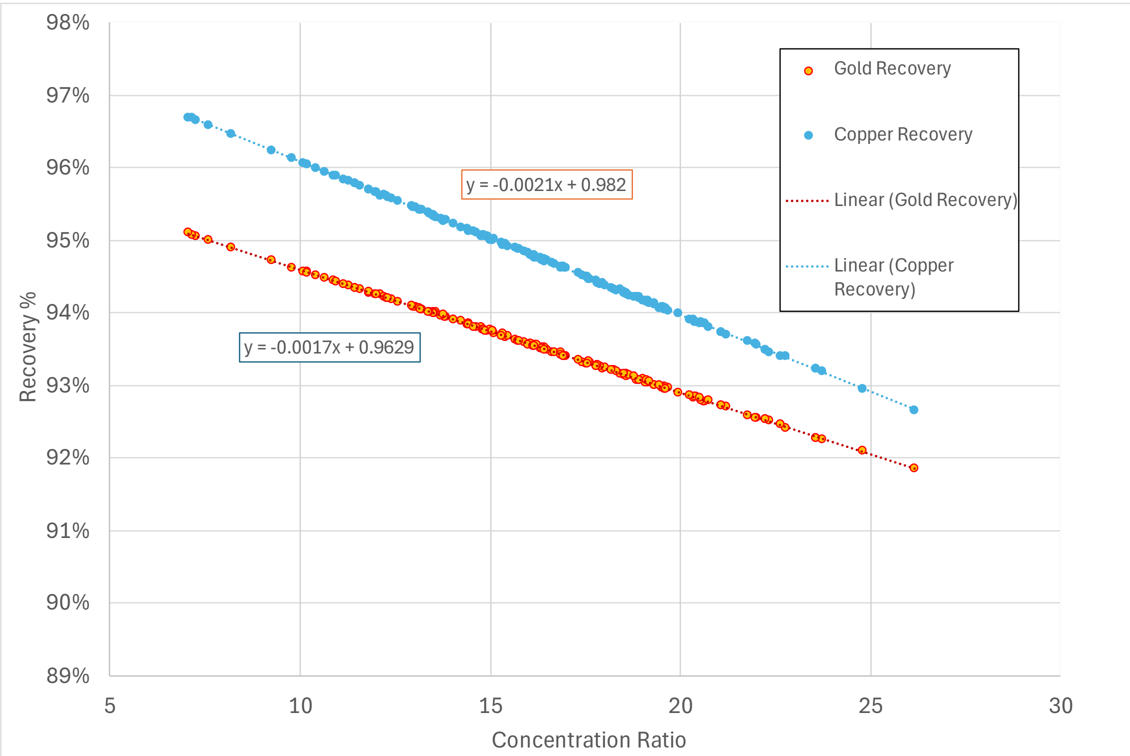


Figure 13-32 Gold and Copper Recovery vs CR Estimates for the 2024 Mining Schedule

The wide range of variability in ore head-grades over the duration of the mining schedule is shown in Figure 13-33. The production estimates based on the relationships between Recovery and CR are robust estimates of production parameters, even with this level of variability.

13.15.3 Production Estimates

The spreadsheet calculations can be used to generate monthly gold and copper production estimates from the mining schedule. Figure 13-34 shows the estimated metals recovered monthly (ounces of gold and tons of copper) by the processing plants.

Other production parameters are also estimated. Figure 13-35 shows the monthly gold and copper recovery percentages, whilst Figure 13-36 shows monthly concentrate grade estimates.

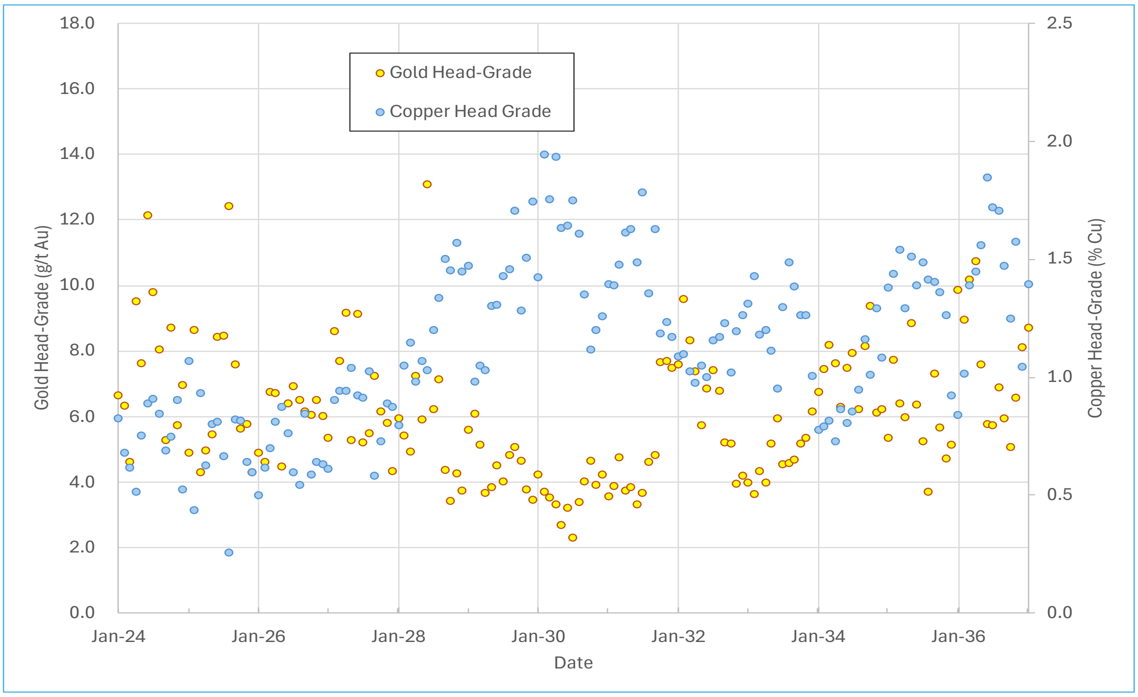


Figure 13-33 Gold and Copper Head-Grades in the 2024 Mining Schedule

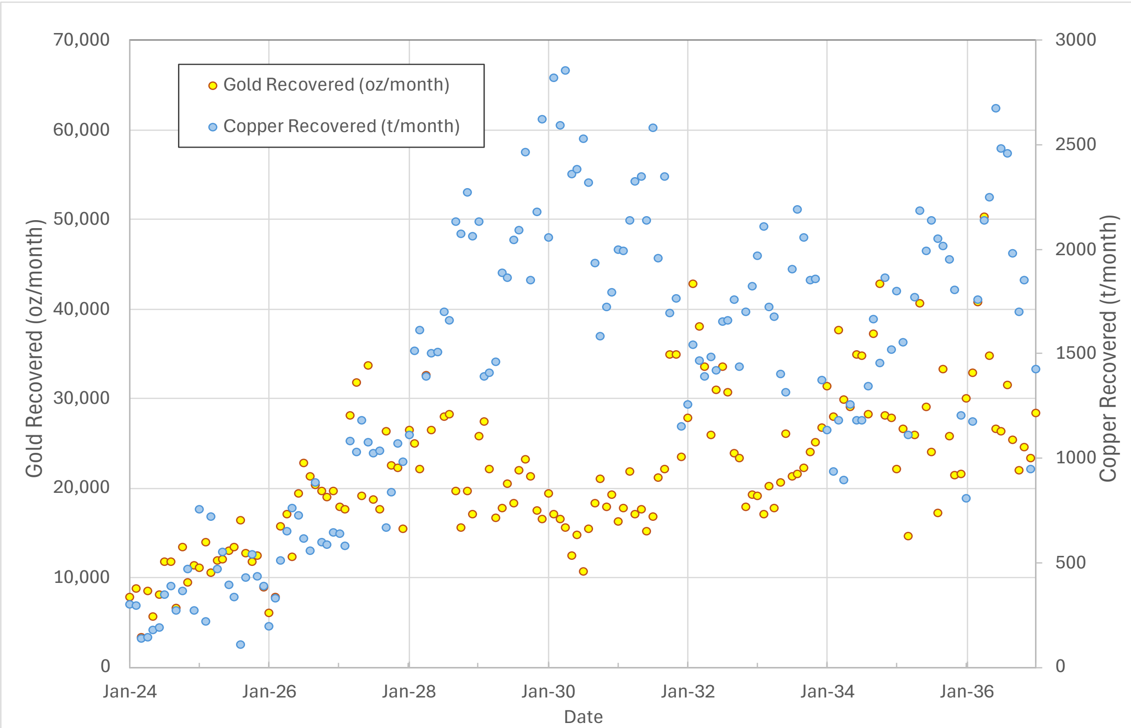


Figure 13-34 Monthly Production Estimates from the Mining Schedule

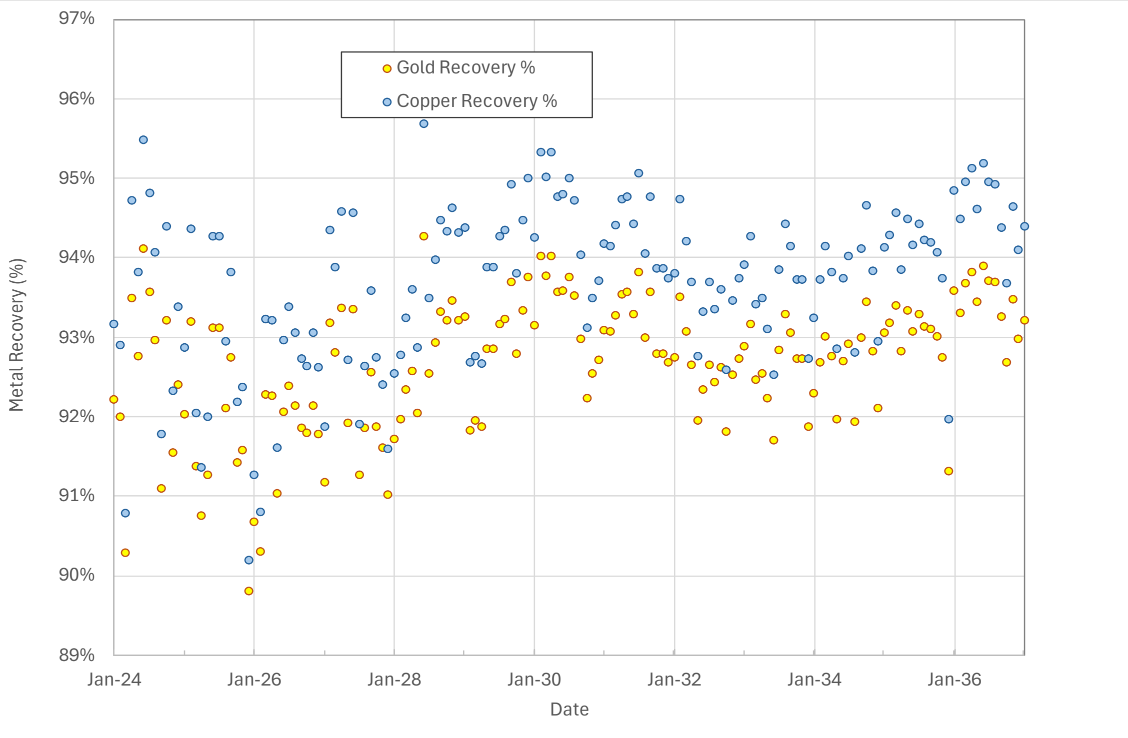


Figure 13-35 Gold and Copper Percentage Recovery Estimates for the Mining Schedule

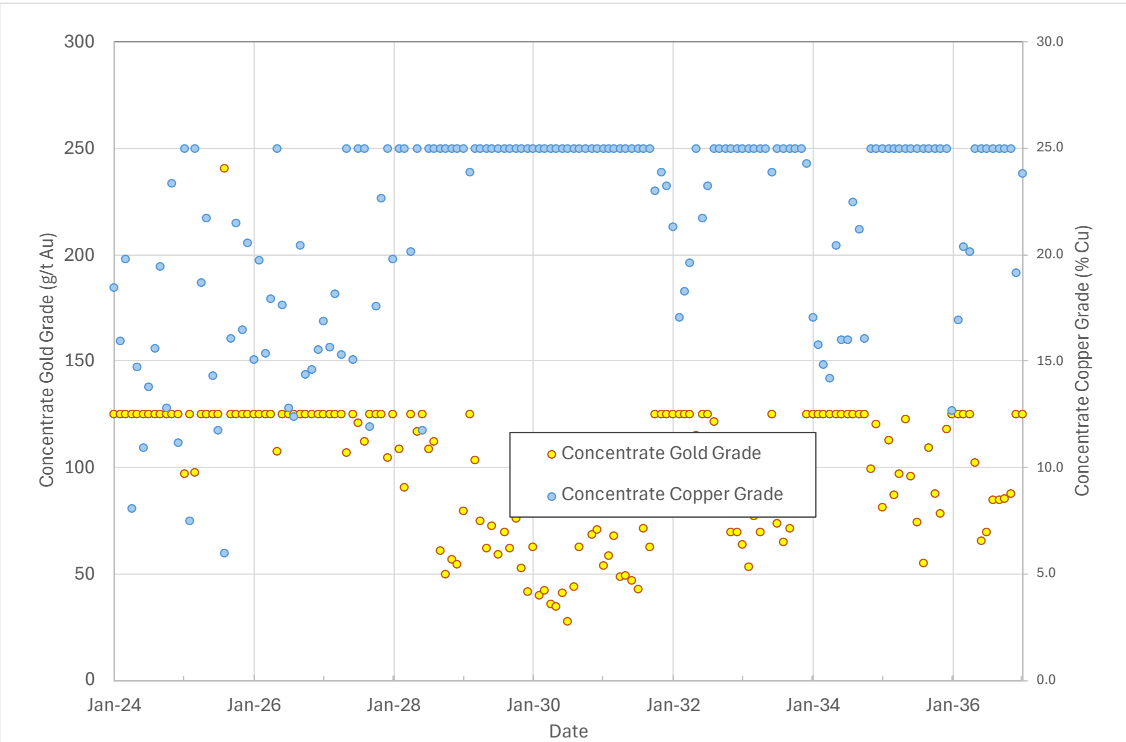


Figure 13-36 Concentrate Grade Estimates for the Mining Schedule

13.16 Paste Laboratory Testwork

13.16.1 Introduction

This section presents the key results of the laboratory testwork undertaken on materials used to establish the baseline for the study to inform the paste plant process design. This test program is meant to build on and complement the previous work completed by others [Lycopodium, 2023]. Testwork presented include material characterization, rheology assessment, paste flow loop analysis, dewatering (thickening and filtration) and unconfined compressive strength (UCS) testing of cured paste cylinders. All testwork was undertaken in accordance with relevant North American standards, where applicable or using well established standard laboratory procedures developed by WSP. Tailings, binder and water samples used in this laboratory campaign were all supplied by K92.

13.16.2 Characterization

This section sets out characterization information related to the tailings and water used in the testwork. All samples received by WSP are subjected to characterization tests to establish material properties and allow for comparison with both past and future samples.

Tailings Particle Size Distribution (PSD)

Particle size distribution (PSD) was determined using mechanical sieving and a Malvern Mastersizer 3000 laser particle size analyzer in accordance with ASTM D4464. Specific values are present in Table 13-31 The gradation parameter DXX, tabulated in microns, refers to the average particle diameter that XX% by weight of material is smaller than. Figure 13-37 presents the results. A PSD test was also performed after the flow loop testing was complete to confirm that the PSD did not change significantly as some materials particles do break down.

Previous test work (Lycopodium, 2023) illustrated a coarser tailings split with approximately 35% passing 20 µm. The tailings PSD is typical of gold tailings.

Table 13-31 Particle Size Distribution

Sample	D_{10} (µm)	D_{30} (µm)	D_{50} (µm)	D_{60} (µm)	D_{80} (µm)
CA0012623.1156 Tailings	4	12	30	46	110
CA0012623.1156 Tailings - after flowloop	3	11	26	41	103

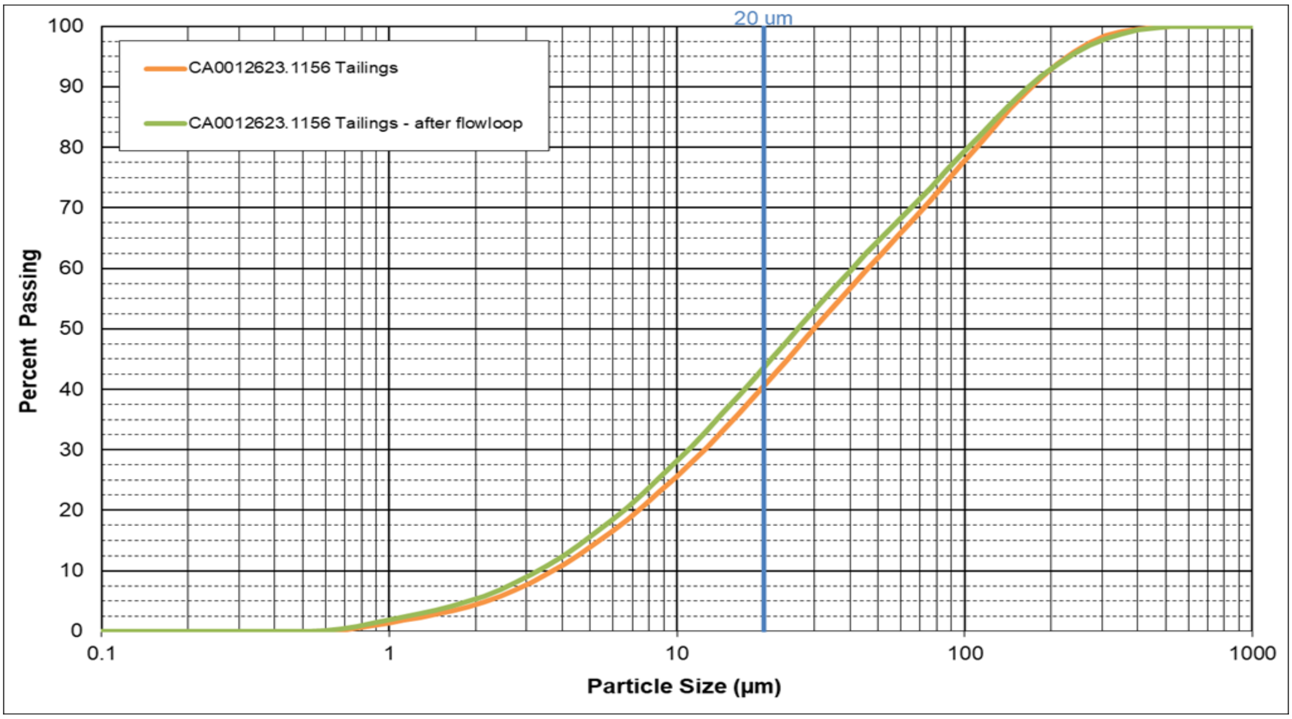


Figure 13-37 PSD of tailings sample used in this laboratory test program

13.16.3 Tailings Characterisation

To characterise the tailings, chemical and mineralogical analyses were performed using whole rock analysis (WRA) by inductively coupled plasma (ICP) and X-ray diffraction (XRD) via semi-quantitative analysis by Rietveld Method, respectively. Whereas sulphur analysis was performed by LECO method.

The mineralogy results are presented are presented in Table 13-32. By comparison to the previous test work (Lycopodium, 2023), the tailings testing contained similar amounts of Quartz, less Muscovite, but more Pyrite and Chlorite. The sulphur content on the current tailings is about 5.5 wt% per the LECO analysis.

Table 13-32 Semi-quantitative Mineralogical Composition - CA0012623.1156 Tailings

Mineral SQ-XRD	Chemical Formula	Wt%
Quartz	SiO_2	54.18
Muscovite	$KAl_2(Si,Al)_4O_{10}(OH)_2$	31.40
Albite	$Na(AlSi_3O_8)$	5.82
Clinochlore	$(Mg,Fe)_6(Si,Al)_4O_{10}(OH)_8$	4.33
Pyrite	FeS_2	4.27

Proctor Testing

Proctor testing was performed following ASTM D698 Compaction Characteristics of Soil Using Standard Effort and ASTM D4718 Correction for Oversize Particles. The optimum geotechnical moisture content (water / solids by mass) for these tailings is about 12.1%, which translates to 10.7% metallurgical moisture content (water / water+solids by mass). Typically, a filtered tailings, which can be readily transported, placed and compacted is within 2% of optimum moisture content. As a result, the target filter cake moisture content was set at 12 wt% (metallurgical).

Note, unless otherwise indicated, metallurgical moisture content is used in this section of the report.

Transportable Moisture Limit (TML) Testing

To further examine the transportability of the filtered tailings, as the filtered tailings is produced at the mill area and transported to a higher elevation for over 6 km by truck, a TML test was completed. This confirmed that a 12 wt% filtered tailings was indeed a suitable target. Further testing on K92's filtered tailings may indicate that a broader TML range could be applicable.

TML testing was performed by Bureau Veritas Laboratory. Table 13-33 present the results.

Table 13-33 TML Results

Sample	TML (wt%)	FMP (wt%)
CA0012623.1156 Tailings	12.5	13.9

TML - Transportable Moisture Limit

FMP - Flow Moisture Point

Tailings Specific Gravity

The specific gravity (SG) of the tailings sample was determined using vacuum de-aerated water. Each slurry samples was also vacuum de-aerated prior to SG measurement. The results are presented in Table 13-34.

Table 13-34 Specific Gravity of Tailings used in Testwork

Sample	Trial 1	Trial 2	Average
CA0012623.1156 Tailings	2.91	2.91	2.91

13.16.4 Mix Water Characterization - Water Quality Analysis (WQA) Testing

WQA testing was performed by AGAT Laboratories on representative samples of CA0012623.1156 800 Portal Water, CA0012623.1156 Twin Portal 800 Water, and CA0012623.1156 CRK Confluence Past 800 Power House (Upstream) Water.

Based off these WQA results, it was concluded that the CA0012623.1156 Twin Portal 800 Water would provide optimal strength results for cemented backfill. Where CA0012623.1156 800 Portal Water may not perform as well for use in cemented backfill. UCS testing was conducted which compared these different water sources. The results and summary can be found in Section 13.16.7 of this report.

13.16.5 Tailings Filtration Testwork

Vacuum Filtration

Dewatering tests were performed on the CA0012623.1156 Tailings using vacuum and pressure filtration techniques. The stability of the filter cake is typically assessed via Proctor testing results which is normally more conservative than the value provided via TML testing. Vacuum filtration testing is used to compare the filterability of the tailings received with previous results [Lycopodium 2023] prior to advancing to pressure filtration. In the author's experience, a stable filter cake cannot be achieved with vacuum disc filtration technology and only occasionally with vacuum belt filtration. With the Proctor test results and TML results converging to a filter cake solids of 87.5 to 89 wt%, it is believed that pressure filtration remains the most viable option for dewatering the K92ML tailings.

Vacuum Filtration Testing were conducted with a range of feed consistencies: 64, 59 and 54 wt% solids.

The filter leaf was equipped with a small section of industrial grade polypropylene felt filter cloth. The leaf was immersed into the slurry and simulated production scale vacuum filters where the sectors dipped into the slurry in an agitated filter tank as the disc rotated. Proper technique and cycle times simulating continuous filters provide an estimate of cake loading, moisture and discharge characteristics.

Since the test was performed in the laboratory, under ideal conditions, actual cake loading as presented in the following tables is derated to reflect variable or upset conditions which may occur in plant operations.

An example of the vacuum filtration test results for the 59 wt% feed solids is presented in Table 13-35 with corresponding Figure 13-38. Figure 13-39 illustrates the cake loading time at the various cycle times. The cake loading rate is considered low in comparison to other tailings types.

Table 13-35 Filtration Results - 59 wt% Solids

Vacuum Level (mm Hg)	Form Time (sec)	Dry Time (sec)	Cycle Time (sec)	Cake Thickness (mm)	Cake Loading (kg/m^2/hr)	Cake Moisture (wt%)	Final Density (wt% Solids)
599	10	12	30	3.8	450	18.7	81.3
599	15	18	45	4.8	368	19.0	81.0
599	30	36	90	5.7	218	19.0	81.0
599	45	54	135	6.4	193	19.5	80.5
599	60	72	180	7.7	179	18.5	81.5
508	30	36	90	5.4	224	19.5	80.5

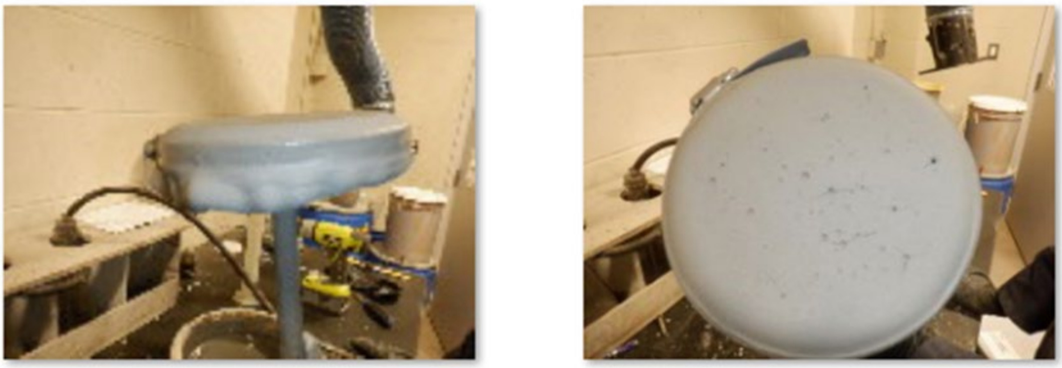


Figure 13-38 Filter Cake 59 wt% Solids Feed, 90 second Cycle Time

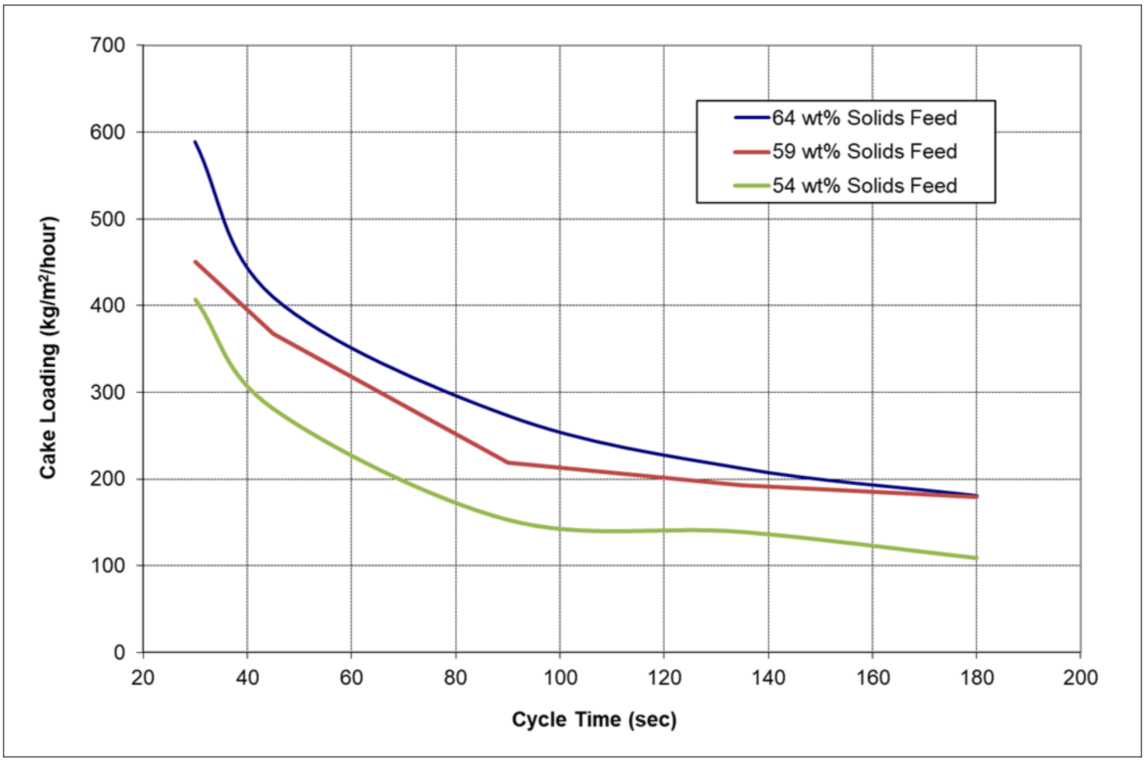


Figure 13-39 Cake Loading vs. Cycle Time

Pressure Filtration Testing

Pressure filtration tests were conducted using a bench-scale testing unit at a slurry feed content of approximately 59 wt% solids. The testing assessed different filter cloths and the effect of adding on drying time using compressed air. Slurry feed pressures of 10 and 15 bar were tested as well as three chamber/spacer thicknesses of 25 mm, 32 mm, and 50 mm.

Based off this series of testing, the optimal pressure filtration results were produced using a 20-30 cfm filter media, 15 bar feed pressure with a 2-to-3-minute form time and in a recessed 32 mm chamber. The dry time can be varied to obtain the target cake moisture content.

The results of the 32 mm chamber tests with a 15-bar feed pressure are presented in Figure 13-40 to Figure 13-43. An additional test with the slurry adjusted to a pH of 10 was completed to mimic the process conditions. The dry time required for the filter cake to reach the target moisture content of 12 wt% can be achieved within 5 to 6 minutes.

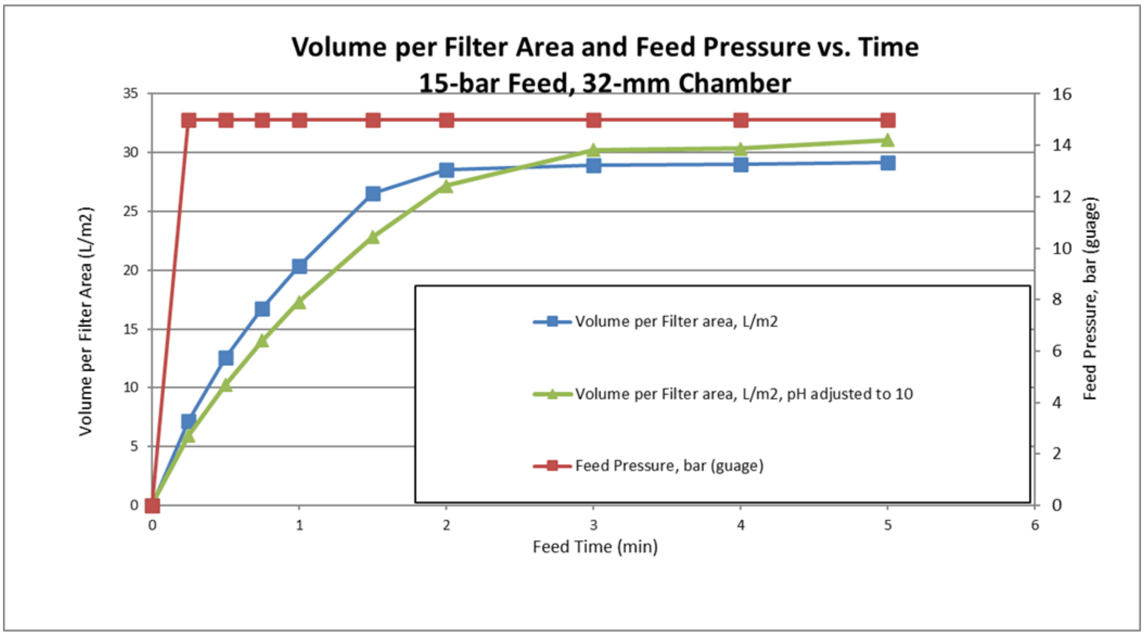


Figure 13-40 Volume per Filter Area and Feed Pressure vs. Time - 15-bar Feed, 32-mm Chamber

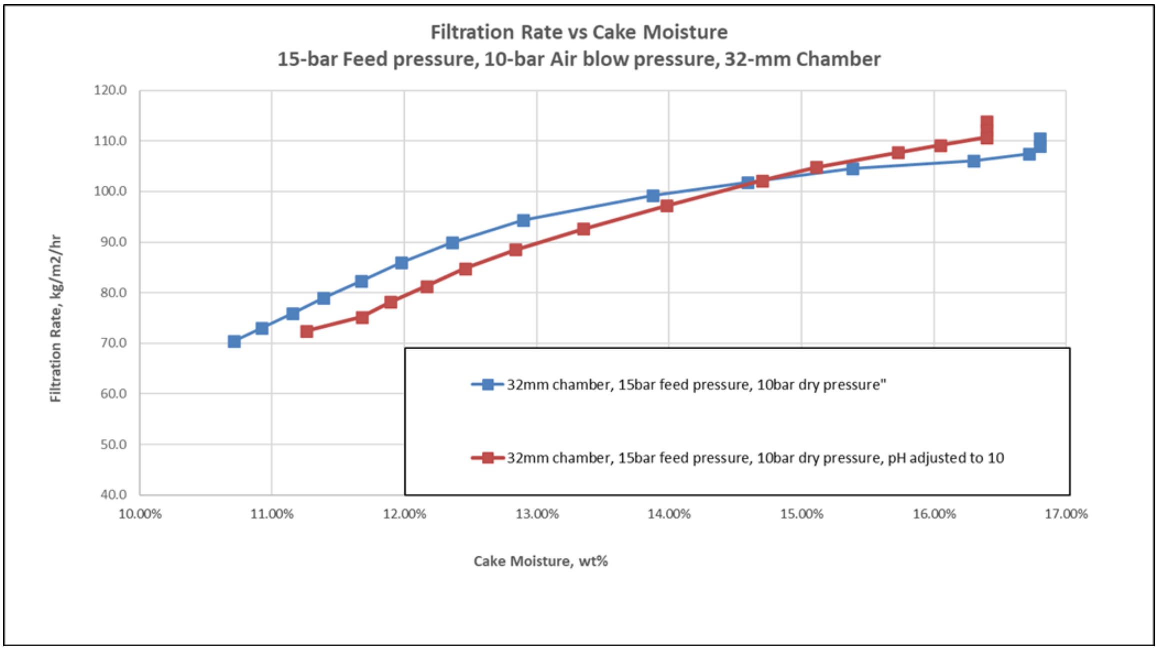


Figure 13-41 Filtration Rate vs. Cake Moisture - 15-bar Feed, 7-bar Air Blow, 32-mm Chamber

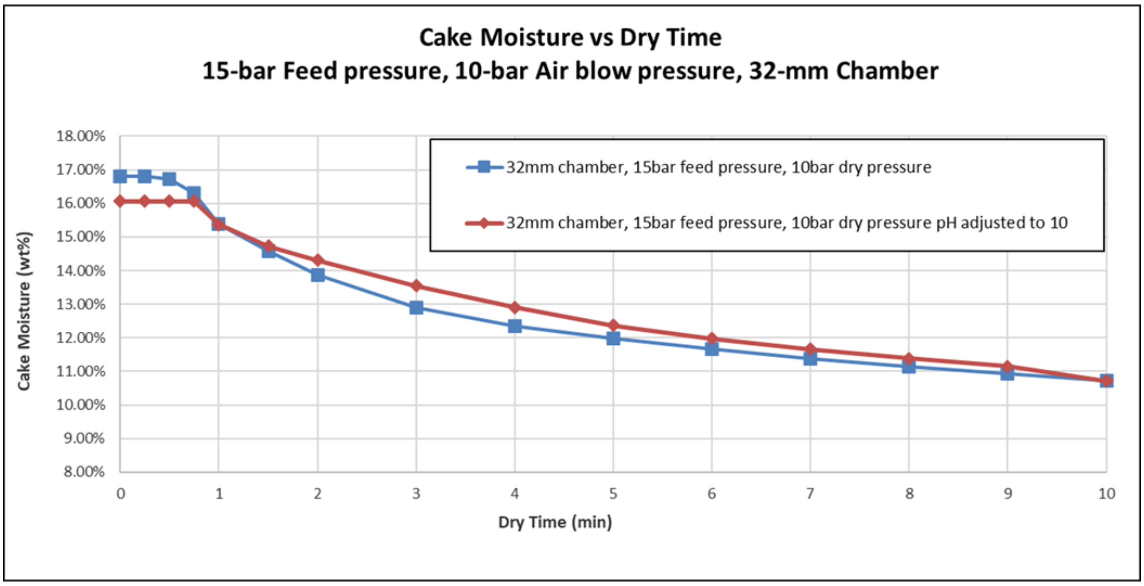


Figure 13-42 Cake Moisture vs Dry Time - 15-bar Feed, 7-bar Air Blow, 32-mm Chamber

Figure 13-43 Filtered Cake 15-bar Feed, 7-bar Air Blow, 32-mm Chamber, final filtered cake moisture content = 11.8 wt%

13.16.6 Rheology Testing

Rheological testing was carried out to evaluate flow and handling properties. These tests provide an indication of the tailings behaviour while mixing, slump adjustment, pipeline transport and also while sitting idle. Rheological characterization provides data for the selection of process equipment such as mixers, pumps, and pipelines.

Slump vs. Solids Content

To gauge sensitivity to water additions, small increments of water were added to the bulk sample. After each addition, slump and solids content was determined. This generates a relationship between slump and solids content which is typically used to determine the degree of process control required to maintain slump control of the final product. The results are presented on Figure 13-44.

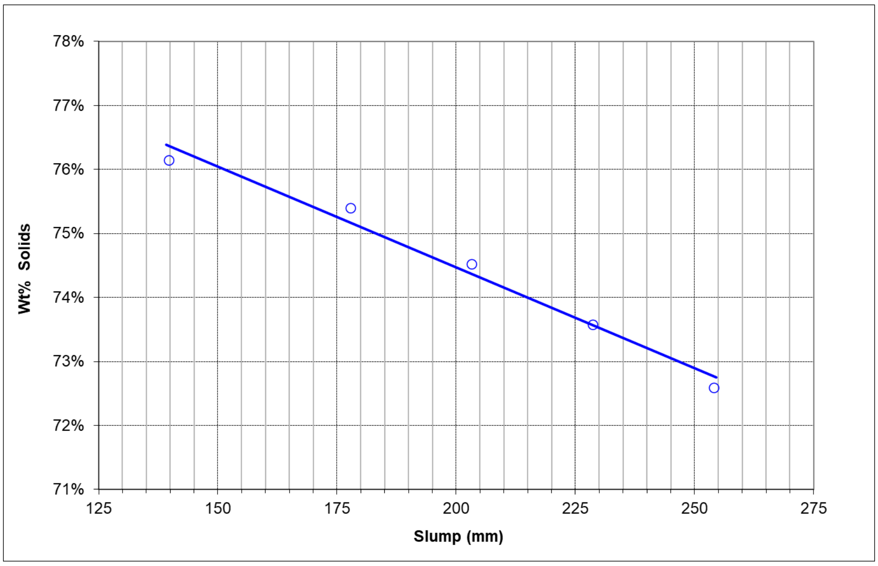


Figure 13-44 Solids Content vs. Slump

Static Yield Stress Testing

Yield stress is defined as the minimum force required to initiate flow. Static yield stress was determined by using a very slow moving (0.2 RPM) vane spindle attached to a torque spring. The spindle was immersed in the sample and measurements were taken at various solids contents (Figure 13-45).

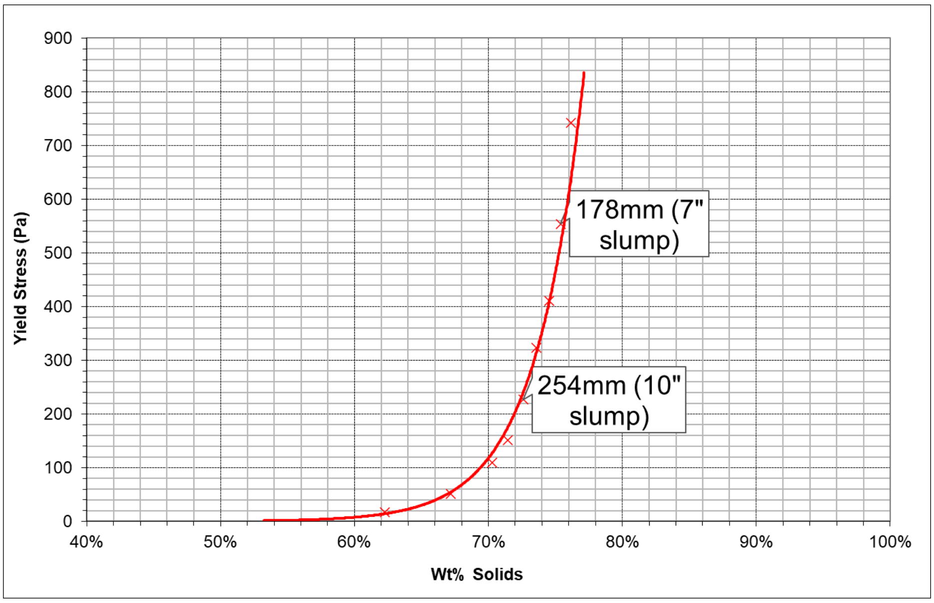


Figure 13-45 Static Stress vs. wt% Solids

Water Bleed and Yield Stress vs. Time

Moisture retention testing was carried out to assess the water bleed properties of the paste while sitting idle in test beakers. Two slump consistencies were tested at four timed intervals. At each time interval the water bleed and yield stress values were measured. Figure 13-46 presents the results.

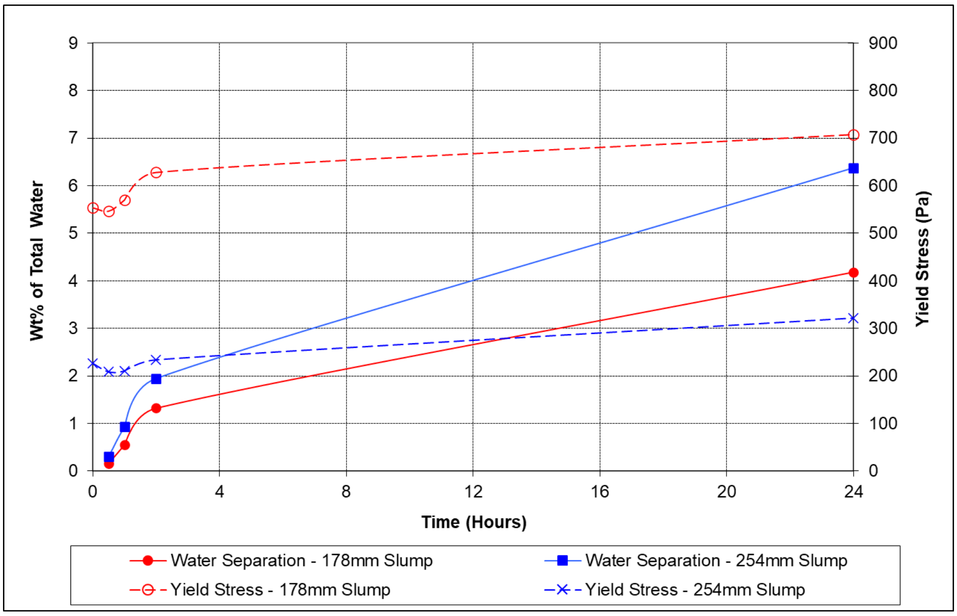


Figure 13-46 Water Bleed and Yield Stress vs. Time

Viscosity and Dynamic Yield Stress Determination

Viscosity testing provides bench scale level flow properties and fluid characterization. Dynamic viscosity and yield stress data is essential for mixer, pump and pipeline design. In order to compare or duplicate viscosity results of non-Newtonian fluids, it is important to test according to the same conditions. Test conditions and parameters such as cycle time and instrument sensor configuration are critical to producing usable data from bench scale viscometers.

The viscosity and dynamic yield stress results are used to estimate pipeline friction losses for paste consistency tailings.

Summarized test results are presented in Table 13-36 as well as on Figure 13-47 and Figure 13-48.

Table 13-36 Bingham Viscosity and Yield Stress Summary

Wt% Solids	Bingham Yield Stress (Pa)		Bingham Viscosity (PaS)	
	Ramp Up	Ramp Down	Ramp Up	Ramp Down
76.7	657	698	1.884	1.606
74.6	423	428	1.033	1.015
73.2	306	315	0.554	0.545
71.6	196	203	0.290	0.280
68.1	83	84	0.098	0.098
63.8	31	32	0.041	0.040
60.6	17	17	0.027	0.026
52.5	4	4	0.012	0.012
45.5	0	0	0.011	0.011

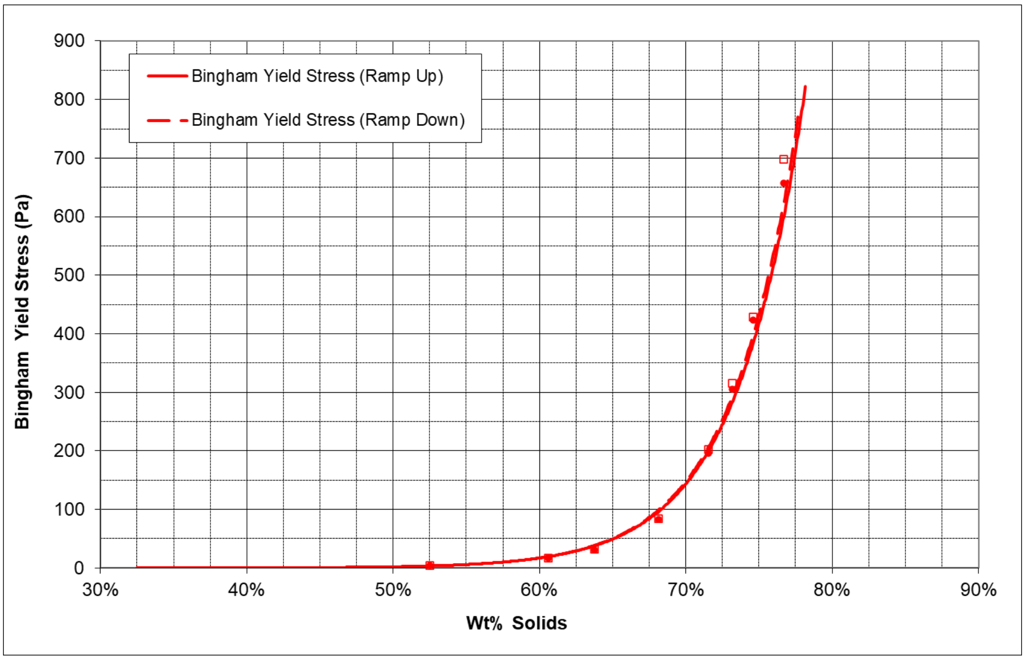


Figure 13-47 Bingham Yield Stress Results

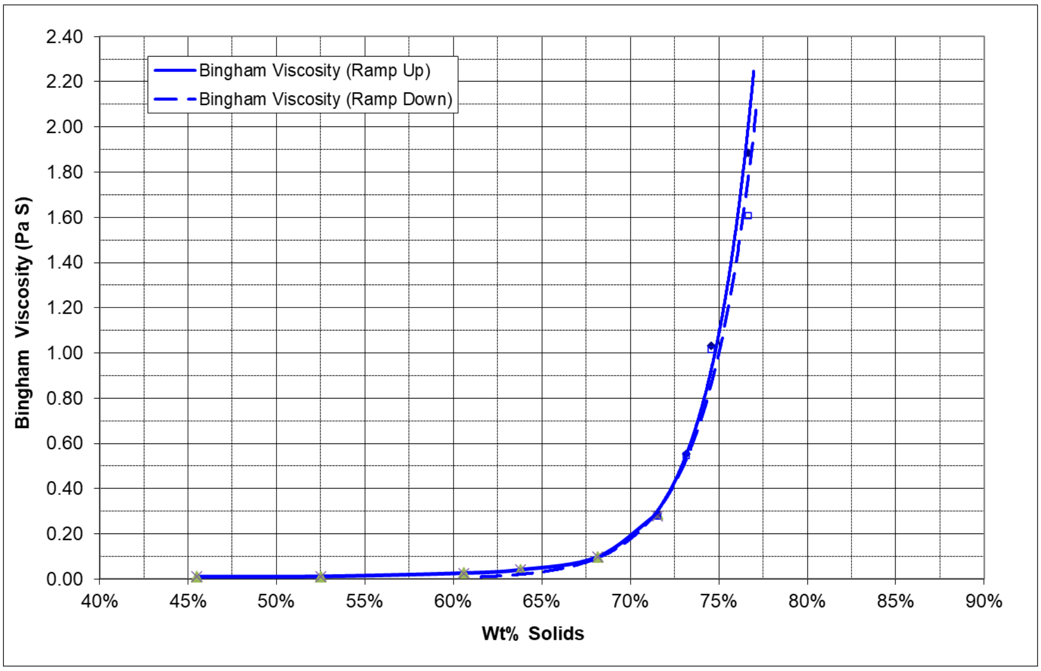


Figure 13-48 Bingham Viscosity Results

Flow Loop Testing

Flow loop testing provides essential data for pump sizing further definition on pipeline friction loss estimates. An examination or interpretation of the data allows for the characterization of the fluid with its corresponding viscosity and yield stress values.

The laboratory scale flow loop system used in the tests consisted of 50 mm (2 inch) and 75 mm (3 inch) schedule 40 steel piping, a progressive cavity pump, pressure transmitters and a magnetic flow meter. Instrumentations were tied into a high-speed data acquisition system.

During the flow loop, a series of test runs were performed at each density to measure pipeline friction loss at several flow rates. Figure 13-49 presents the y-intercept (pipeline yield stress), and Figure 13-50 is the corresponding viscosity.

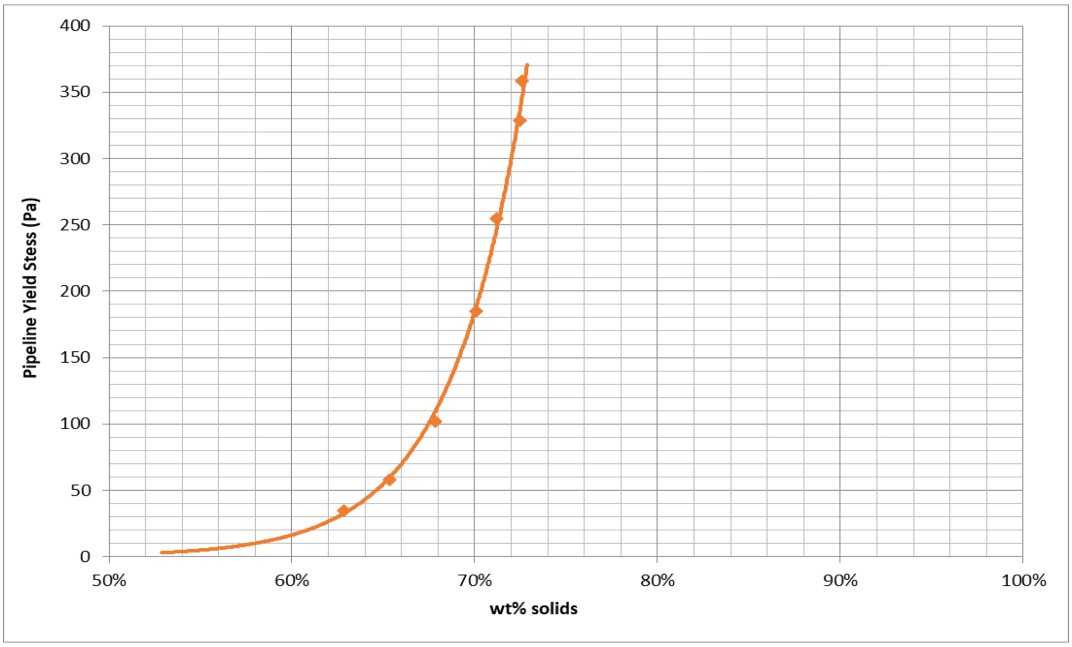


Figure 13-49 Solids Content versus Pipeline Yield Stress

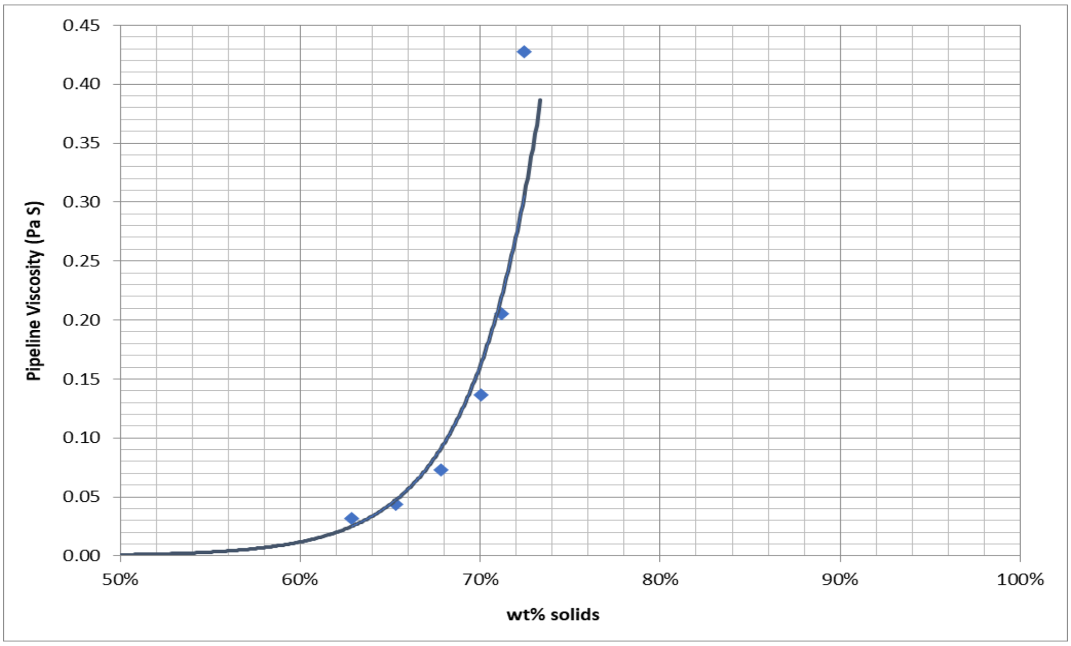


Figure 13-50 Solids Content versus Pipeline Viscosity

It is worth noting that Weir had previously completed flowloop testing on K92's tailings as well. A comparison of the solids content versus friction loss was conducted and illustrated in Figure 13-51. These pipeline friction loss estimates were used for flow modelling purposes to examine the locations which can be reached via gravity and those that require pumping based on the paste slump. Note that the Weir flow loop data was interpreted by WSP.

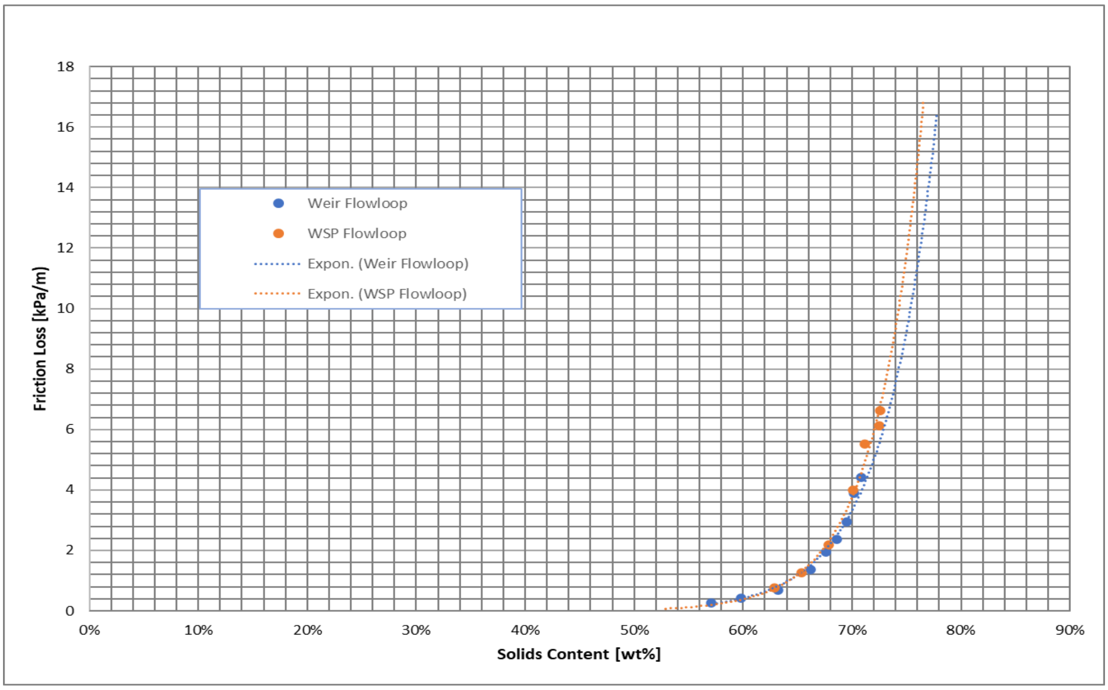


Figure 13-51 Solids Content versus Friction Loss

13.16.7 Unconfined Compressive Strength (UCS) Testing

With a significant portion of paste plant operating costs attributed to the cost of the binder, identifying the binder type with readily available supply, is critical to the long-term operability of the facility. Previous UCS testing [Lycopodium 2022] indicated that a combination of blast furnace slag and general use cement yielded favourable results. This test program started from that premise and further advanced it in a phased manner, examining the impact on UCS based on:

- Different binder types, a blend of blast furnace slag (different sources) with locally sourced cement versus locally sourced cement only.
- Different sources of water available at site.
- Impact of water addition (slump).

13.16.8 Unconfined Compressive Strength Testing

The UCS program was carried out to assess the backfill strength using 51 x 102 mm (2" x 4") cylinders. The cylinders were cured in a high humidity environment maintained at 20 to 25°C (68 to 77°F). Three cylinders per curing period were cast and the results were averaged. The test program is presented in Table 13-37 and the results are presented in Figure 13-52 and Figure 13-53.

Table 13-37 UCS Testing Program

Mix	Wt% Binder	Binder	Material	Water	Slump (inch)	Curing 3 days	Curing 7 days	Curing 28 days
1	2	CA0012623.1156 Cement	CA0012623.1156 Tailings	CA0012623.1156 Tailings Decant	7	3	--	3
2	5					3	--	3
3	7					3	--	3
4	10					3	--	3
5	2					3	3	3
6	5					3	--	3
7	7					3	--	3

Mix	Wt% Binder	Binder	Material	Water	Slump (inch)	Curing 3 days	Curing 7 days	Curing 28 days
8	10	90% CA0012623.1156 Ground Slag (China) / 10% CA0012623.1156 Cement				3	--	3
9	5	100% CA0012623.1156 Cement		CA0012623.1156 Twin Portal 800		3	--	3
10	5			CA0012623.1156 800 Portal Water		3	--	3
11	5	90% CA0012623.1156 Ground Slag (China) / 10% CA0012623.1156 Cement		CA0012623.1156 Twin Portal 800		3	--	3
12	5			CA0012623.1156 800 Portal Water		3	--	3
13	5	90% CA0012623.1156 K92 Slag (Australia) / 10% CA0012623.1156 Cement		CA0012623.1156 Tailings Decant		3	--	3
14	10					3	--	3
15	3	90% CA0012623.1156 K92 Slag (Australia) / 10% CA0012623.1156 Cement			7	3	3	3
16	3				10	3	3	3
17	4				7	3	3	3
18	4				10	3	3	3
19	5				10	3	3	3

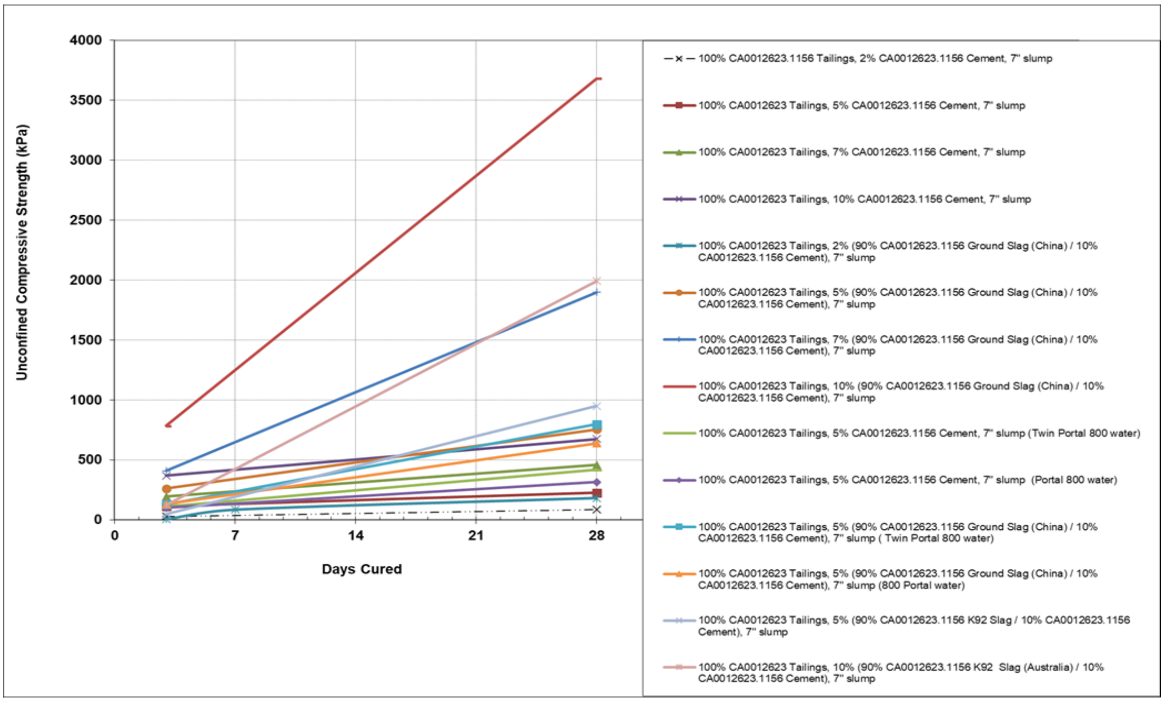


Figure 13-52 UCS Results - comparing all binder and water sources

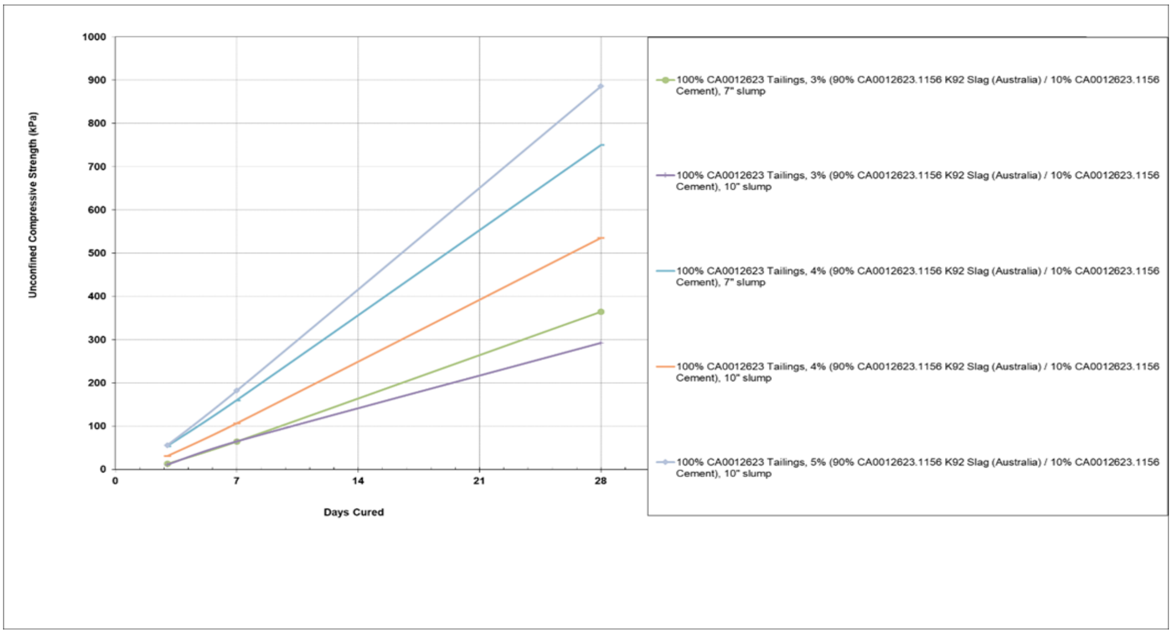


Figure 13-53 UCS Results - comparing low binder contents (3-5 wt%) using K92 Australia Slag

- A mixture of blast furnace slag and cement performed better than cement alone when comparing similar binder contents. The exception is on early breaks (3 days) and low binder content (2 wt%) where cement out performs 90/10 slag cement.
- If cement is used alone, based on the 5 wt% binder test results, the Twin Portal 800 water used in the paste mix yielded higher UCS at 28 day break.
- The slag from China resulted in higher UCS compared to the slag from Australia at 5 wt% binder and 28 day break.
- The slag from China resulted in higher UCS compared to the slag from Australia at 10 wt% binder and 28 day break.

- The Twin Portal 800 water and the decant water yielded similar results when comparing the 5 wt% binder and 28 day break data. The Portal 800 water yielded the lowest UCS by comparison.

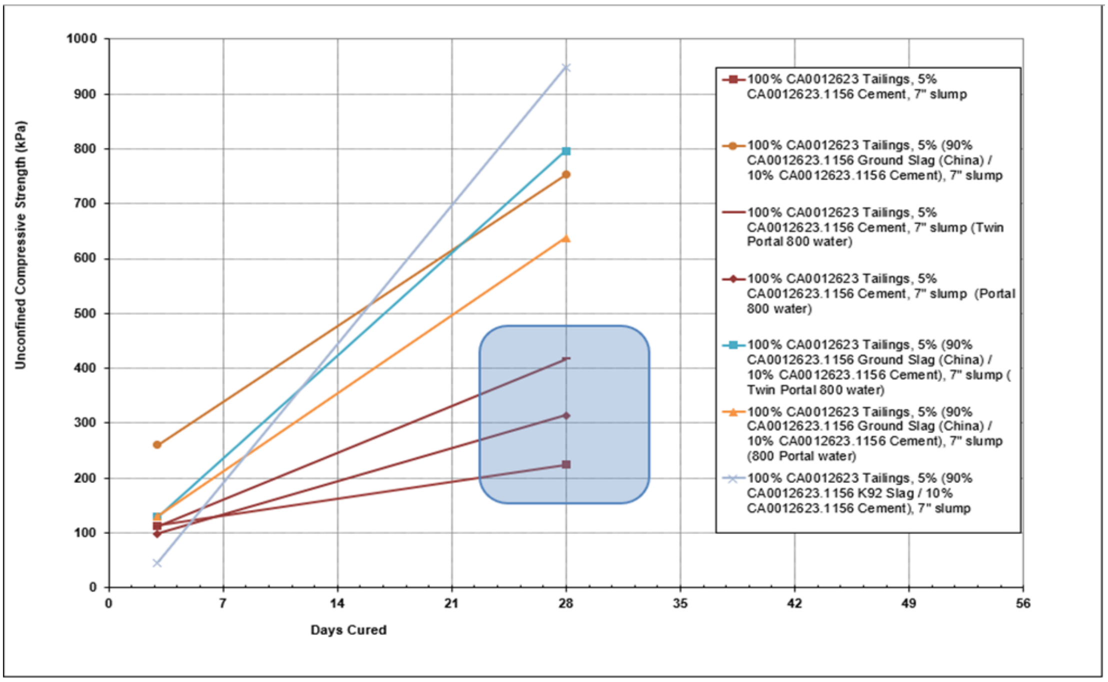


Figure 13-54 UCS results comparing different binder types and water sources at 5wt% binder and 7" slump paste

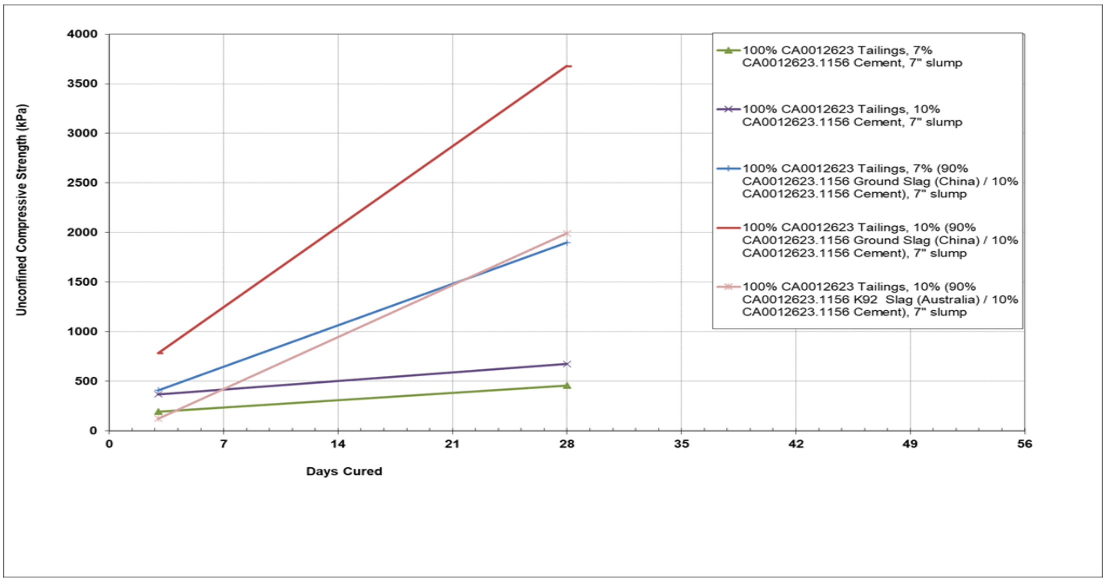


Figure 13-55 UCS results comparing different binder types and water sources at 7wt% and 10wt% binder and 7" slump paste

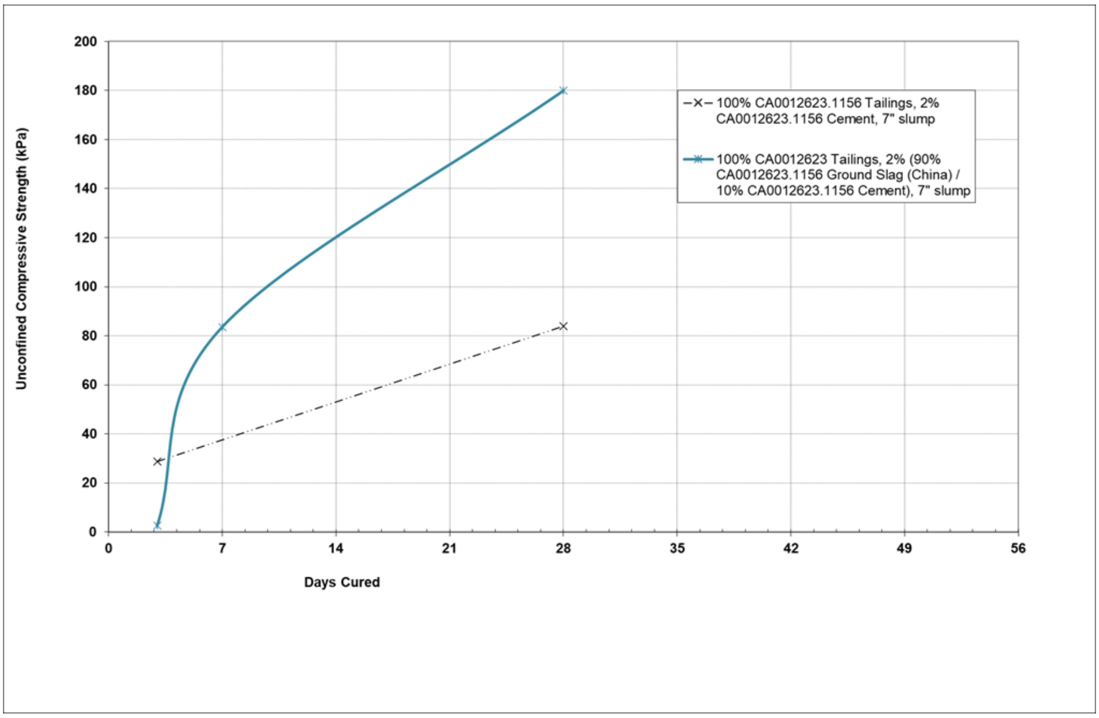


Figure 13-56 UCS results comparing different binder types and water sources at 2wt% binder and 7" slump paste

Further testing is currently being conducted from different blast furnace slag sources, for 14-day breaks and longer term breaks to provide a better understanding of UCS behaviour.

13.17 Process Selection and Design Paste Plant And Design Paste Plant

13.17.1 Kora Mine Backfill Physical Requirements

Introduction

Mine backfill is to form an integral part of the proposed Kora underground mining process by helping to maintain stable ground conditions, maximizing the ore extraction ratios and assisting with the disposal of mine tailings. To adequately service the mining process, the backfill system must be capable of:

- Producing fill with the required mechanical properties.
- Delivered at rates that do not constrain mining activities.
- Have suitable infrastructure to permit a utilisation capable of satisfying the mining schedule.

This section describes the specific backfill requirements in relation to the proposed K92ML mining process.

Mine Backfill Mechanical Design Requirements

The Kora underground orebody is a narrow vein sub-vertical orebody, with weak hanging wall host rock. Ore is to be extracted using a combination of continuous retreat top-down and bottom-up longitudinal open stoping with mine backfill. The mine portal is at approximately RL 800 m, from this point, a set of dual inclines have been developed from the 800 Portal, laterally 2,400 m, to intersect the Kora orebody at circa. 900 Level. The Kora orebody extends approximately 1,100 vertical metres (from elevation 713 to 1,832 m, i.e. over 1,000 m above the 800 Portal) and extends a distance of 800 m laterally. A long section showing existing development (as white strings) and future development (floor strings, in magenta) is presented in Figure 13-57. The location of the Kora orebody is shown by the cluster of development on the LHS.

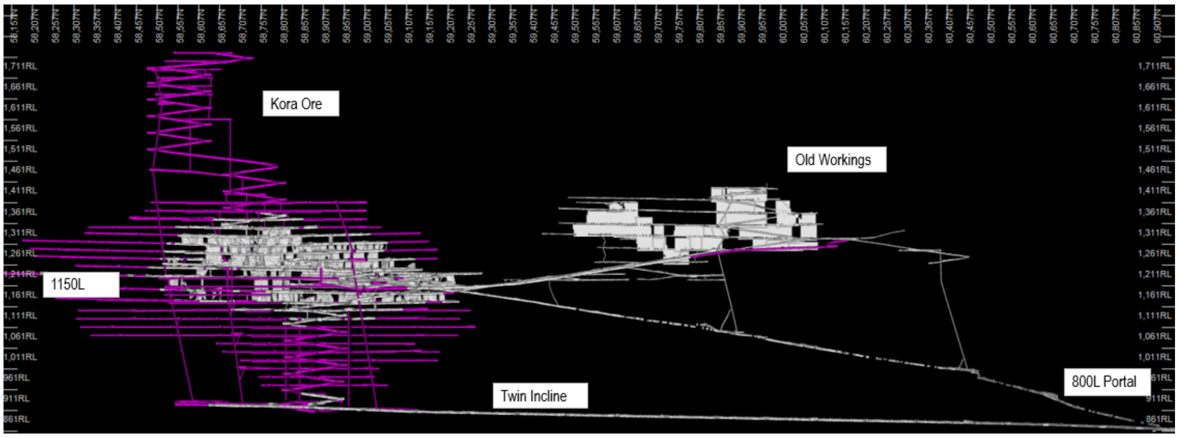


Figure 13-57 Long Section Showing Proposed Kora Underground Mine

The Kora orebody is understood to exist in a series of closely spaced narrow veins and it is proposed to mine these veins independently with relatively narrow stopes, leaving a slender pillar between ore veins. The proposed stope extraction sequence is to involve both a 'bottom-up' and 'top-down' extraction sequence.

To fulfil its duty in the mining cycle, Kora backfill must:

- Provide a suitable bulking material that restrains rock mass unravelling.
- Remain self-supporting during nearby blasting activities.
- Maintain stability when exposed vertically and horizontally during adjacent mining activities.
- Provide a stable working platform for bogging overlying stopes.

The mass of ore extracted from different stope widths is presented in Figure 13-58. This figure shows that 88% of Kora stopes (and therefore fill exposed) are expected to have an across strike span less than 10 m, 11% of stope range in width from 10 to 15 m, while only 1% of stope spans are greater than 15 m. The floor-to-floor sub-level interval throughout the mine is 20 m.

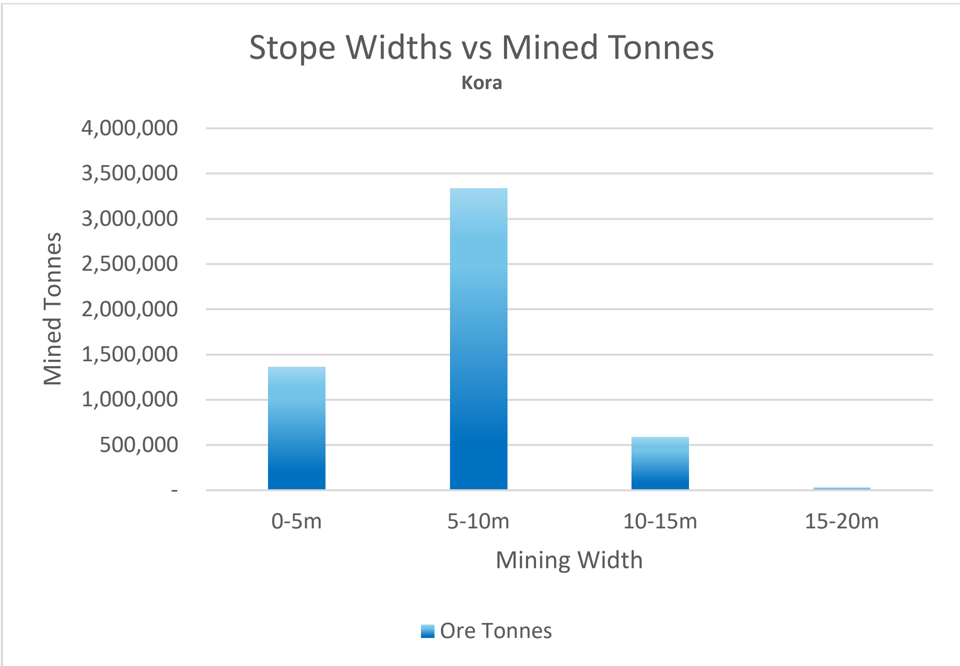


Figure 13-58 Distribution of Stope Widths within Kora

The Kora orebody is to be mined with minimal pillars (i.e. near 100% ore extraction) and as such, to maintain stability during adjacent mining activities the fill is required to achieve a specific design strength. Due to the relatively narrow stope widths and short (20 m) sublevel interval, relatively low strengths will be required for vertical and horizontal exposures. However, due to the small stope volumes the mining sequence demands fast stope turnaround times, which requires these low strengths to be achieved after short hydration periods. The Kora paste bond strength is to be achieved using a combination of GP cement and Ground Granulated Blast Furnace Slag (Slag) and the metering and mixing of these binders with the paste forms a critical part of the paste system.

Fill masses that are not exposed must be either adequately free draining or possess sufficient bond strength to avoid liquefaction during seismic events (such as nearby blasting or an earthquake event). Testwork presented in Section 13.16 indicates that this minimal strength requirement can be achieved using a combination of 1% GP cement and 1% Slag.

Minefill reported in 2022 that where fill is to provide a stable working platform it is expected that paste with a minimum unconfined compressive strength (>200 kPa) would be placed and sheeted with a thin layer of waste rock to provide a firm surface with adequate wheel traction.

Fill Strength Requirements

Introduction

To determine the Kora paste fill strength requirements, a rigorous three-dimensional numerical modelling approach was applied and undertaken by Geotechnica. This section provides a summary extract of the main conclusions from this work that was completed to define the Kora paste strength requirements. The full report called 'Analysis of Paste Fill Exposure Stability' was done to inform this work based on an updated assessment of strength requirements for the project.

Geometry

Mining at Kora involves extraction of ore immediately adjacent to, above and beneath previously deposited fill masses. During this process, fill is exposed both vertically and horizontally. This analysis considers the strength requirements for both of these exposure events.

Fill Strength Recommendations

The modelling results for longitudinal mining areas have been summarised in a stability chart to determine the strength requirements to maintain stability over a range of typical stope configurations. Table 13-38 provides the recommended CPB strength requirements in 50 kPa increments.

Table 13-38 CPB Strength Requirements

Exposure Type	Stope Width (m)	Design Strength (kPa) (FoS = 1.5)
Simultaneous Underhand and Vertical	0-5	250
Simultaneous Underhand and Vertical	5-10	300
Simultaneous Underhand and Vertical	10-15	450
Simultaneous Underhand and Vertical	15-20	500
Vertical Only	0-5	150
Vertical Only	5-10	150
Vertical Only	10-15	200
Vertical Only	15-20	200
Vertical Only	20-25	200
Vertical Only	25-30	250

It must be noted that the numerical modelling results represent the static behaviour of the CPB (cemented paste backfill) exposures and do not account for the dynamic effects of adjacent blasting or seismic events.

Detailed laboratory and in situ testing of Kora Paste material is required to confirm the modelling results presented within this report. This should include triaxial and tensile strength testing. Ongoing, robust quality control unconfined compressive strength testing of the paste fill is also required to provide confidence in the paste fill exposure stability. This should include in situ sampling and testing of the placed paste fill material.

It is also recommended to conduct some longer-term UCS tests including 56, 112 and 224 day curing to assess the potential for long-term strength degradation of the paste fill, which has been observed at other paste fill operations.

Optimisation of Binder Additions

The analysis of homogeneous fill strength provides a basic assessment of the strength required to maintain stability of the planned vertical and horizontal exposures. Further optimisation of the cement addition can be achieved by progressively reducing the strength as the height increases in the vertical fill column.

Generic analyses have been conducted to provide simple guidelines for the possible reduction of strength during placement. For ease of analysis and practical implementation, a generic fill column, was divided into 5 intervals. Using the strength obtained from each stable exposure analysis and reducing the fill strength in the upper portion of the fill column according to the criteria presented in Figure 13-59, the stability of all vertical and underhand exposure configurations analysed were predicted to remain stable. This criterion has been successfully implemented at multiple CPB operations worldwide. It is recommended that the minimum 28-day CPB strength placed within any stope is 100 kPa.

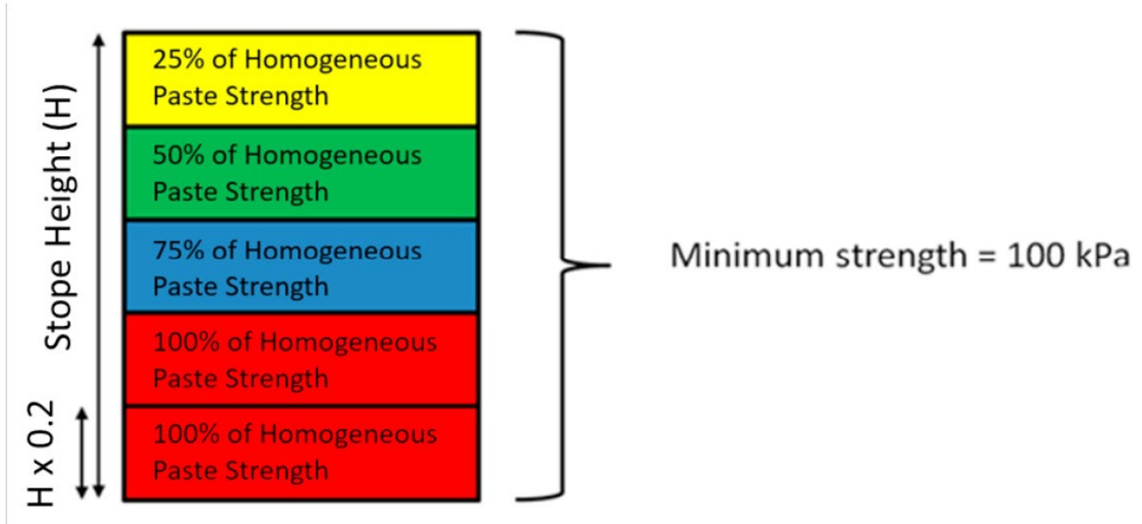


Figure 13-59 Fill Strength Reduction Criterion Simulated

As with any paste cement optimisation program, it is essential that any reduction in cement addition is managed in a progressive fashion with thorough recording and monitoring the performance of each individual stope.

13.18 Mass Balance and Fill Production Rate

13.18.1 Fill Demand

To satisfy the mine backfill requirements, it is necessary for the system to be capable of filling the underground mine void space in accordance with the mining schedule requirements. The forecast monthly fill requirements are presented in Figure 13-60. This figure shows the monthly fill requirements plotted against sequential mining month.

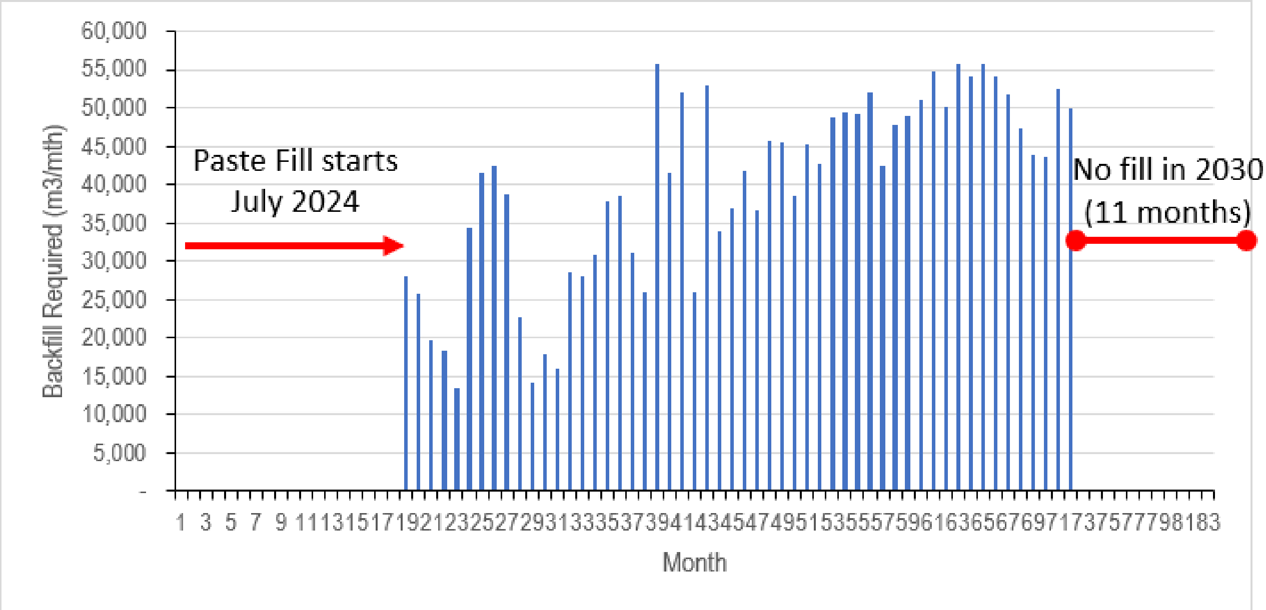


Figure 13-60 Kora Fill Schedule

Figure 13-60 shows that a life of mine fill quantity of approximately 2.2Mm^3 is required with a maximum monthly fill production rate of ~60k m^3/mo and maximum fill demand (over any 12-month period of 619,000 m^3) is necessary to ensure mine production is not constrained.

13.19 Material Availability

13.19.1 Fresh Tailings

The Kora mining and processing operations are focused on a total ore processing rate of 1,200,000 tpa. Based on calculations provided by Evan Kirby (MMS) and adapted in the DFS cost model, the process plant would remove on average 5% of the total mass with the remaining 95% reporting to tailings. Assuming an in-situ tailings settle density of 1.35 t/m^3 (ATCW), the monthly available mass and required backfill mass are compared in Figure 13-61. Also plotted on the RHS axis is the portion of available tailings that must be utilised for filling, on a 3-month rolling average basis.

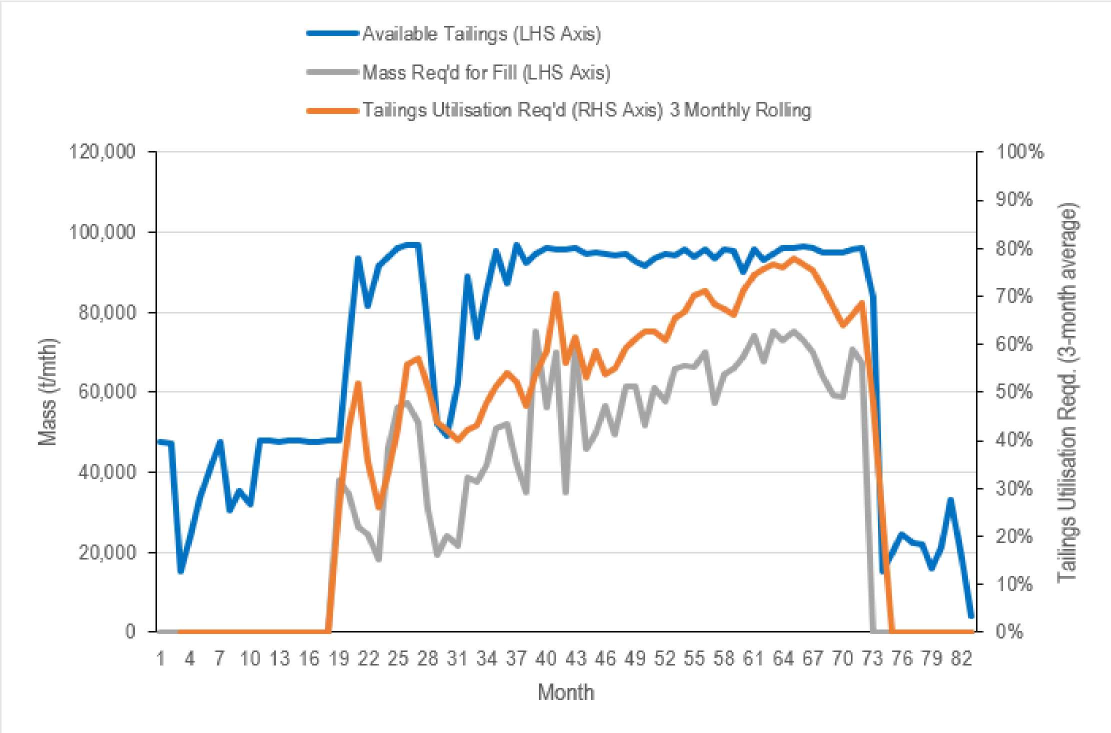


Figure 13-61 Mass of Available Tailings and Tailings Mass Required for Filling Against Production Month

Figure 13-61 shows that, from when backfill placement starts in month 19 up to month 52, the required tailings utilisation is approximately 50%, with this increasing to approximately 70% beyond 52 months to when filling ends in month 72. This is due to the increased void filling requirements.

Using the monthly fill demand, from the Kora mining schedule, the fill demand was converted to an instantaneous fill production rate, if the system is utilised 50% and 70% of the time. The instantaneous fill demand for these utilisation rates is plotted against mining month in Figure 13-62.

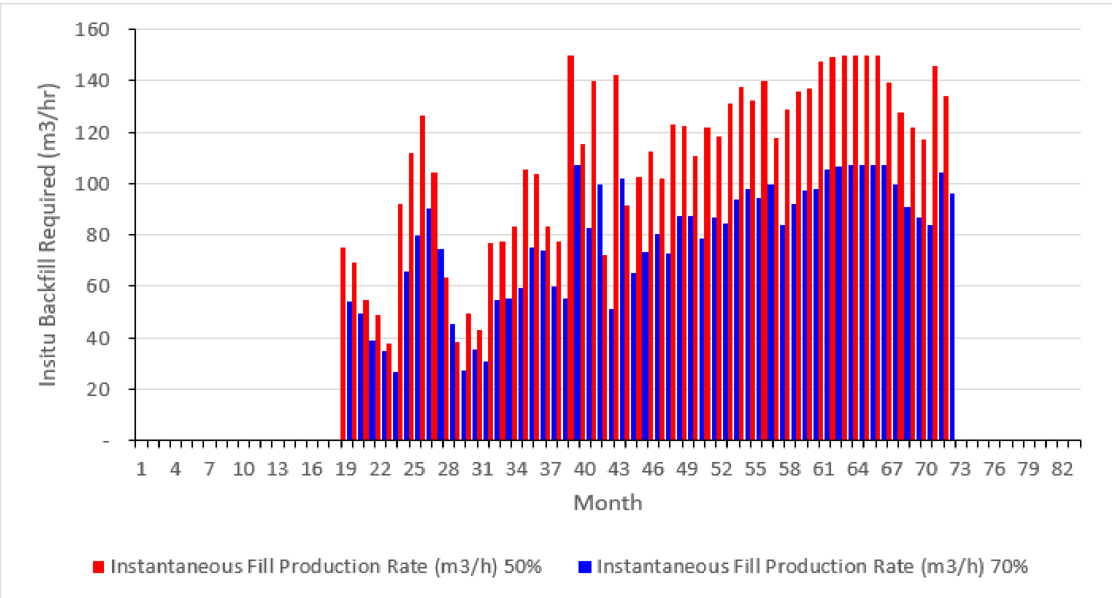


Figure 13-62 Mass of Available Tailings and Tailings Mass Required for Filling Against Production Month

Figure 13-62 shows a maximum instantaneous fill demand of just over 105 m^3/hr (at 70% utilisation) and 150 m^3/hr (at 50% utilisation). At the in situ dry density of 1.35 t/m^3 (previously 1.23 t/m^3 in 2022) this equates to a maximum fill tailings consumption of 142 and 203 t/hr for 70% and 50% utilisation, respectively.

The pressure filters at the 1.8M tpa rate are designed to process 145 tonne/h of tailings to meet backfill requirements. The pressure filtration plant has the ability to produce more tailings than the paste plant requires. The pressure filtration plant has a utilisation rate of 85% and at 30 days of production equates to 89kt of tailings solids. The paste plant has a backfill rate of about 148 m3/h at 60% utilisation this translates to about 64k m3/mo of paste or consumption of 89kt of tailings solids. The difference in tailings throughput between the mill production and the filtration plant demand would require that a portion of the tailings be discharged from the tailings thickener to the TSF.

13.19.2 Paste Fill - Process Description

Scoping and pre-feasibility analysis determined that paste backfill is the preferred backfill method for the Kainantu mine. As the design evolved through the feasibility stage of the engineering cycle, previous assumptions about the feasibility of pumping slurry from the mill to a thickening station located near the portal were found to be undesirable. The ruggedness of the terrain, the pumping power required and the narrow amount of road width available for pipeline installation meant that a pumped solution for the slurry would be more challenging to implement.

The design shifted to incorporate the trucking of pressure filter cake from a pressure filter plant located near the mill up to a staging area located at the "bus stop" and then onto the paste plant located underground. The bus stop is a portion of flat ground located just before the main mine road passes through the local village prior to accessing the main portal. The transfer of cake from the filter plant to the bus stop will utilize the back haul of the main haul fleet to reduce haul road traffic. Cake is then transported from the bus stop to the main paste plant located on the 1225 level of Kainantu mine. Paste produced at the plant is transported to the stopes via a paste reticulation system that utilizes a piston diaphragm pump to supply the pumping pressure required.

The paste process begins at the discharge of the tailings thickener located at the K92ML mill facility. Tailing is pumped from the thickener to the filter feed tank located at the pressure filtration building. The filter feed tank feeds tailing to a pressurized tailings pipe which feeds two pressure filters. The pressure filters are horizontal type and produce filter cake at a target of 88 to 86 wt% solids concentration and the filter cake discharges onto filter cake feeders located below the pressure filter units. The filter cake feeders will discharge to a common loadout conveyor which will transfer filter cake to a staging area where the cake will be loaded onto trucks for hauling to the bus stop storage area. At the bus stop the haul trucks will off load to a central stockpile which will be used to create an operational buffer between the filtration and backfill production plants. After dumping their load, the haul trucks will be rinsed off at a wash bay prior to continuing on their travel cycle to pick up ore from the mine. On the opposite side of the stockpile, loaders will transfer filter cake from the stockpile into a fleet of dedicated filter cake haul trucks which will deliver cake to the paste backfill plant.

13.19.3 Paste System Process Design Criteria

The tailings filtration plant and the paste backfill plant are inherently linked and both plants are effectively sized to meet the backfill requirements dictated by mine planning. It is understood that that the maximum monthly demand is 46,523 m^3 and 63,637 m^3, respectively, for the current 1.2 million tonne per annum and the expanded 1.8 million tonne per annum mining rates. The design maximum paste plant pouring rate was then calculated based on a 175 mm slump material which would be the highest density paste poured underground and translating to the highest demand for filtered tailings. Table 13-39 summarizes the key elements in the design criteria.

Table 13-39 Summary of Key Design Criteria Values

Parameter	Unit	Value	Notes
Maximum Paste Required	m^3/month	46,523	At 1.2 million tonne/annum ore
Maximum Paste Required	m^3/month	63,637	At 1.8 million tonne/annum ore
Paste Solids Content at 175 mm Slump	wt% solids	75	
Paste Solids Content at 250 mm Slump	wt% solids	72.5	
Paste Plant Capacity (Wet)	m^3/h	148	7″ slump and 5% binder
Target Filter Cake Solids Content	wt% solids	88	
Filter Plant Capacity (Dry)	tonne/h	117	At 1.2 million tonne/annum ore
Filter Plant Capacity (Dry)	tonne/h	160	At 1.8 million tonne/annum ore

14. MINERAL RESOURCE ESTIMATES

The effective date of the MRE for the Kora and Judd lodes is the 12th of September 2023, which was the date that the latest database was received by H&SC.

The updated MRE for the Kora deposit including the Judd Lode system are based on samples from diamond drillholes (surface and underground) and face sampling of development drives. There is some very minor underground sludge hole sampling included which was aimed at defining mineral limits to the face sampling intervals.

Table 14-1 provides summary details of the sampling for the overall deposit area. The significant proportion of the K92ML drilling has been on the Kora section of the mineralization.

Table 14-1 Summary Details of Sampling Methods

Company	Year	Location	Type	No of Holes	Metres	Ave Length (m)
Barrick	2008 - 2015	Surface	DD	24	10,690	445.4
		UG	DD	6	808	134.7
Highlands & Others	1990 - 2007	Surface	DD	79	16,596	210.1
	(Irumafimpa)	UG	DD	562	26,514	47.2
			Total	671	54,608	
K92ML	Oct 2017 - Sept 2018	UG	DD	83	9,564.2	115.2
			Face	461	1,499.3	3.3
K92ML	Oct 2018 - Feb 2020	Surface	DD	16	7,390.0	461.9
			DDwedge	3	719.7	239.9
		UG	DD	96	21,224.5	221.1
			DDwedge	7	935.2	133.6
			Face	312	1,657.5	5.31
K92ML	Mar 2020 - Oct 2021	Surface	DD	21	6,155.1	293.1
		UG-Kora	DD	231	36,152.6	156.5
			DDwedge	3	471.7	157.2
			Face	509	2,798.6	5.5
	Mar 2020 - Dec 2021	UG_Judd	DD	49	8,935.6	182.4
			Face	193	1,059.8	5.5
	Oct 2021 - Sep 2023	Surface	DD	53	22,134.3	417.6
		UG-Kora	DD	184	38,232.1	207.8
			Face	425	2,438.6	5.7
	Dec 2021 - Sep 2023	UG-Judd	DD	159	31,089.4	195.5
			Face	209	1,205.4	5.8
		Kora Total Drilling		727	154,477.4	
		Kora Total Face		1,707	8,394	
		UG-Judd Total Drilling		208	40,025	
		UG- Judd Total Face		402	2,265	
		Grand Total Drilling		1,576	237,612	

(DD = diamond drilling; UG = underground)

The MRE for the Kora and Judd deposits were prepared using the Ordinary Kriging (OK) option in the H&SC in-house GS3 modelling software package. H&SC considers OK to be an appropriate estimation technique for the type of mineralization, its extent and the nature of the available data. The resource estimation includes some internal low grade material. The drillhole data and resulting GS3 models were loaded into the commercially available Surpac mining software for geological interpretation, composite selection, block model creation and validation, resource estimate reporting and to facilitate any transition to future mining studies.

The GS3 modelling software was developed by Neil Schofield (ex-Stanford University) and has capacity for both Multiple Indicator Kriging (MIK) and OK modelling techniques.

The approach to resource estimation for the Kora and Judd deposits is relatively straightforward. A 3D geological interpretation of mineral domains as wireframes for the K1, K2, Kora Link and Judd lodes was completed using the Surpac software. These wireframes were then used to select 1m composite data from samples in the drillhole database which were then subject to data analysis including aspects of spatial distribution (variography). OK modelling was used with four search domains for each lode, based on subtle variations in dip and strike of the lodes, with the resulting 3D models loaded into a Surpac block model. Post-modelling processing, including block model validation and reconciliation, was undertaken in Surpac. The newly generated resource estimates were classified taking into account a number of factors including sample spacing and distribution, variography, geological understanding, QAQC procedures and outcomes, density data, core recovery and reconciliation with production.

The Mineral Resource estimates (MRE) reported in this section have been classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves.

14.1 Supplied Data

H&SC was supplied by K92ML the following items:

- A drillhole database in MSAccess for direct use in Surpac. Drillhole data comprises surface and underground diamond cored holes and face sample lines from the current underground mining.
- 3D wireframes for topography, the underground development and the stope solids generated from mining. Updated mineral wireframes were supplied as guidance for the geological interpretation.
- Mill feed figures for tonnes, gold, copper and silver grades.
- A suite of reports covering previous drilling and resource estimation for the Kora deposit and different aspects of the underground drilling for Kora e.g. analysis of recovery, density data, QAQC procedures and outcomes.

A database summary for Kora is presented in Table 14-2, showing the data provided for a range of items. Table 14-3 is the database summary for Judd.

Table 14-2 Database Summary for Kora

	Number of holes with records		
Item	**DD Holes**	**Face Samples**	**Records**
Collar	817		162,800.46 m
Collar		1,825	9927.14 m
Survey	817	1,825	22,976
Au g/t	775	1,821	116,290
Cu %	775	1,821	104,221
Ag g/t	775	1,821	113,664
Lith	776	0	45,708
Altn	408	0	17,685
Min	400	0	17,640
RQD	632	0	112,020

Table 14-3 Database Summary for Judd

	Number of holes with records		
Item	**DD Holes**	**Face Samples**	**Records**
Collar	259		55,551.05 m
Collar		433	2,755.42 m
Survey	259	433	10,187
Au g/t	249	433	37,613
Cu %	249	433	37,615
Ag g/t	249	433	37,611
Lith	247	0	15,029
Altn	36	0	1,476
Min	33	0	1,361
RQD	227	0	46,536
Density	176	0	1,393

14.2 Geological Interpretation

The Kora deposit consists of a relatively complex, dilational structural zone hosted in Tertiary phyllites. The mineralization zone comprises narrow, high grade gold quartz-sulphide veins, veinlets and disseminations associated with clay shear zones, the result of brittle/ductile shearing. Historically at the old Kora North deposit the K1E lode was delineated in the footwall of the structural zone and showed a marked higher gold grade than other lodes in the general Kora/Irumafimpa area. The K1W lode was identified immediately adjacent to the hangingwall of the K1E lode and had a much lower grade gold zone associated with it. This lode is then structurally overlain by a clay fault zone of varying widths, the Kora Fault. A distinct separation comprising relatively barren phyllite then occurs before reaching the hangingwall zone of the dilational structure where the K2 lode resides. This lode is noted in the Kora area for its relative higher copper content compared to K1, occurring as chalcopyrite blebs, veins and masses associated with modest amounts of quartz veining. Initial modelling in 2018 by H&SC used the K92-supplied K1E, K1W and K2 lode wireframes, which were based on a 1g/t gold cut off and had resulted in wireframes considerably narrower than the current development drives and stope widths. Reconciliation with mill production indicated that using the narrow wireframes considerably understated the amount of tonnes sent to the mill and the gold ounces produced by the mill.

Work completed by Simon Tear in 2018 comprised a site visit and a reassessment of the geological interpretation of the mineral lodes, in conjunction with the current mining method, all of which led to the 2018 MRE.

The resulting geological interpretation featured a larger K1 wireframe shape, combining K1E and K1W lodes, and was based simply on the presence of gold mineralization derived from the assays i.e. using a much lower, circa 0.1 to 0.2 g/t, gold cut-off grade, in combination with some geological control, spatial patterns and a nominal minimum mining width of the current stoping and development i.e. approximately 5.2 m wide. A similar process was applied to the K2 lode which also resulted in a larger wireframe. For the 2020 MRE work, the wireframes were expanded to incorporate the recent and historic surface drilling, which involved including the previously defined Kora and Eutompi mineral zones. The outcomes of merging these two areas into the K1 and K2 wireframes appeared reasonable and were partly confirmed by surface drilling of Eutompi, which resulted in minimal changes to the wireframes.

From the 2020/2021 drilling the geological interpretation of mineral domains was confirmed with only very minor modifications to the domain designs, although locally there was some increased level of complexity ascertained from either the face sampling or in areas of more peripheral drilling to the main mineral zone. The main changes are a trimming at depth of the K1 lode in the south, where the original lode interpretation was based on very weak intercepts that often ran below cut-off grade. This was offset by the addition of material to the K2 lode in the same area.

The Kora Link lode is a relatively smaller interstitial mineral zone between K1 and K2 where there is the appearance of possible multiple narrow, partially overlapping mineral vein systems. These veins appeared at times to be parallel to the bounding K1 and K2 structures, and other times they appeared slightly oblique, transecting from the K2 footwall to the K1 hangingwall. The 2020 geological interpretation favoured 3 sub zones but with expanded domain boundaries, such that most of the area between K1 and K2 for a sub-area of the whole deposit was included. The lode was subject to further geological reassessment following a substantial amount of infill drilling and some face sampling such that H&SC considered the most appropriate geological model for the lenses of quartz/gold mineralization is to treat the whole area between the K1 and K2 lodes, where there is significant segregation, as a single mineral domain i.e. the Kora Link lode. This was alluded to in the 2020 mineral estimation report.

The Kora geological interpretation for the 2023 MRE has seen a considerable expansion to the south of the K2 as a result of the additional expansionary drilling. There is some expansion of K1 in the upper, nearer surface, zone again the result of additional drilling. Most of this drilling has shown the veins to be of lower gold grades but generally with higher copper grades. The additional infill drilling in the central Kora area has provided for a very modest down dip expansion of K1 and K2, plus has also given some indications of further complexity between K1, K2 and Klink in local areas.

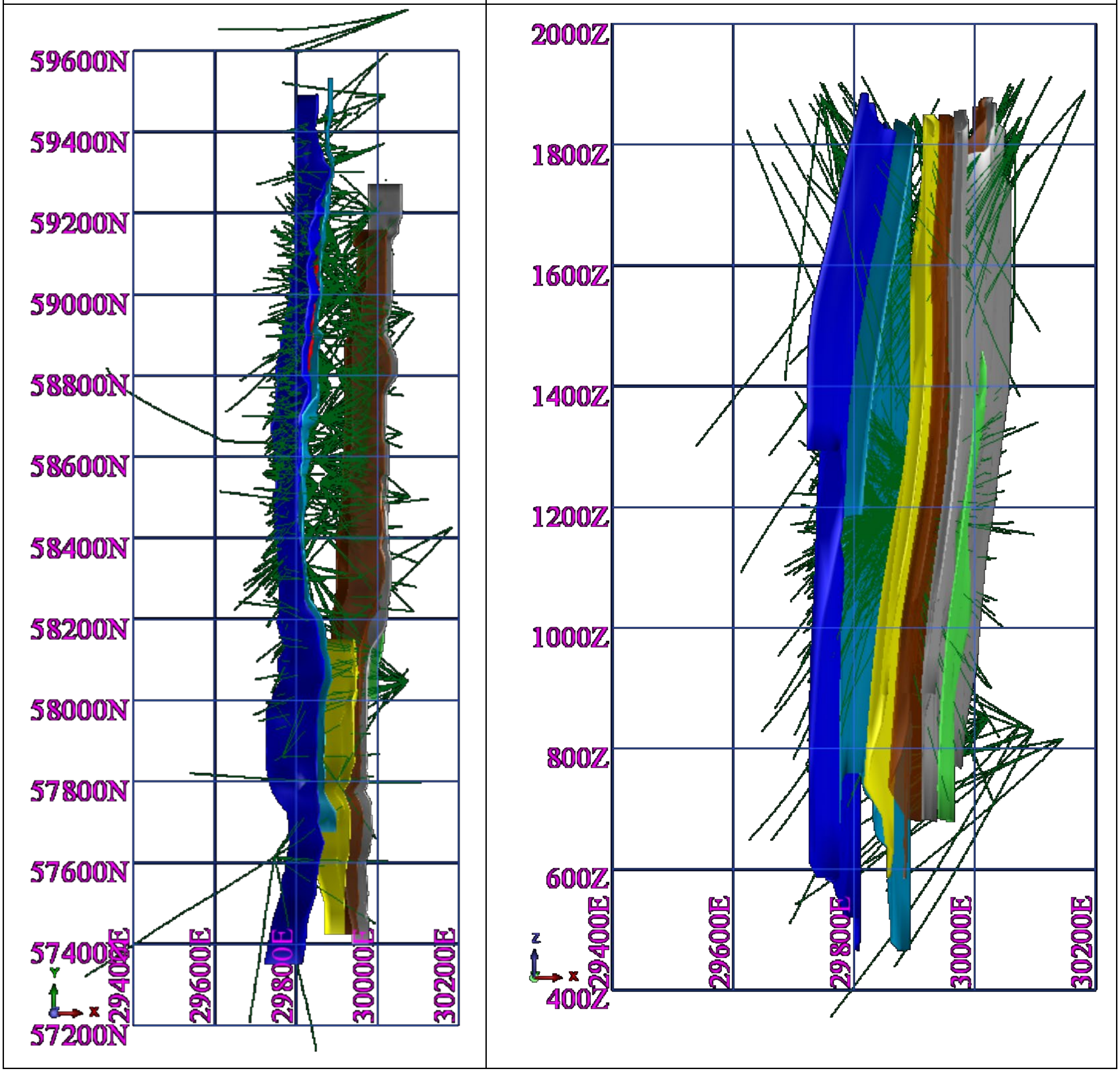
In 2021 a 3D geological interpretation of the J1 lode was developed by H&SC based on a supplied interpretation by K92ML and was consistent with the geological interpretation for the main Kora K1 and K2 lodes i.e. based on a nominal average minimum mining width of 5.2 m. Geological continuity of the lode was regarded as good for the bulk of the drilling/sampling except for the southern end where the lode appeared to break down into a series of much narrower and lower grade lodes that seemingly were non-aligned with the main lode. Further drilling was recommended to resolve the lode definition, and this was completed in 2022/3.

The increased amount of drilling and face sampling at Judd has allowed for an expansion of the original J1 lode in all directions plus the interpretation of three other peripheral lodes, J2 and J3 in the footwall to J1 and J1W in the hanging wall. The J1 lode is a parallel structure to the main Kora mineral system, approximately 150 m to the east. The vein system is slightly narrower than the K1 or K2 lode but has a very similar dip and strike. The lode was inspected in several places by Simon Tear of H&SC during a site visit conducted in June 2023.

Figure 14-1 shows the spatial relationship between the Kora lodes and the Judd lodes.

The geological interpretation for both Kora and Judd was completed in cross section. 10 m spaced sections were used for the central part of Kora increasing to 25 m spaced E-W sections in the southern area and 20 m spaced sections for the northern area. A similar strategy was employed for Judd with a northern set of 10 m sections and a southern set of 25 m spaced sections. All wireframes were snapped to drillholes.

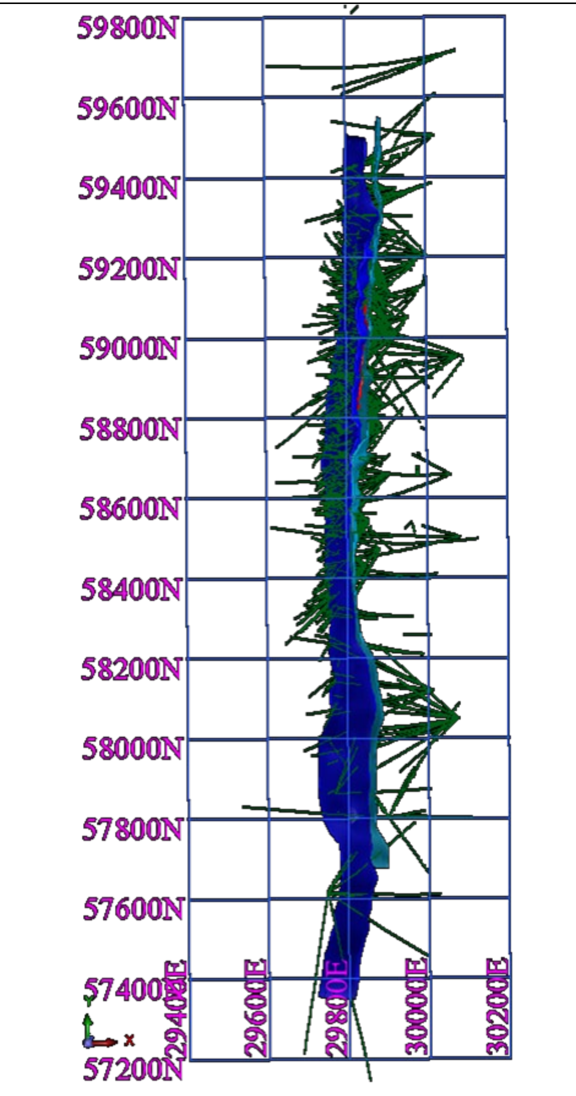
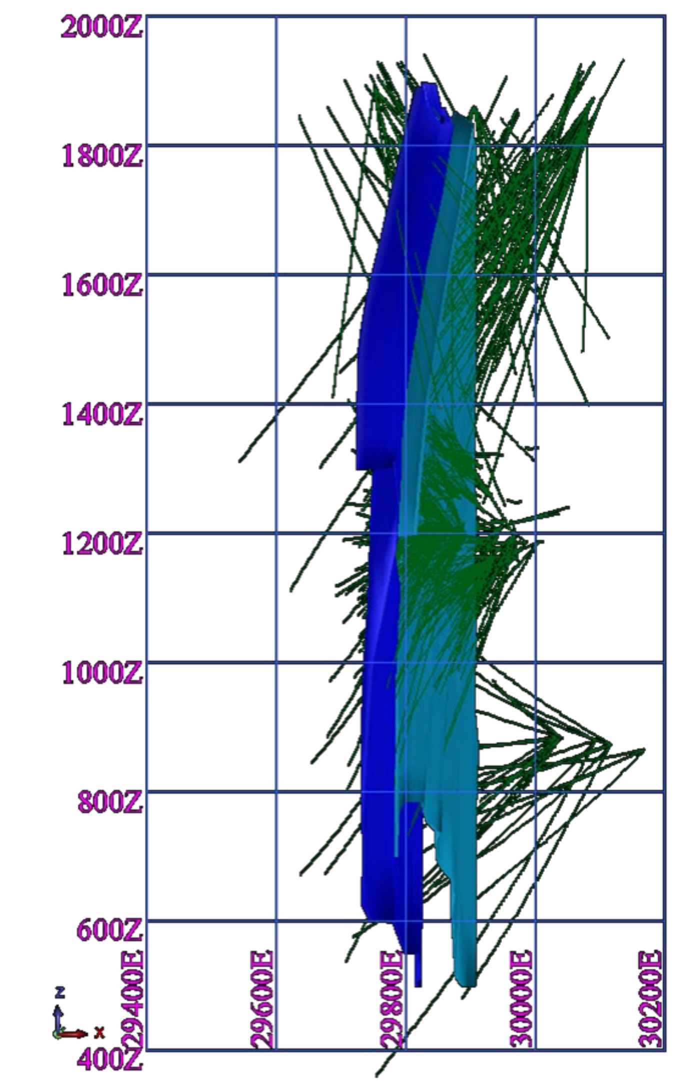
Plan	Cross Section



Cyan = K1 Lode; Blue = K2 Lode; Red = Kora Link; Brown = J1 Lode; Grey = J2 Lode; Green = J3 Lode; Yellow = J1W Lode

Figure 14-1 Plan and Cross Section of the Kora and Judd Mineral Lodes (H&SC)

The parallel K1 and K2 lodes for the Kora deposit generally strike north-south (local grid) with a very steep dip to the west (Figure 14-2, Figure 14-3 and Figure 14-4). However, there are subtle undulations for both dip and strike in both lodes, which may be due to the ductile nature of the structural zone (from wrench fault tectonics) and/or possibly very minor cross-cutting and/or oblique offsetting faults. There is a divergence of the K1 and K2 lodes at the northern and southern ends of the mineral system and at the upper and lower margins. Overall the general geometry of the of the Kora mineral zone is that of a tilted sigmoid with a sub-horizontal/plunging down to the north long axis striking 5° east of north with the northern and southern tails striking north-south. The geological interpretation extends approximately 75 m to 100 m along strike and down dip beyond the last mineral intercept, sufficient for grade interpolation and the provision of target areas for further exploration and possible resource extension. The sub-parallel Kora Link zone is a relatively small area compared to the K1 and K2 lodes by having a much shorter strike and dip length; the zone generally dips steeply to the west in its southern and upper parts and dips almost vertical in its northern and lower parts.

Plan	Cross Section
	

Cyan = K1 Lode; Blue = K2 Lode; Red = Kora Link; drillhole traces & face sampling included

Figure 14-2 Plan and Cross Section of the Kora Mineral Lodes (H&SC)

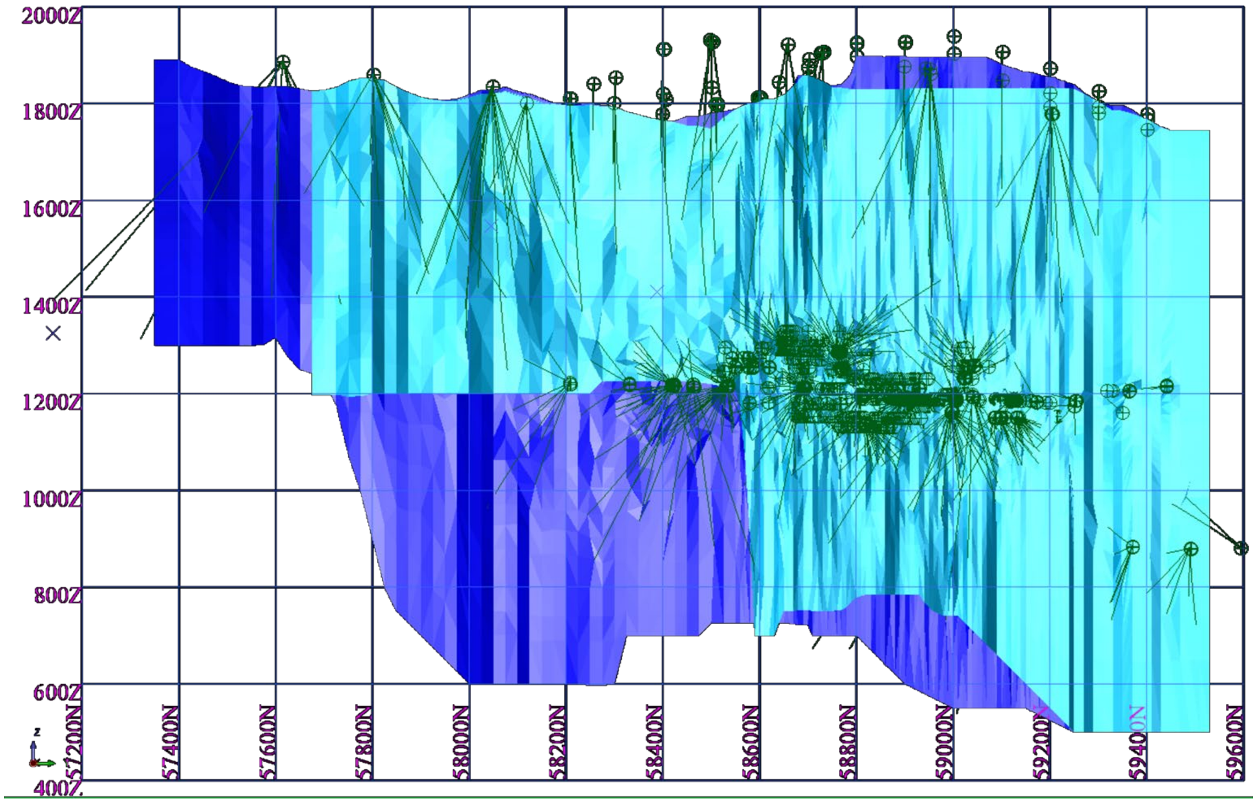


Figure 14-3 Long Section of the Kora K1 and K2 Mineral Lodes - looking west (H&SC)

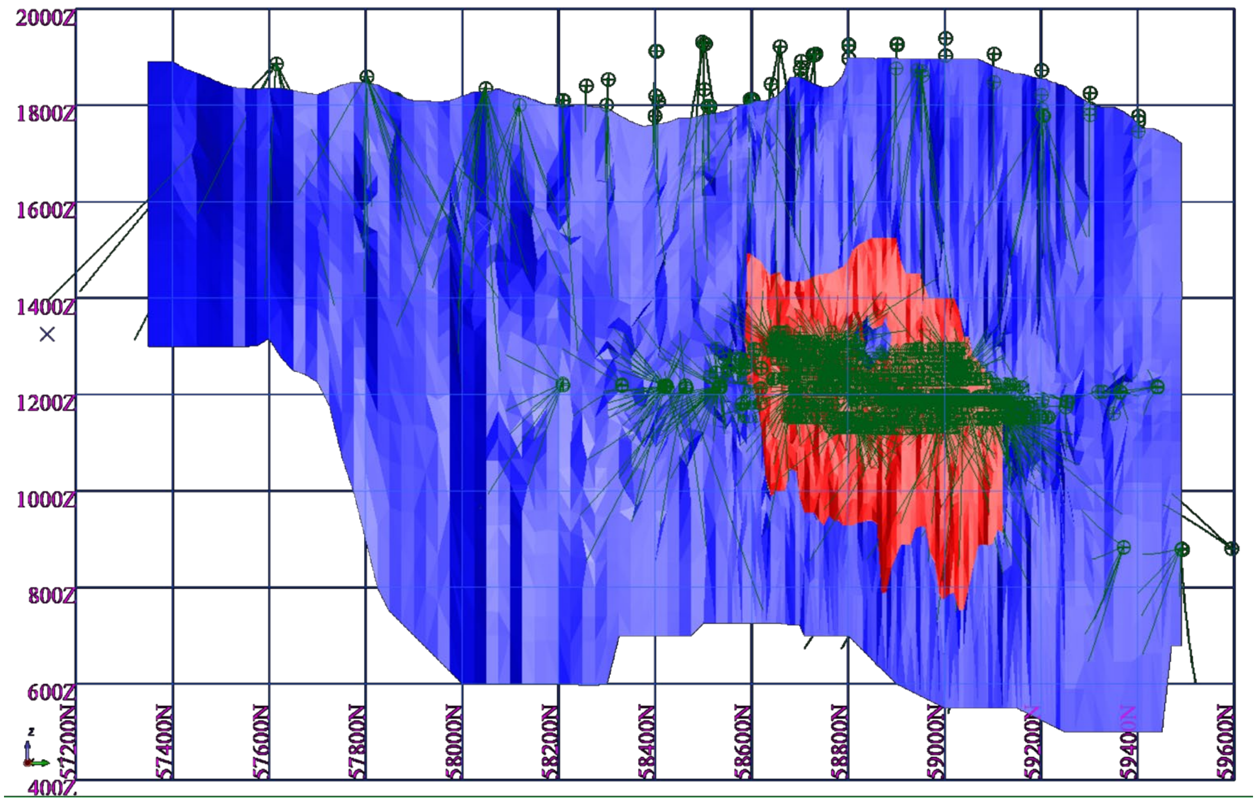


Figure 14-4 Long Section of the K2 and Kora Link Mineral Lodes - looking west (H&SC)

Figure 14-5 shows the comparison of the 2021 K1 mineral interpretation (cyan colour) with the 2023 interpretation (grey). Figure 14-6 shows the comparison of the 2021 K2 mineral interpretation (blue colour) with the 2023 interpretation (grey).

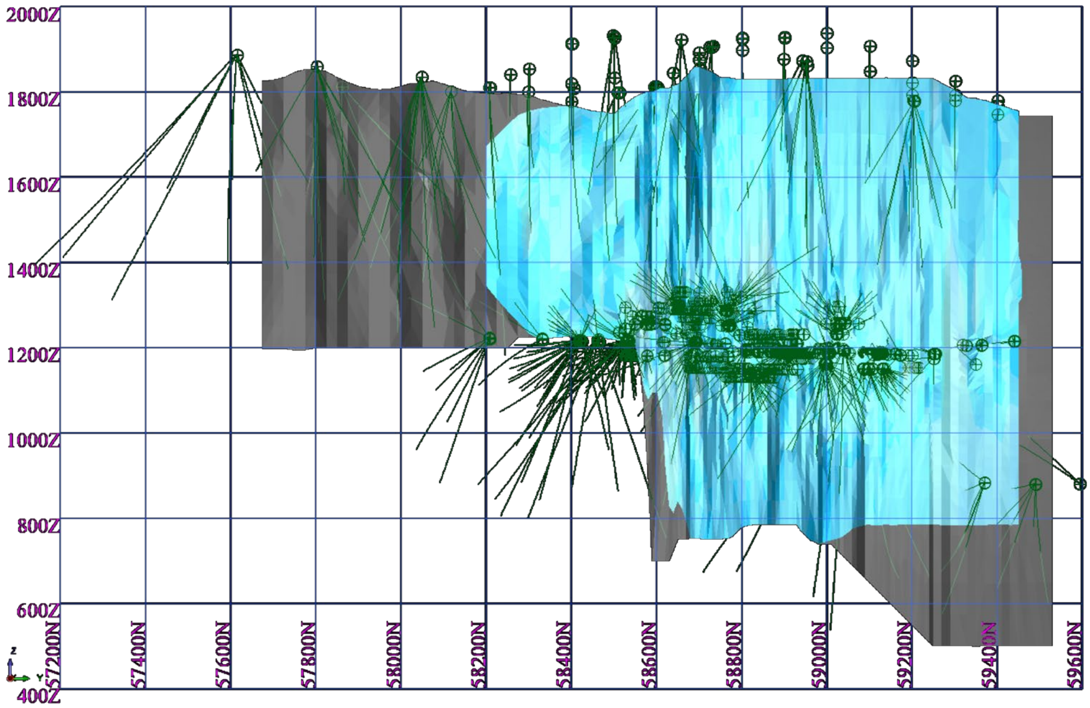


Figure 14-5 Long Section Comparison of the 2021 and 2023 K1 Mineral Lodes - looking west (H&SC)

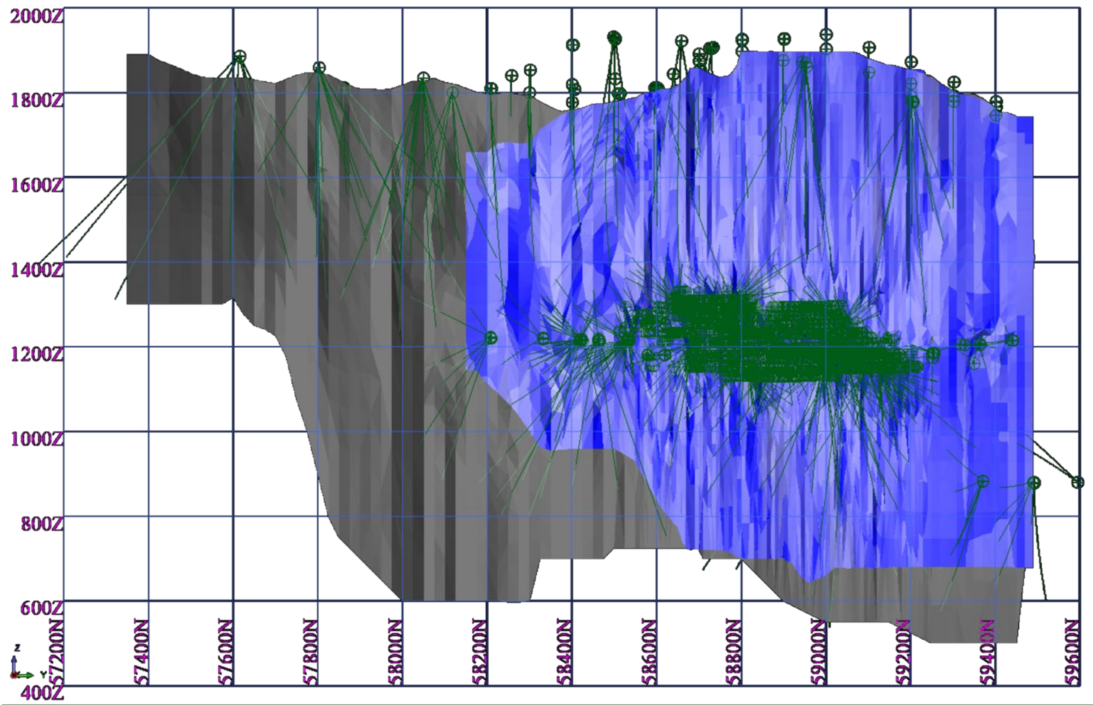
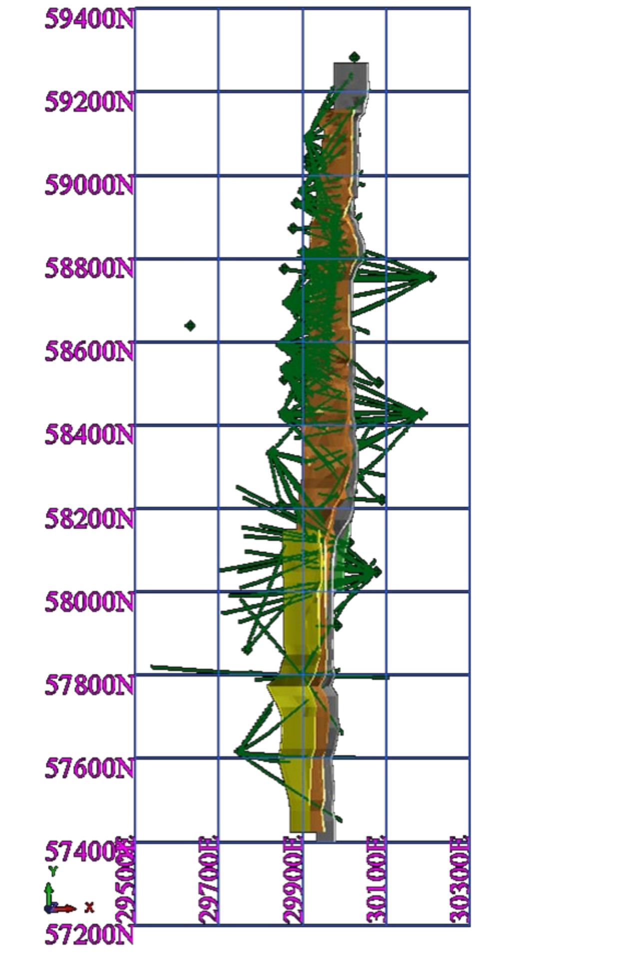
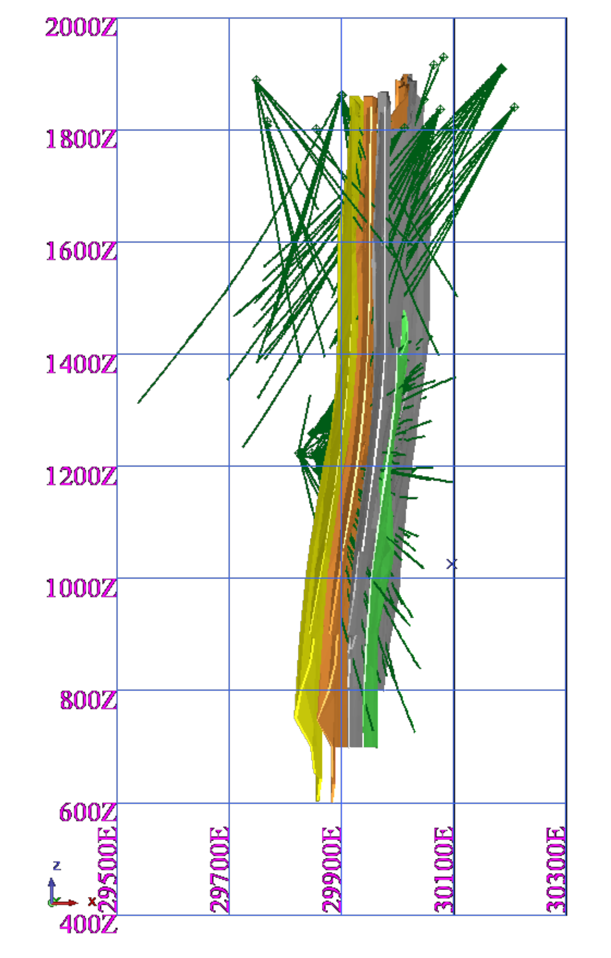


Figure 14-6 Long Section Comparison of the 2021 and 2023 K2 Mineral Lodes - looking west (H&SC)

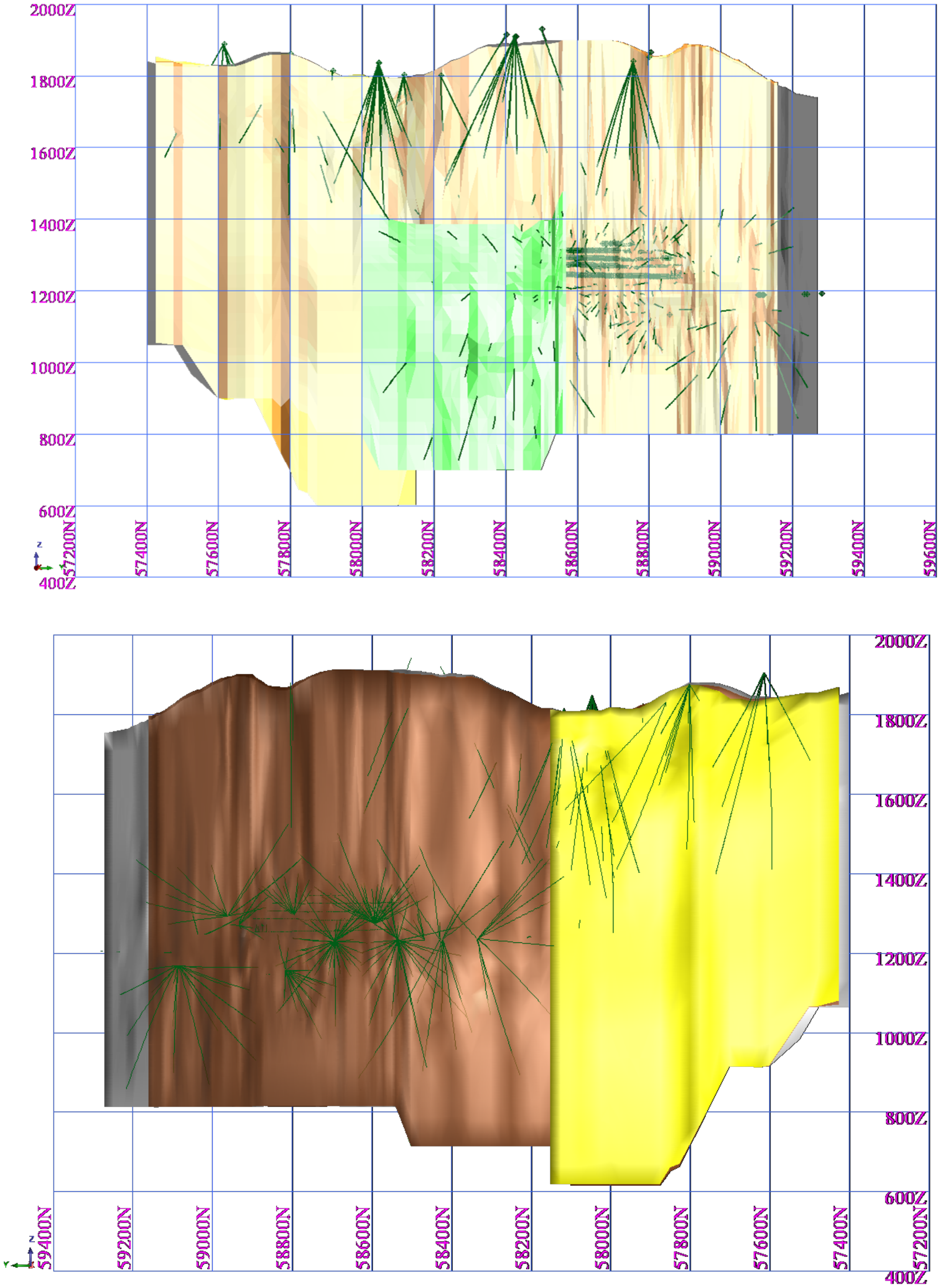
The advantage of the wireframing technique for K1 and K2 is to first avoid over-constraining of the gold assay data, as the contacts are not always sharp, and hence avoid potential overstatement of the gold grade in any subsequent grade interpolation. The use of high grade domains is considered by H&SC for this type of deposit to be a higher risk strategy, working on the fundamental principle that the higher the grade the shorter the continuity. Second, the lode shapes better reflect what is actually mined and sent to the mill for processing, which will allow for the opportunity of better reconciliation. There is no obvious evidence at the south end that K1 that it combines with K2; K1 tends to just peter out. Comparison with the previously reported Irumafimpa mineral wireframes strongly suggests that the original mining at Irumafimpa was not on either of the K1 or K2 lodes. Both lodes also tend to narrow slightly both up and down dip of the current underground mining area although occasionally there were localised ‘blow out’ areas in both grade and width for both lodes.

The geological interpretation for the Judd lodes has increased in complexity at the request of K92ML, with the introduction of three other lodes, J2, J3 & J1W. The design of the J1 lode wireframe is relatively straightforward especially where there is face sampling control. The southern end of the lode has been defined albeit with lower confidence due to a lack of drilling and the presence of multiple drillhole options for the lode intersection. The current configuration is considered a best fit to date but may change with further drilling. Figure 14-7 and Figure 14-8 show the orthogonal 2D views of the lode interpretation relative to the drilling. The J2, J3 and J1W are simply parallel lodes to J1 and are primarily grade shells developed from mineral intercepts in the appropriate positions.

Plan	Cross Section
	

(Brown = J1 lode; Grey = J2 lode; Green = J3 lode; Yellow = J1W lode; drillhole traces & face sampling included)

Figure 14-7 Plan and Cross Section of the Judd Mineral Lodes (H&SC)



(Brown = J1 lode; Grey = J2 lode; Green = J3 lode; Yellow = J1W lode; drillhole traces & face sampling included)

Figure 14-8 Long Section of the Judd Mineral Lode - looking west(upper), looking east (lower) (H&SC)

Figure 14-9 shows the comparative change in the J1 lode size in long section from 2021 to 2023. The liberal interpretation for the lower southern section is designed to help with further exploration targeting.

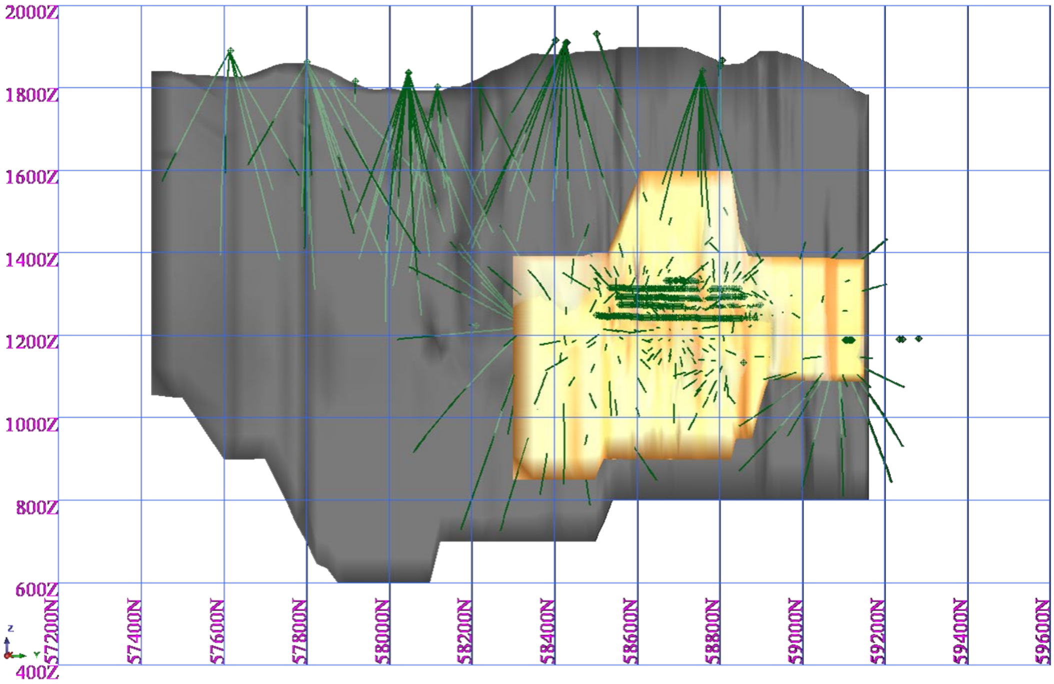


Figure 14-9 Long Section Comparison of the 2021 and 2023 J1 Mineral Lodes - looking west (H&SC)

Dimensions of the mineral zones are listed in Table 14-4. The increase in volume for all previously reported lodes is due to the increase in size interpreted from the recent drilling. Average widths for the lodes have been maintained from the 2021 work with a slight decrease in the Kora Link lode due to the extension of the lode into areas where the gap between K1 and K2 narrows.

Table 14-4 Dimensions of the Mineral Lodes

Lode	Strike (m)	Dip (m)	Ave Width (m)	Volume (m3)
K1	1,850	530-1,330	3.8	7,087,651
K2	2,150	550-1,350	4.8	10,346,432
Kora Link	500	600	7.7	2,320,615
J1	1,740	930-1,180	3.7	6,811,021
J2	1,870	960-1,180	3.0	6,032,444
J3	570	690	3.2	1,243,798
J1W	725	800-1,200	3.6	2,792,093

Figure 14-10 is a cross section example of the geological interpretation for Kora using both drilling and face sample data. The image shows the K1 lode (red dash), the K2 lode (green dash) and the Kora Link lode (light brown dash) abutting K1 HW and K2 FW. The figure also shows the logged vein on one side of the drillhole trace along with the gold grade as a scaled coloured bar on the other side. The development drives and stopes are shown as coloured solids. The lengths of the colour bars are intended to be relevant to the gold grade, but gold grades are so high that the maximum length has been limited at 50 g/t.

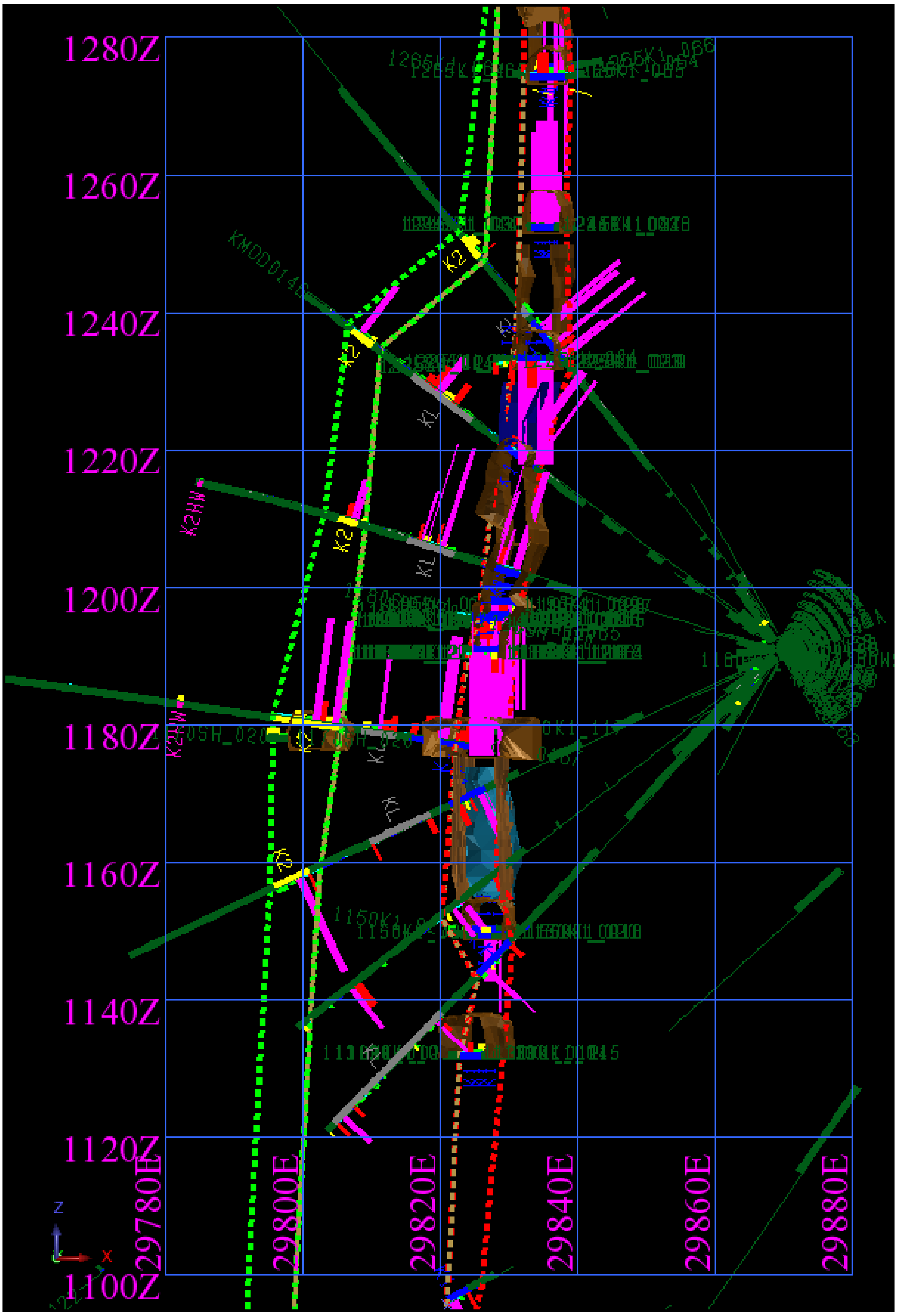


Figure 14-10 Geological Interpretation for Kora North   Cross Section 58900mN (H&SC)

Figure 14-11 is a cross section example of the geological interpretation for Judd using both drilling and face sample data. The image shows the J1 lode (green dash) and the J2 lode (red dash) along with drive and stope development (as solids). The figure also shows the logged vein on one side of the drillhole trace along with the gold grade as a scaled coloured bar on the other side. The lengths of the colour bars are intended to be relevant to the gold grade, but gold grades are so high that the maximum length has been limited at 50 g/t.

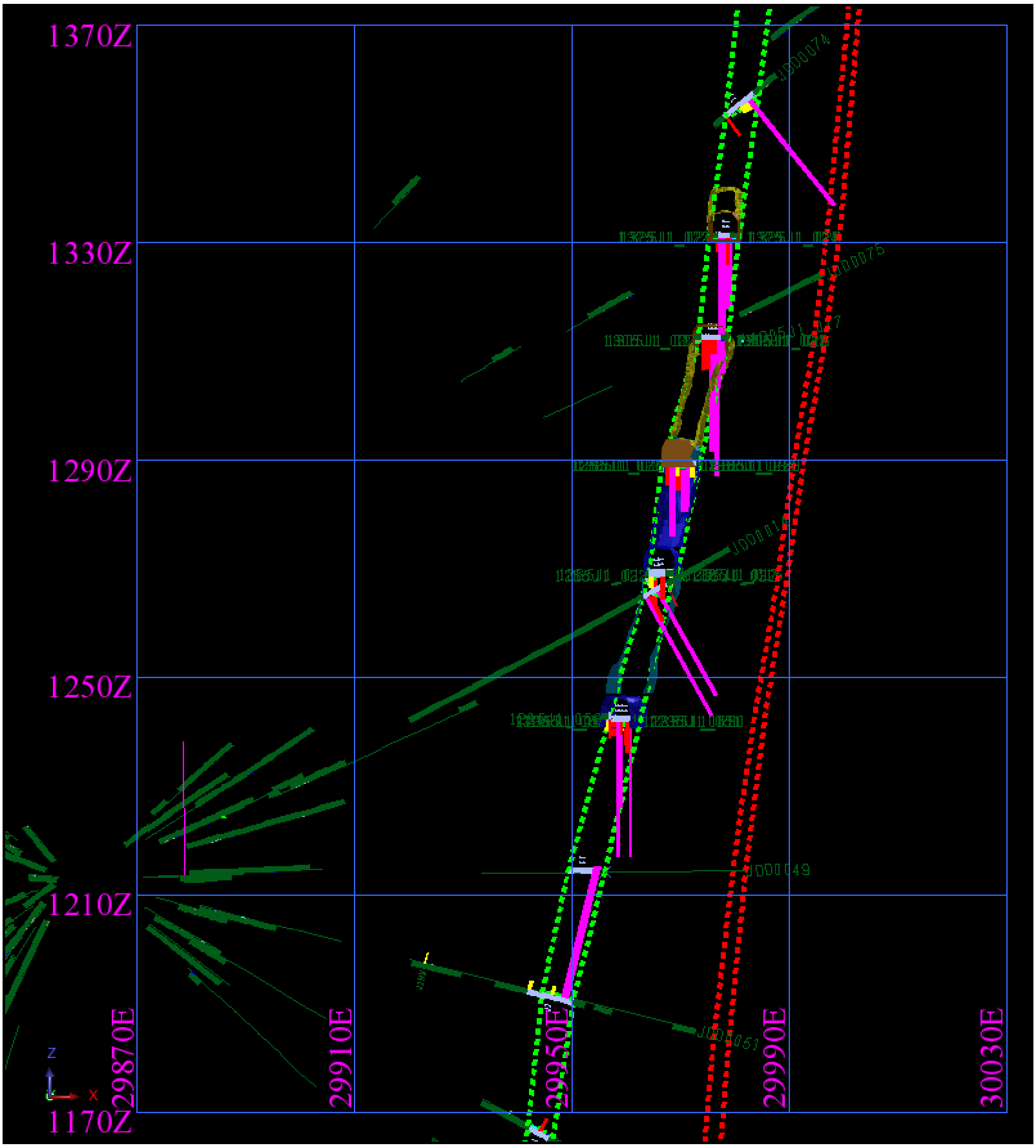


Figure 14-11 Geological Interpretation for Judd Cross Section 58640mN (H&SC)

Drillhole spacing is of the order of 10-25 m in the general underground mining area expanding to approximately 100 m for the rest of the mineral lode.

14.3 Data Analysis

Sampling was under geological control with a minimum sampling width of 0.1 m and a nominal maximum of 2 m. The smaller sample intervals were utilised to sample individual sub-veins/stringers and sulphide intercepts. Core was sampled to at least 5 m either side of each mineral lode, including any stringer style mineralization away from the lodes. There are minor zones of unsampled material in the Kora Link lode but the lack of data, presumably not considered mineralized, has limited impact as often the surrounding sample grades were low grade.

Figure 14-12 shows the range of sample intervals for the K1 lode wireframe with the dominant sample interval being 1 m. The number of samples for the lode has increased by approximately 50% compared to the 2021 model, this is due to the extra drilling and face sampling data.

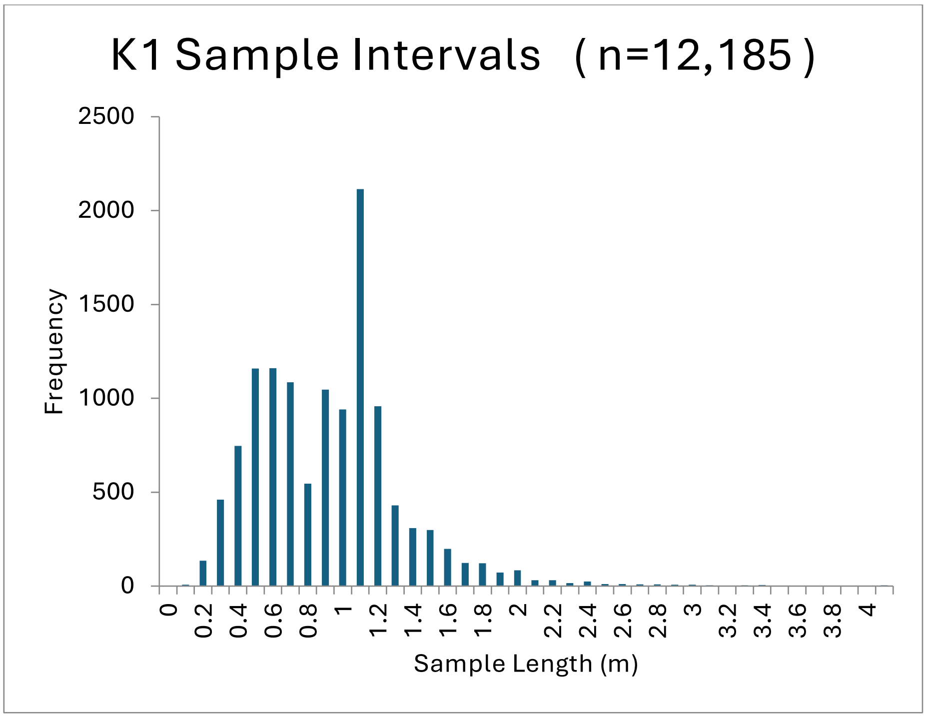


Figure 14-12 Sample Interval Histogram for the K1 Lode (H&SC)

Figure 14-13 shows the range of sample intervals for the K2 lode. It is very similar to the data for the K1 Lode. Again, the number of samples for the lode has increased by approximately 50% compared to the 2021 model, this is due to the extra drilling and face sampling data.

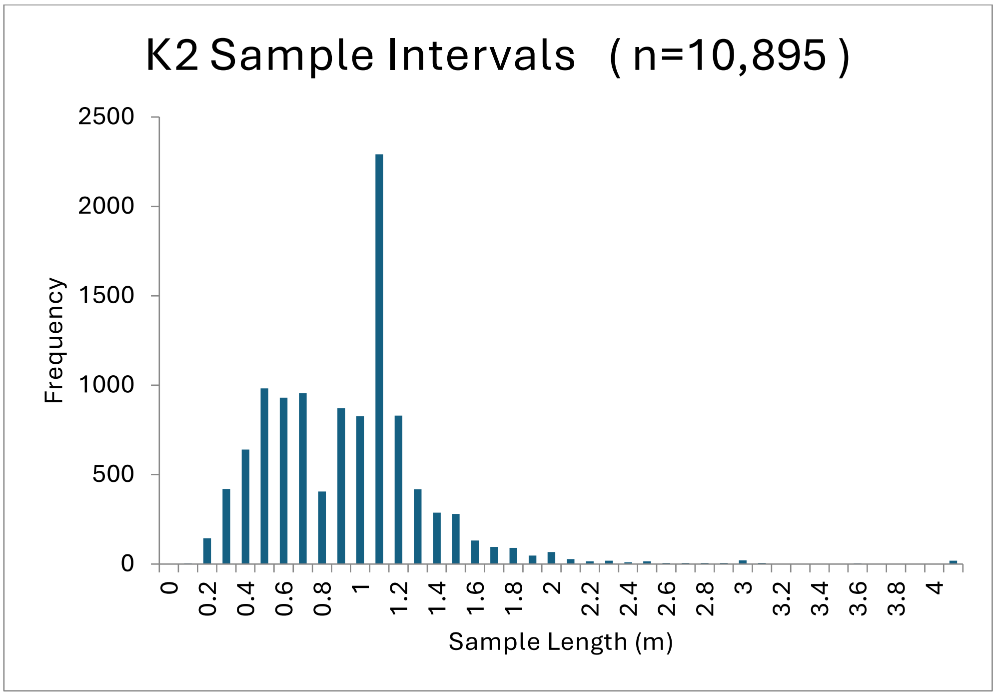


Figure 14-13 Sample Interval Histogram for the K2 Lode (H&SC)

A similar pattern is observed for the Kora Link Lode (Figure 14-14) with a 40% increase in the number of samples compared to the 2021 model.

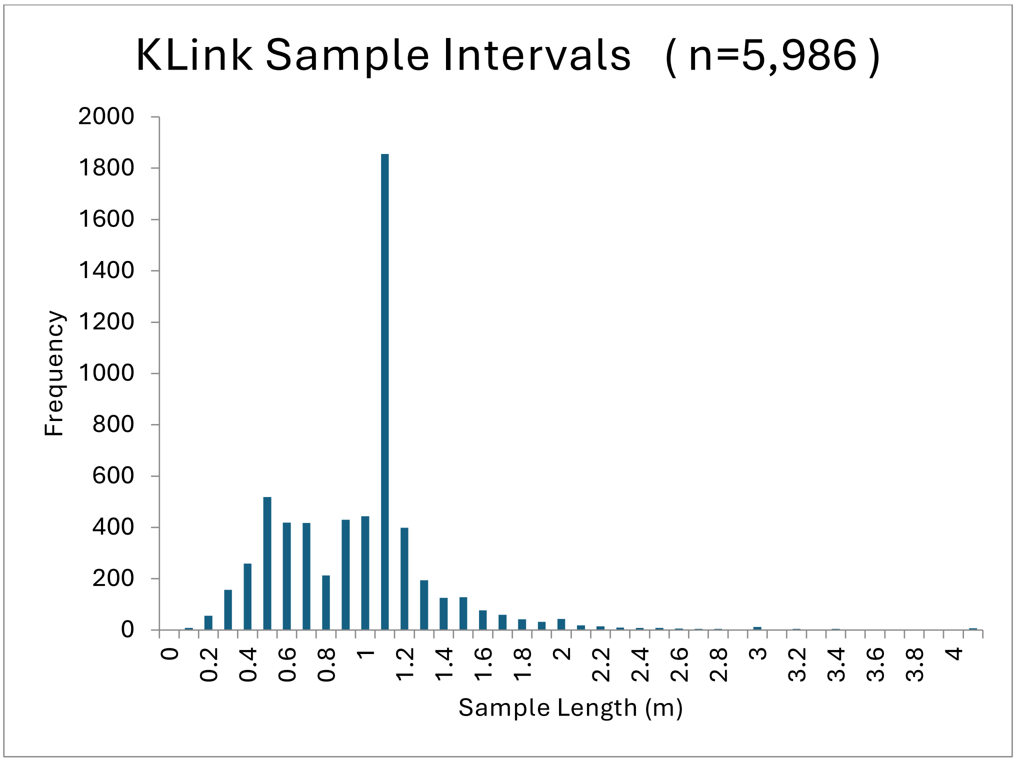


Figure 14-14 Sample Interval Histogram for the Kora Link Lode (H&SC)

A similar pattern is observed for the J1 Lode (Figure 14-15) with an almost trebling in the number of samples compared to the 2021 model. Similar graphs occur for J2 (Figure 14-16), J3 (Figure 14-17) and J1W (Figure 14-18) lodes representing new lodes. In all cases the dominant sample interval is 1 m.

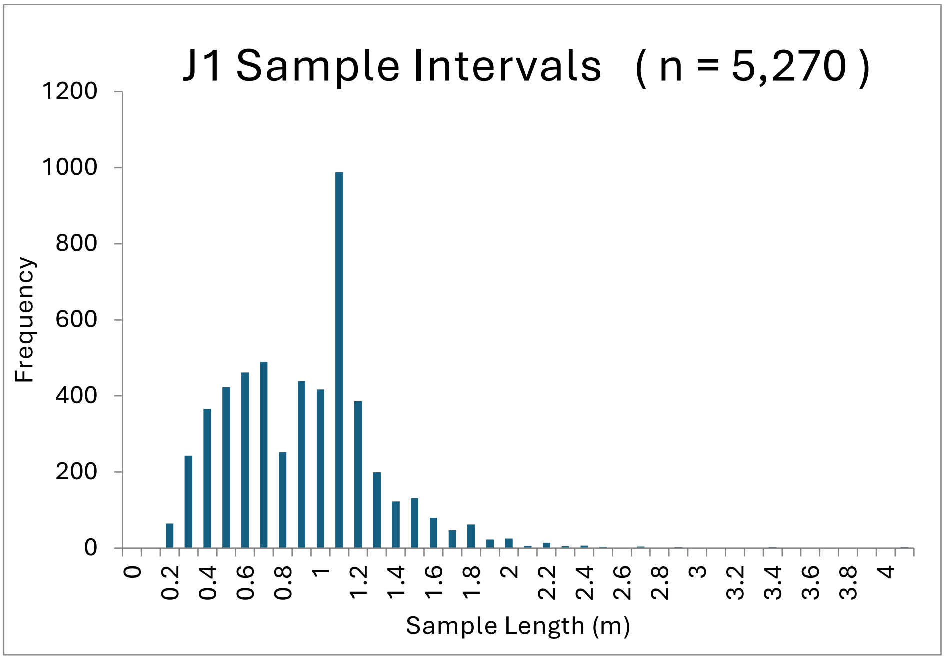


Figure 14-15 Sample Interval Histogram for the J1` Lode (H&SC)

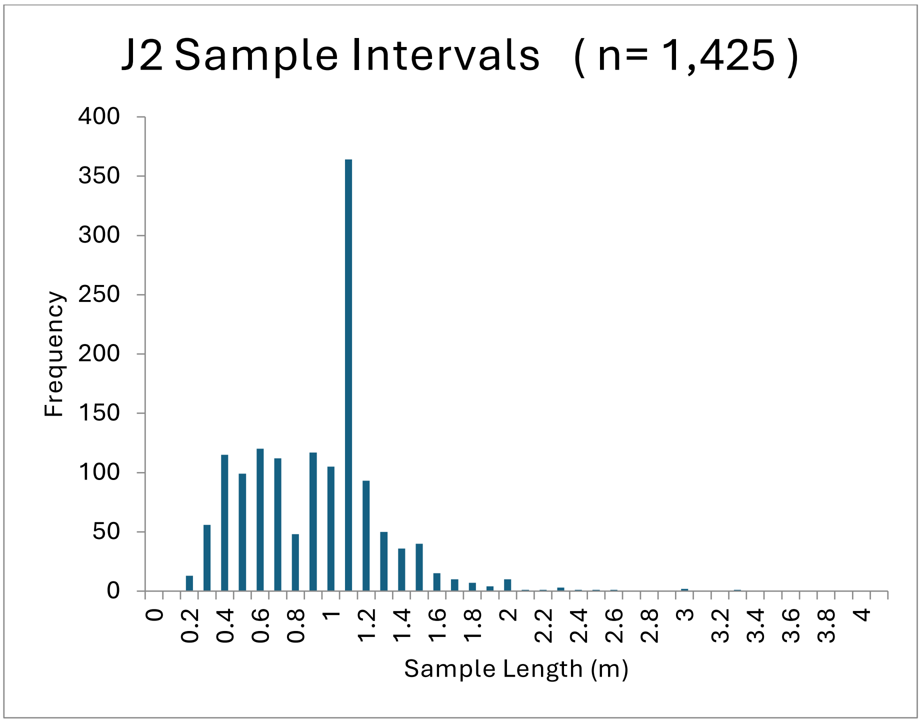


Figure 14-16 Sample Interval Histogram for the J2 Lode (H&SC)

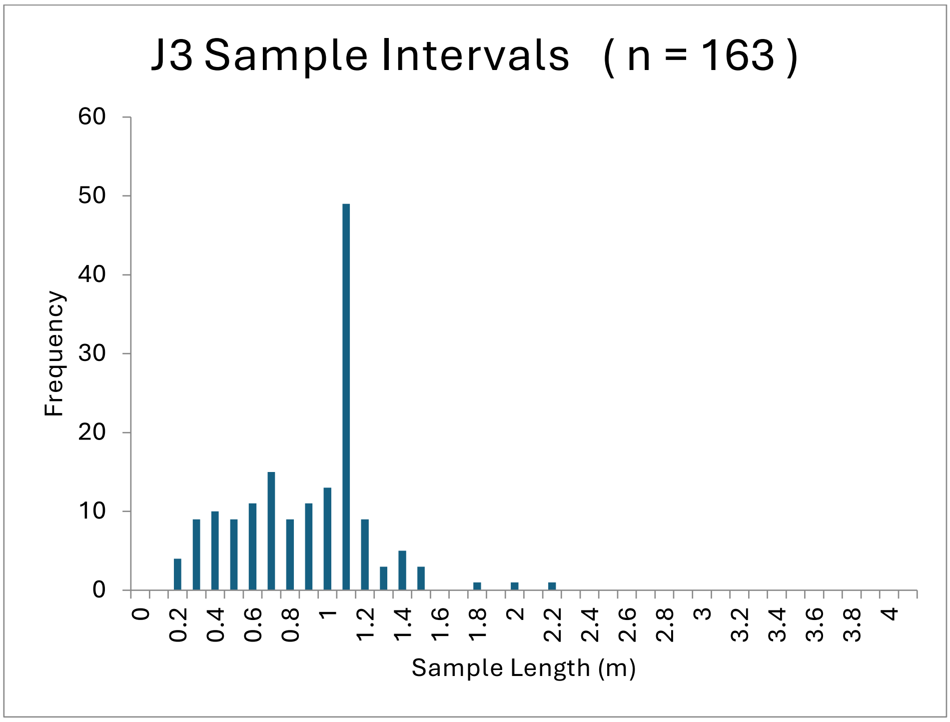


Figure 14-17 Sample Interval Histogram for the J3 Lode (H&SC)

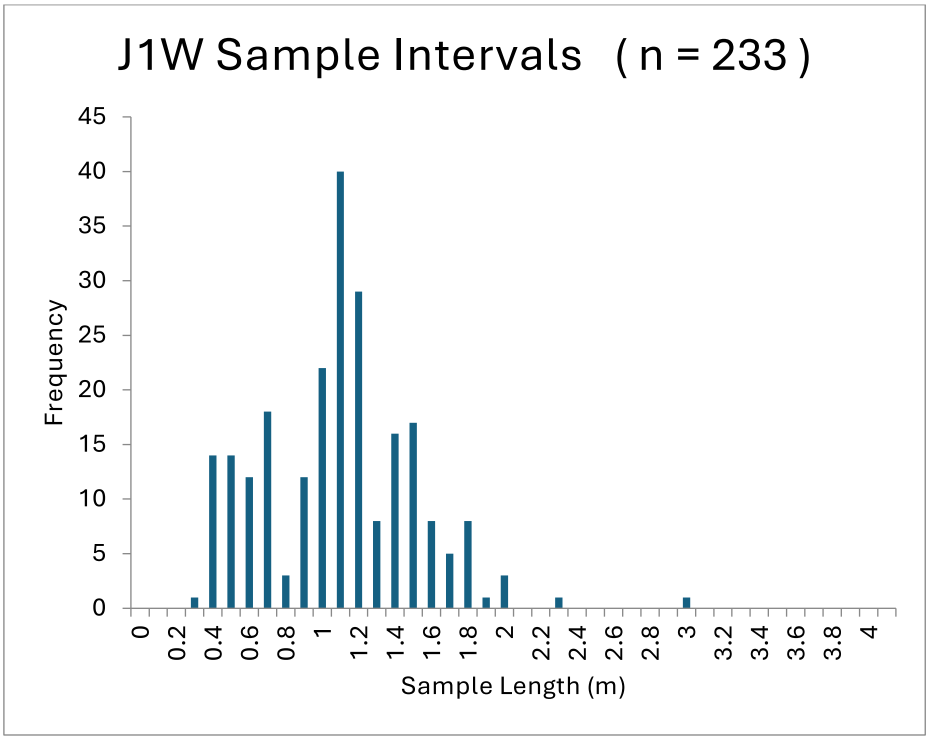


Figure 14-18 Sample Interval Histogram for the J1W Lode (H&SC)

The above diagrams were used to decide that a 1 m composite interval would be appropriate for subsequent block grade interpolation. The wireframes were used to generate the 1 m composites within the mineralized zones using the "best fit" sample length option in the Surpac mining software. This option allows for the equalising out of small residual sample lengths into the composite intervals so as to reduce the number of discarded residuals that are less than 0.5 m in length. A total of 24,929 composites for the Kora lodes were extracted from the drillhole & sampling database for gold, copper and silver. This represents a 44% increase on the number used in the 2021 resource estimates. For the Judd Lodes a total of 5,860 composites were extracted, with 4,342 composites for J1 representing a 210% increase in number of composites. Data consisted of both diamond core samples and face sampling with a very minor amount of sludge hole samples. The mineral wireframes represent hard boundaries for the grade interpolation.

Figure 14-19 shows the distribution of the uncut gold composite data for the K1 lode in long section (zoom in on the figure to get better resolution). This figure demonstrates the relatively close-spaced nature of some of the sampling, which is subsequently reflected in the chosen block size and resource classification. There appears to be a termination of higher gold grades in the south of the area (black dashed line marks the divide) which remains unexplained. The other item of note is the high-grade gold zone associated with the wider spaced drilling in the top central area of the section. From this in conjunction with the main samples area that there is a suggestion of a moderately steep north-plunging higher grade ore shoot from top left to bottom right.

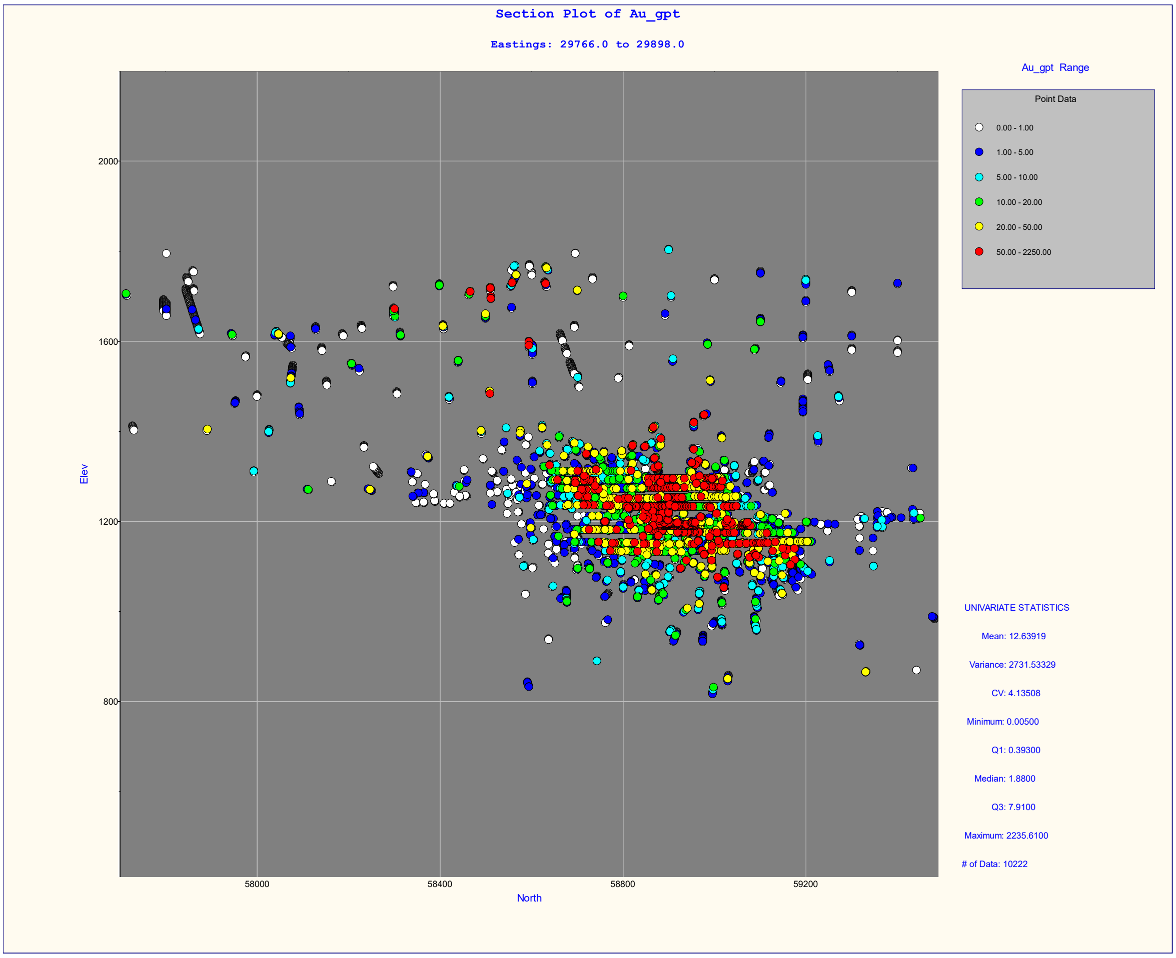


Figure 14-19 Gold Composite Distribution for the K1 Lode Long Section View (H&SC)

Figure 14-20 shows the same K1 long section view for copper composites, with a significantly higher grade copper zone associated with the old Kora deposit (black ellipse). The main Kora mine area is conspicuous by a significant amount of lower grade intercepts relative to the old Kora area. The truncation feature seen in the gold data is not apparent in the copper data. However, there is a hint of a copper truncation line at the southerly end of the lode (red dash).

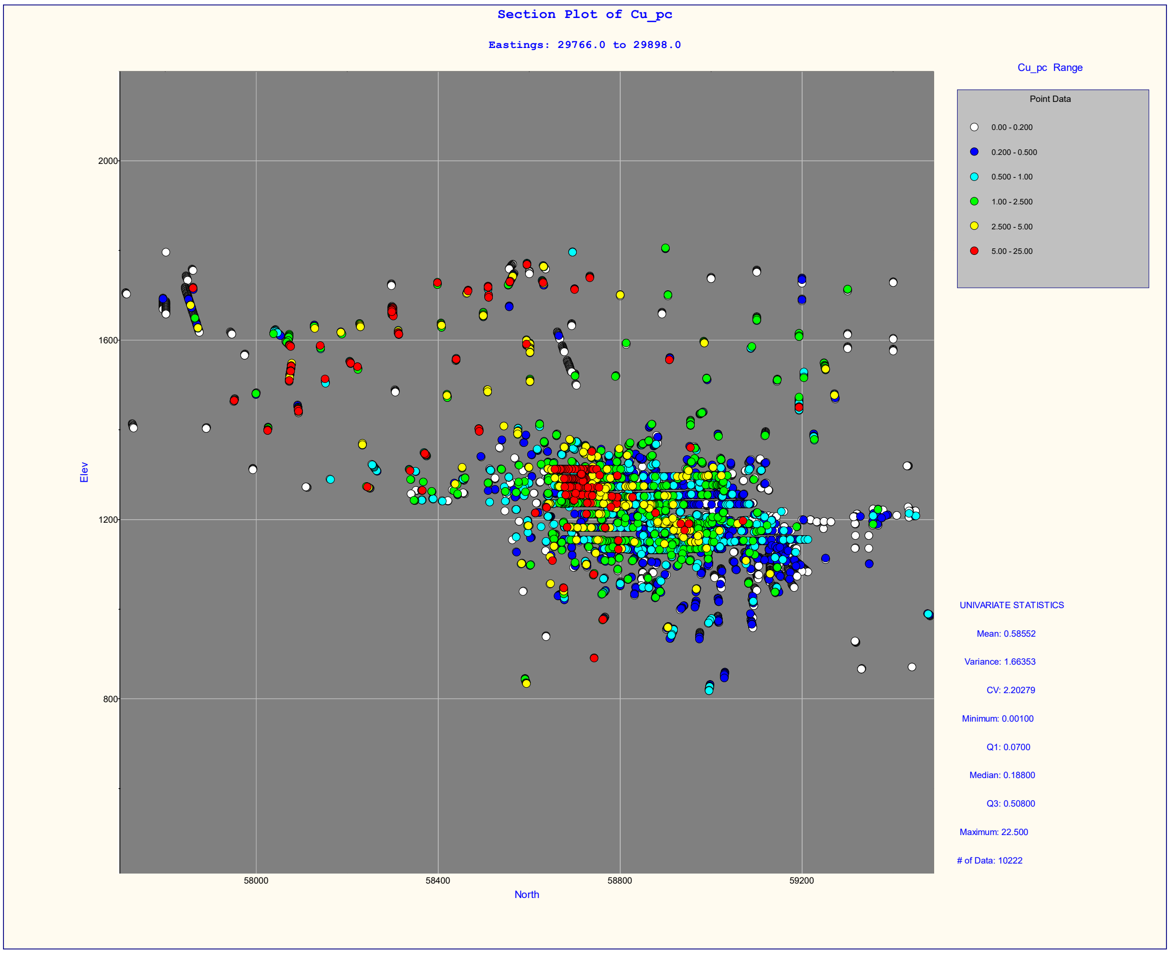


Figure 14-20 Copper Composite Distribution for the K1 Lode   Long Section View (H&SC)

Figure 14-21 shows the distribution of the uncut gold composite data in long section for the K2 lode. The grade continuity in the face sampling is much more limited than for the K1 lode and the distribution of the drillhole grades looks to be more random. There also appears to be a drop off in gold grade in the southern third of the deposit (black dash line indicates possible boundary), which is close to the same truncation line seen in the K1 gold data.

It should be noted that drilling has encountered significantly lower grade intercepts for K2 in the southern section relative to the general underground mining area.

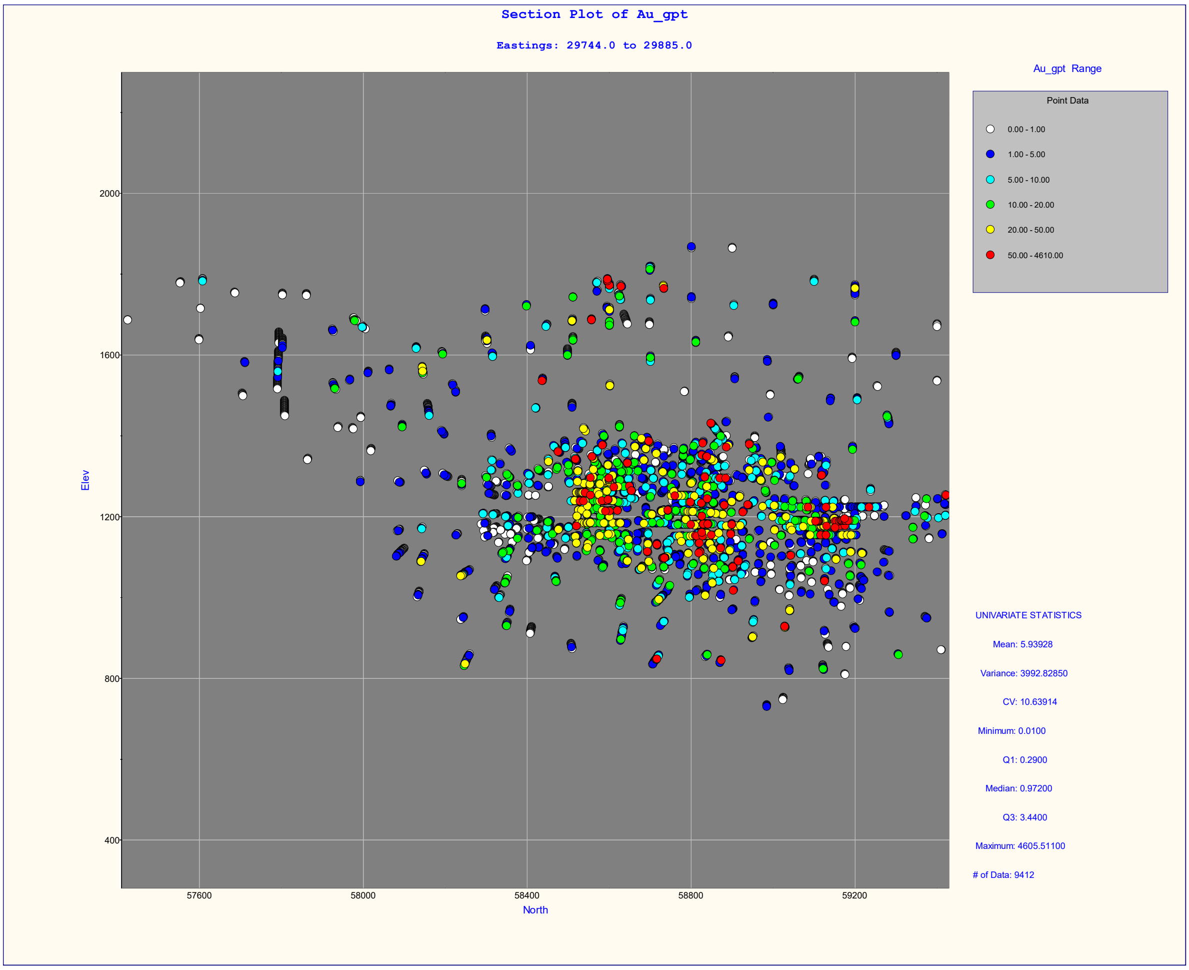


Figure 14-21 Gold Composite Distribution for the K2 Lode   Long Section View (H&SC)

The K2 copper composite grade distribution shows a moderately similar pattern to the K1 copper, especially for the old Kora enrichment zone (Figure 14-22). However there seems to be an abrupt termination of the enriched copper zone at the 58000mN line (red dash), which might suggest the presence of a cross cutting fault zone. This termination appears to be partially matched in the gold grade distribution which provides further evidence for an offsetting fault. Offsetting this is evidence of lode continuity from K1 and K92ML's preference for the continuity of lodes via their interpretation. It is possible that as there is considerable separation between K1 and K2 at the southern end then the offsetting for K2 may be considerably oblique to the lode strike and thus cutting K2 but not cutting K1.

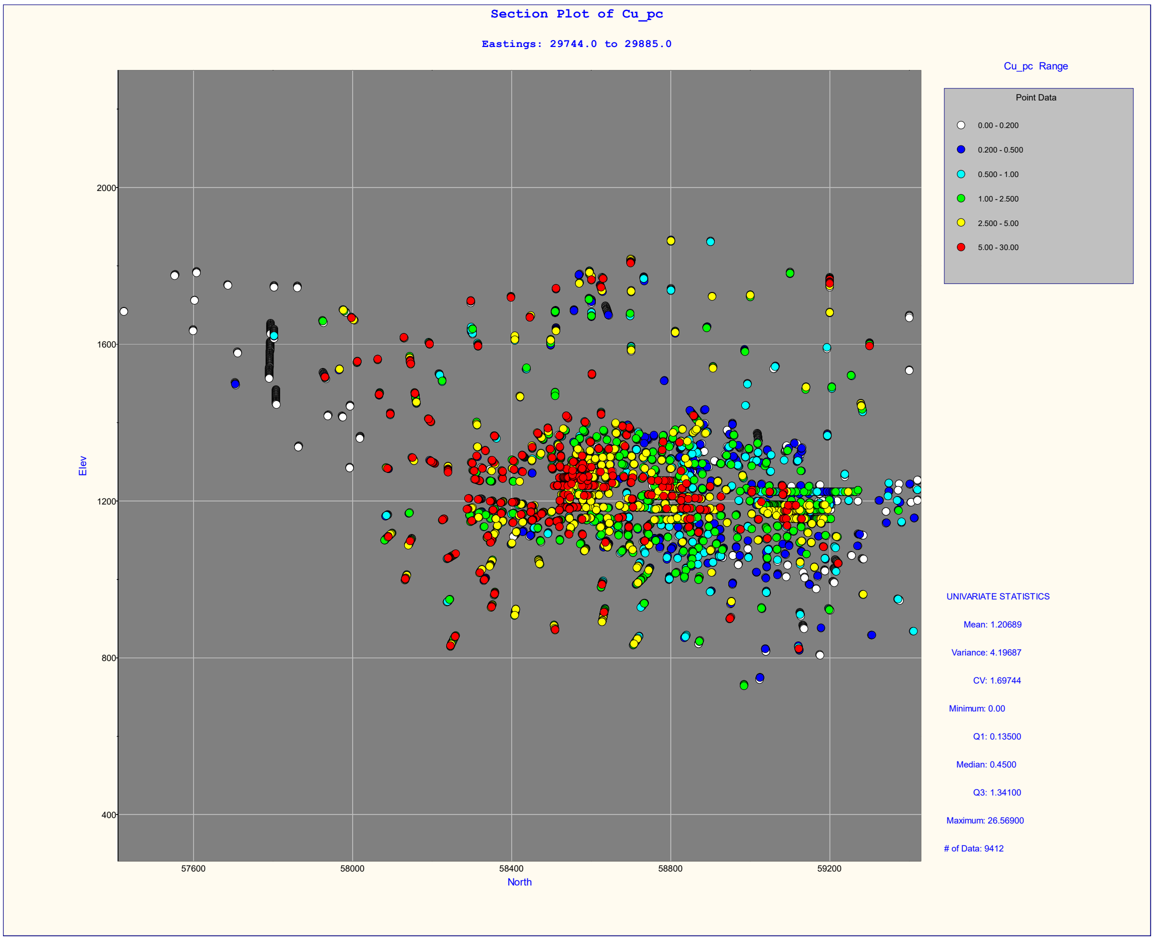


Figure 14-22 Copper Composite Distribution for the K2 Lode   Long Section View (H&SC)

Figure 14-23 shows a long section view of the uncut gold composites for the Kora Link lode. Gold grades are relatively sporadic in their distribution, although there is sufficient low-grade continuity and geological sense to justify interpreting the mineral domain.

In the 2021 estimate a higher section of K1, including face samples, had been interpreted as the Kora Link. This has now been corrected such that the higher grade in the 2022 report diagram is no longer present. The other item of note is that there is a zone where the K1 and K2 lodes appear to butt up against each other to the exclusion of the Kora Link (black ellipse in the diagram). As a result of both infill drilling (at lower grades around the margins) and the re-assignment of higher gold grade zone has caused the average gold grade for the lode to be decreased substantially.

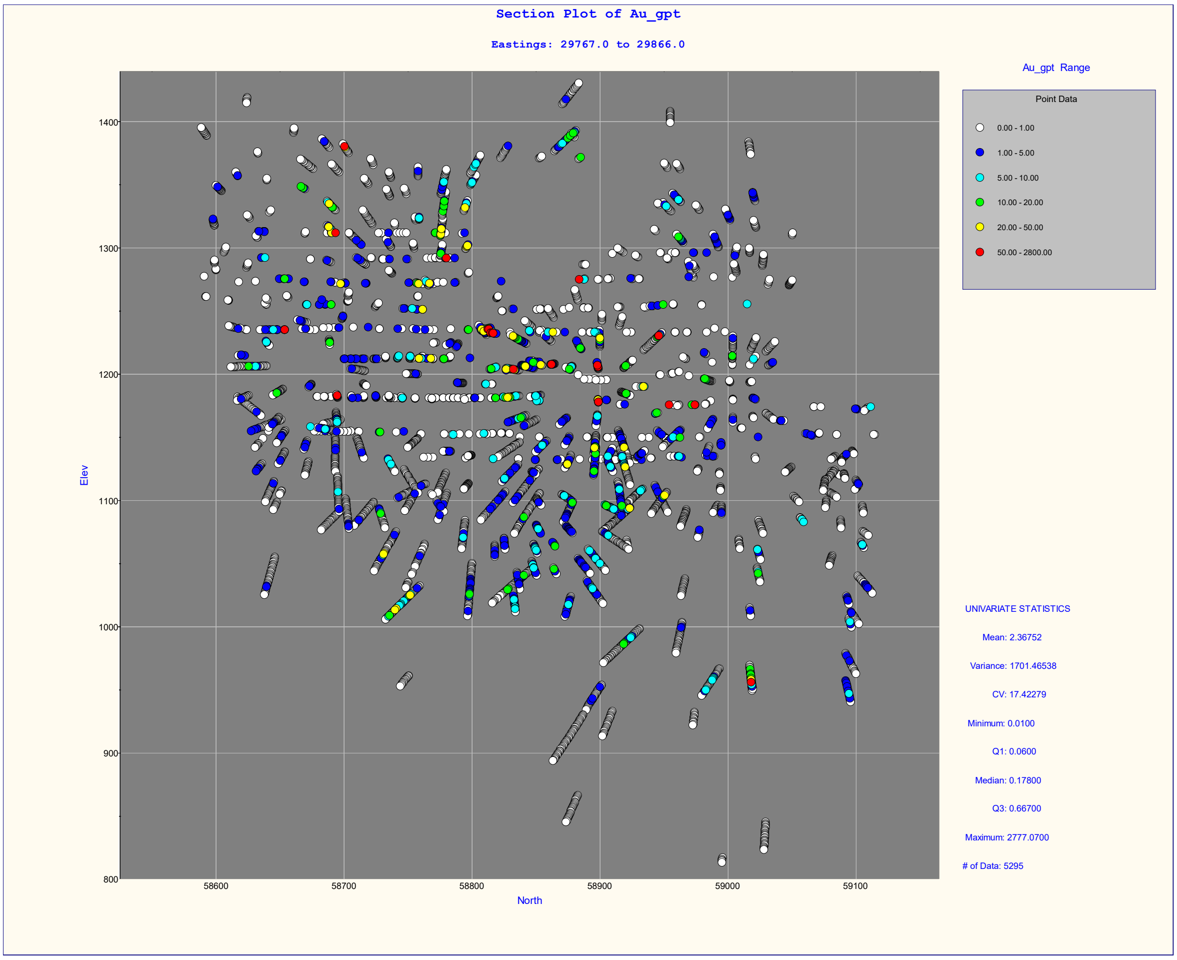


Figure 14-23 Gold Composite Distribution for the Kora Link Lode   Long Section View (H&SC)

Figure 14-24 shows a long section view of the uncut gold composites for the J1 lode at Judd. The relatively limited drilling seems to have defined an enriched gold zone that appears to drop off in gold grade at depth and to the north and south but appears to be still open vertically.

The black dashed line seems to represent a marked change to lower gold grades which may be due to a potentially offsetting fault or a marked change in lithology having both different rheological properties and propensity to fracturing.

Elsewhere there appears to be sufficient low-grade continuity to justify interpreting the mineral domain.

The copper data (Figure 14-25) shows a similar pattern to the gold grade distribution although the relatively abrupt southerly cut off seen in the gold data is not so obvious in the copper data. There does seem to be a steeper truncation line for the copper data that might line up with the copper data for K1 and K2. The potential for offsetting fault(s) may warrant further investigation.

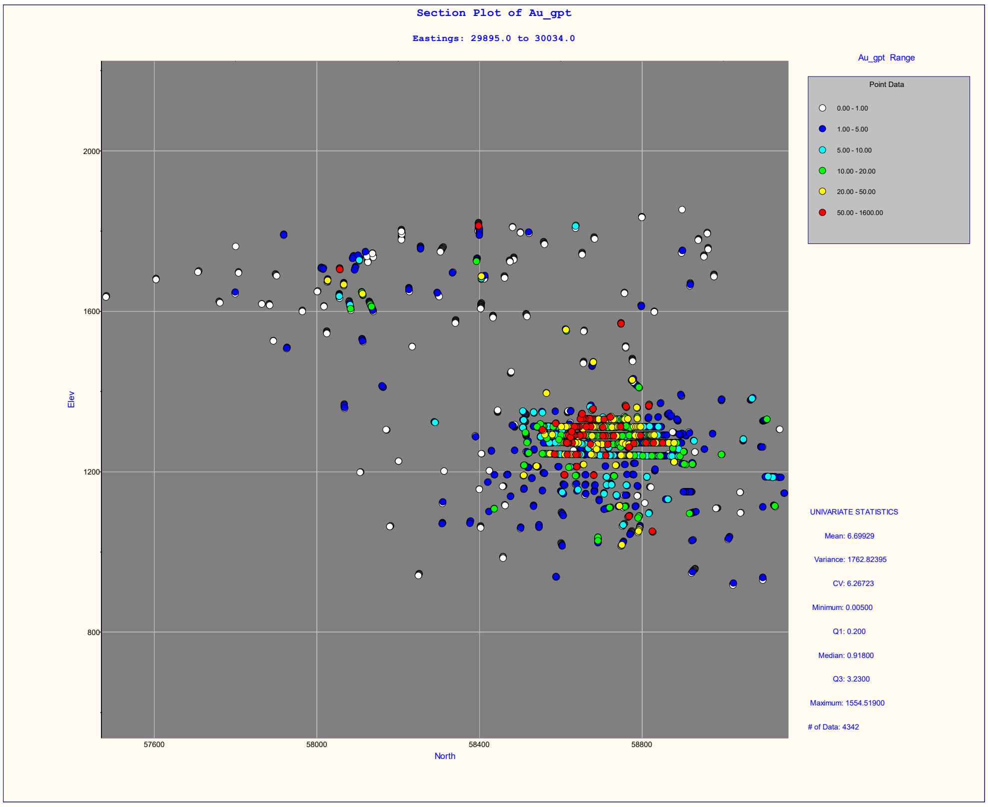


Figure 14-24 Gold Composite Distribution for the J1 Lode   Long Section View (H&SC)

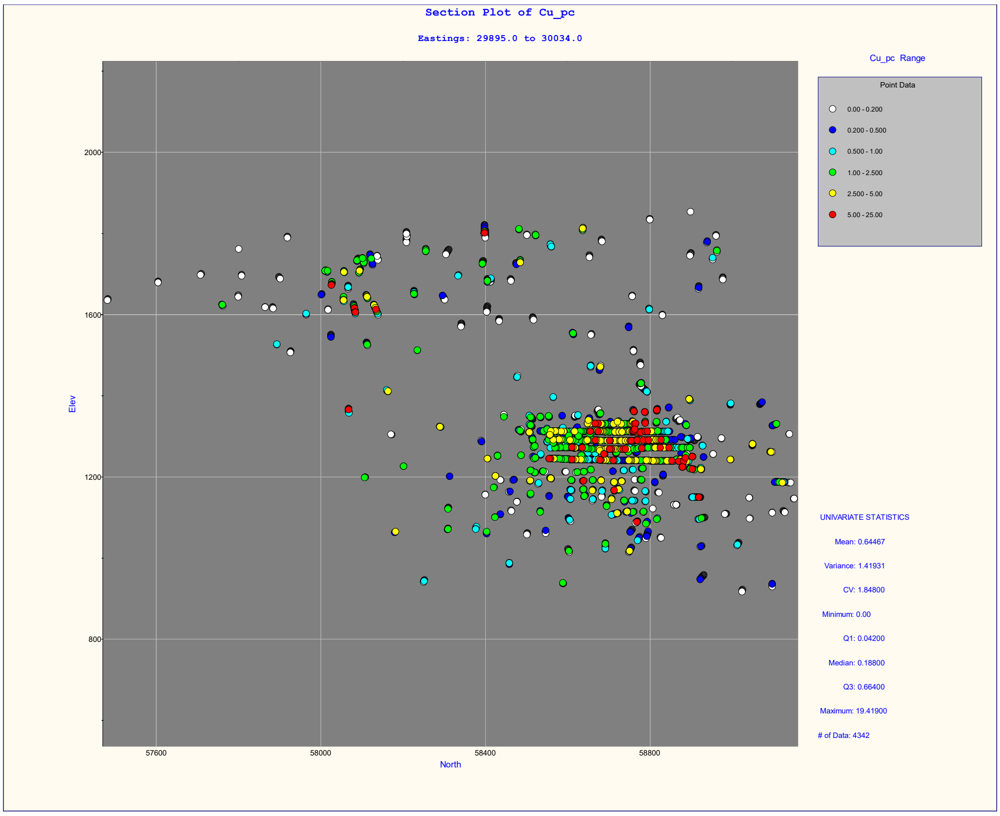


Figure 14-25 Copper Composite Distribution for the J1 Lode   Long Section View (H&SC)

The gold composite grade distribution for the J2 lode is shown in Figure 14-26. It comprises a similar pattern to J1 but is fundamentally at a considerably lower grade.

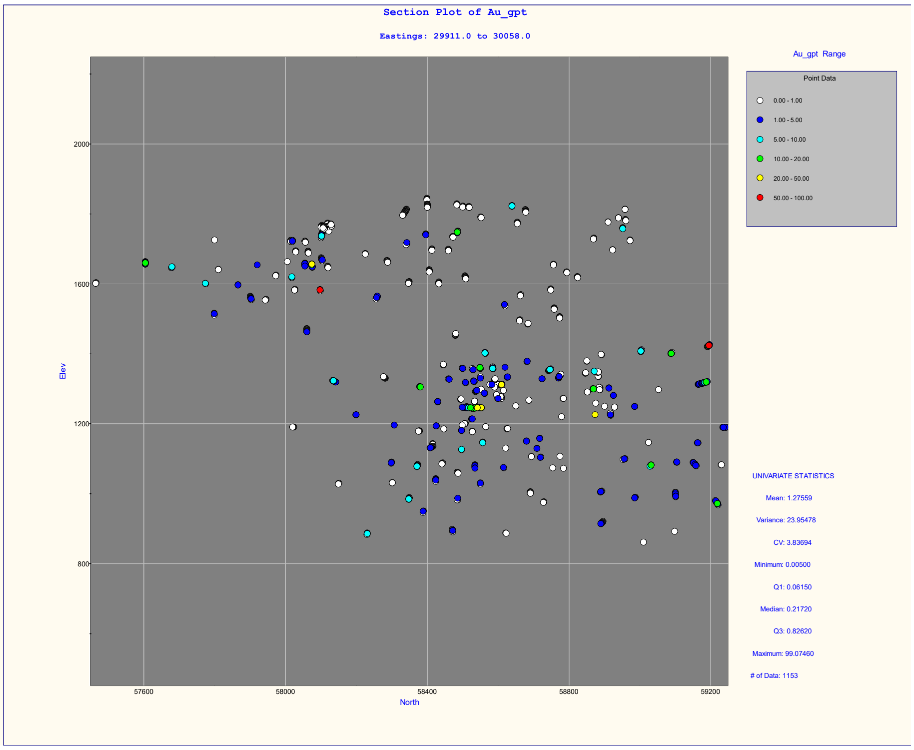


Figure 14-26 Gold Composite Distribution for the J2 Lode   Long Section View (H&SC)

The uncut gold composite grade distribution for J3 is consistent both in grade and spatial distribution but is low in gold grade. The uncut gold composite grade distribution for J1W suggests a possible steeply plunging lode beginning at the 58000mN line and potentially open vertically in both directions and possibly open to the north. The amount of drilling and the narrowness of the intercepts point to limited expansionary resource potential at this stage.

Missing data for the K1 composites i.e. zeros and blanks, were noted for copper and silver with very low grade default values being inserted for the zeroes (Table 14-5). Simple regression equations with gold were used for the inserting values for the copper and silver blanks. The missing data is generally due to a lack of assaying for copper and silver on samples that were originally perceived to be very low grade and significantly peripheral to the mineral lode interpretation made at the time of the drilling.

Table 14-5 Details of Insert Grades for the K1 Lode

		Sample Numbers	
Regressions		**Blanks**	**Zeroes**
Copper	Cu = 0.0401Au + 0.081	224	42
Silver	Ag = -0.0068Au2 + 1.0438Au + 1.4017	53	0
Defaults			
Copper	0.0005%		42
Silver	n/a		n/a

Previous work completed by H&SC for the 2018 and 2021 resource estimates mentioned that the distinct "difference between the gold means for the diamond core and face sampling types could be used to argue against combining the datasets. However, the face sampling is from mined material and is real data and it is quite evident that the [earlier] drilling 'missed' this high grade zone". This statement is still valid. A key feature of previous models was that reasonable reconciliation was achieved with production using all the data with no top cuts, primarily from the K1 lode, and this appears to have been sustained after a further 18 months or so of mining (again mainly of the K1 lode). Experimentation by applying top cuts to the data generally indicated that the top cuts had modest impact, except on certain noted occasions with extreme values.

H&SC prefers to apply minimal top cuts to composite data as firstly, applying top cuts adjusts real data and secondly often the threshold is arbitrarily decided without any statistical or geological validity. H&SC prefers to control any potential higher grade outliers through judicious use of the composite interval, grade interpolation parameters, variography and the geological interpretation. However, in some instances top cuts will be needed for extreme values.

Figure 14-27 shows the cumulative frequency curve for the uncut K1 gold composite data. It indicates that a possible top cut to the data could be applied at 2,000 g/t (black solid line in the figure), at 900 g/t (black dashed line in the figure) or at 600 g/t (black dotted line in the figure). None of the cuts are particularly convincing especially when comparing with the other lodes.

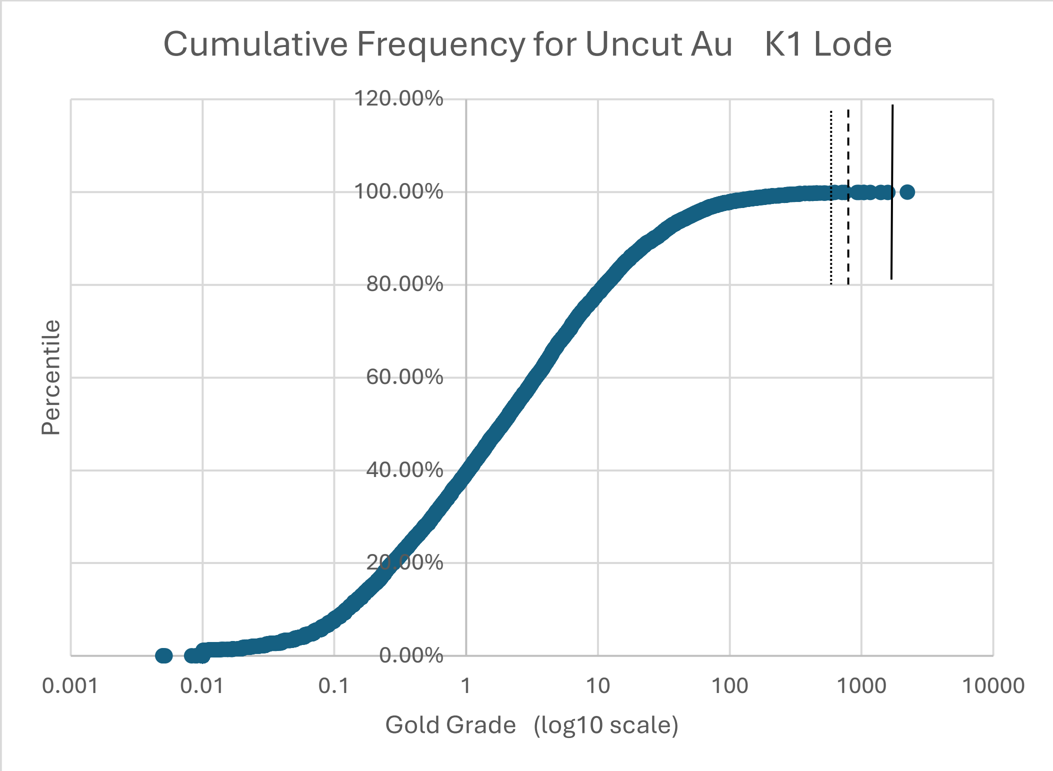


Figure 14-27 K1 Gold Composites Cumulative Frequency Curve (H&SC)

It is worth pointing out that the top three highest gold grades account for only 4% of the total gold content, which is much less than the other lodes e.g. K2. The top 11 highest grade samples for K1, which is a 600 g/t cut off, accounts for 10% of the gold content. However, the source for these top 11 samples is primarily from the face sampling (8 samples) and thus the impact of these extreme values is lessened by the surrounding data and precludes the need for a top cut. The remaining three high grades from the diamond drilling are within the face sampling zone and therefore their impact on grade interpolation is also diminished, negating the need for a top cut.

A top cut of 300 ppm, from previous work, was applied to the K1 silver data and affected 17 samples.

A histogram plot of the gold data indicates a lognormal distribution with a slight positive skew but overall, the histogram resembles the expectation for a single population (Figure 14-28). The smooth cumulative frequency curve indicates a single population.

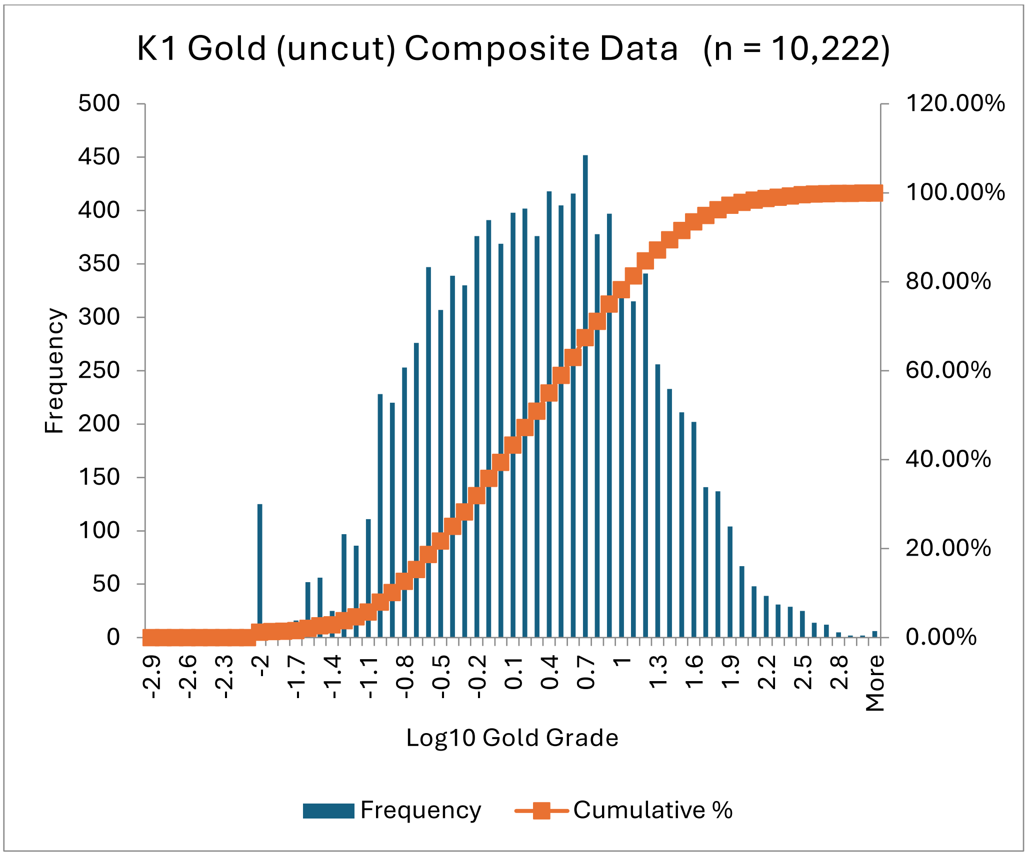


Figure 14-28 Histogram of K1 Uncut Gold Composite Data (H&SC)

Table 14-6 shows the summary statistics for the K1 lode. The data shows relatively modest coefficients of variation (CV = standard deviation/mean) for gold and copper, which might be considered a little surprising considering the type of mineralization but indicates a relative lack of skewed data and possibly a very limited number of data outliers and/or the data represents a single population. These relatively low CVs further suggest that no top cuts are required. This decision is also helped by the fact that the data around the current mining area is well structured. As part of an experimentation process top cuts of 600 g/t and 100 g/t were applied to the gold grades. The immediate result for the 600g/t cut was that it resulted in only a 5% drop in the mean gold composite grade. Silver (uncut) has a higher CV than gold that is due to two extreme values, experimental top cutting of these values showed a very significant drop in CV and a 10% drop in the mean grade.

The inclusion of the 100 g/t top cut is for the benefit of the K92ML mine geologists to see the impact of a very low top cut to help with grade control plans.

Table 14-6 Summary Statistics for the K1 Lode

All data	Gold	Gold (cut600)	Gold (cut100)	Copper	Silver	Silver (cut300)
Mean	12.64	12.07	9.43	0.59	10.42	9.55
Median	1.88	1.88	1.88	0.19	3.83	3.83
Standard Deviation	52.27	39.29	19.28	1.29	54.60	22.54
Coeff of Variation	4.14	3.26	2.04	2.20	5.24	2.36
Minimum	0.005	0.005	0.005	0.0005	0.01	0.01
Maximum	2,235.61	600	100	22.5	4171	300
Count	10,222	10,222	10,222	10,222	10,222	10,222

For K1, using the cut gold composite data, there is no correlation between gold and either copper or silver. However, there is a weak correlation between copper and cut silver (Table 14-7).

Table 14-7 Correlation Coefficients for the K1 Lode Composite Data

	Gold	Gold_tc600	Copper	Silver_tc300
Gold	1			
Gold_tc600	0.92	1		
Copper	0.07	0.1	1	
Silver_tc300	0.1	0.13	0.59	1

Missing data for the K2 composites, i.e. zeros and blanks were noted for copper and zeroes for silver. The missing copper and silver assays are due to the same reasons as for K1. The zeroes were replaced with a low-grade default value and the copper blank values were generated from a regression equation (Table 14-8).

Table 14-8 Details of Insert Grades and Top Cut for the K2 Lode

		Sample Numbers	
Regressions		**Blanks**	**Zeroes**
Copper	Cu = (0.0497*Au)+0.0207	139	25
Defaults			
Copper	0.0002		25
Silver	0.01		28
Top Cuts	Cut		Samples Affected
Gold	1,000 g/t		3
Silver	300 ppm		63

Figure 14-29 shows the cumulative frequency curve for the uncut K2 gold composite data. It indicates that possible top cuts to the data could be applied at 1,000 g/t, 600 g/t or 400 g/t (black solid, dashed and dotted lines in the figure). Application of the 1,000 g/t top cut affected three samples, all from the diamond drilling and all are widely scattered.

A top cut of 300 ppm was applied to the silver data and affected 63 samples.

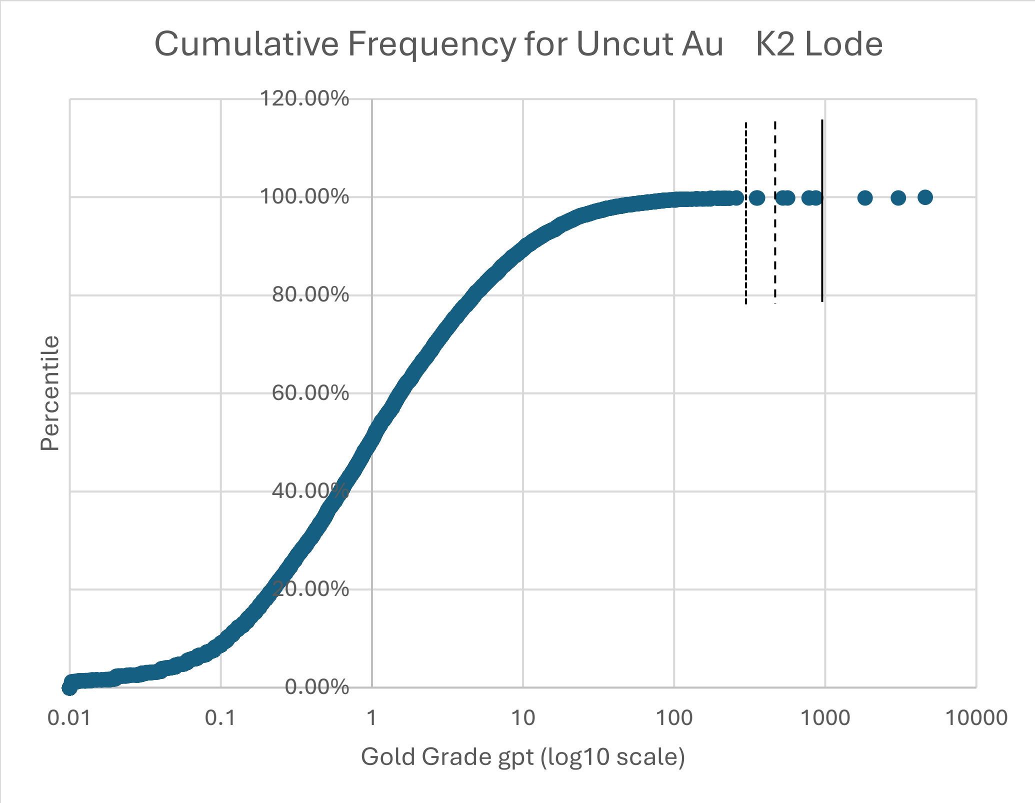


Figure 14-29 K2 Gold Composites Cumulative Frequency Curve (H&SC)

Using the cumulative frequency data above it is possible to state that without a top cut, 17% of the total gold in the composite data is attributable to three samples (the 1,000 g/t cut off). This increases to 20% for the top five samples (600 g/t cut off) and to 22% for the top 7 samples out of 9,412 total samples (the 400 g/t cut off). It is also important to note that nearly all the extreme values are located away from the immediate mine development in areas of limited drilling. The impact on the grade interpolation of these extreme values is likely to be high risk by considerably overstating the Mineral Resource in these areas.

Figure 14-30 shows the distribution of extreme values for the K2 lode in long section. The five highest grade samples (note 2 samples are approximately coincident in the uppermost area) are from drillholes, not face samples, and generally lie out with the general mine area such that their impact with respect to mining/reconciliation is unknown. Samples shown in black circles will have a major impact on surrounding block gold grade interpolation via over-statement of gold grade. The sample in the orange circle, whilst being very high grade, is likely to have a more muted impact on the surrounding block grades on account of the abundance of surrounding lower grade data points. The problem has been managed in the resource classification without using a lower top cut e.g. the 600 g/t or 400 g/t top cut, but in H&SC's opinion this is not desirable.

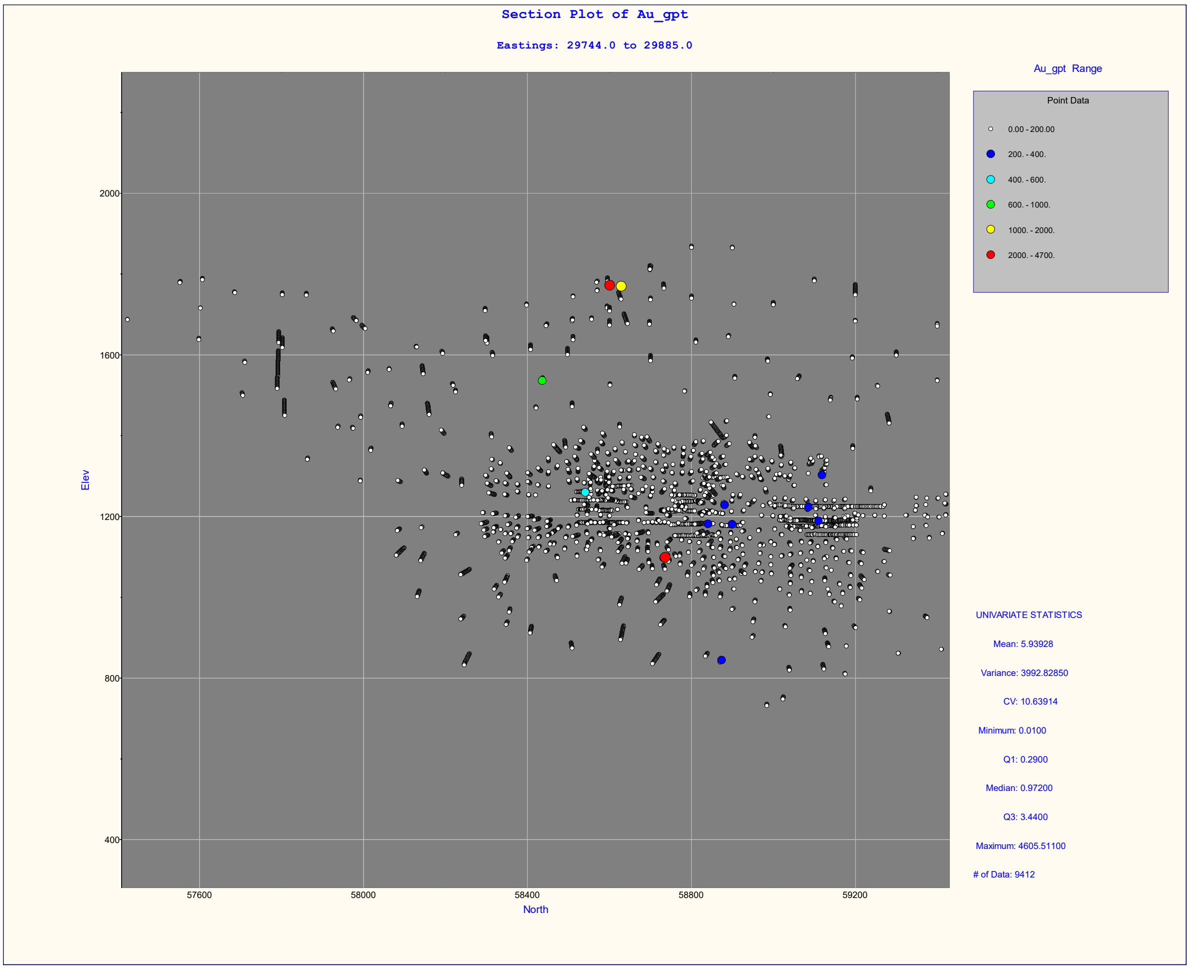


Figure 14-30 Extreme Grade Gold Composite Distribution for the K2 Lode   Long Section View (H&SC)

A histogram plot of the K2 gold composite data indicates a reasonable lognormal distribution with a slight positive skew but overall, the histogram resembles the expectation for a single population (Figure 14-31). The smooth cumulative frequency curve indicates a single population.

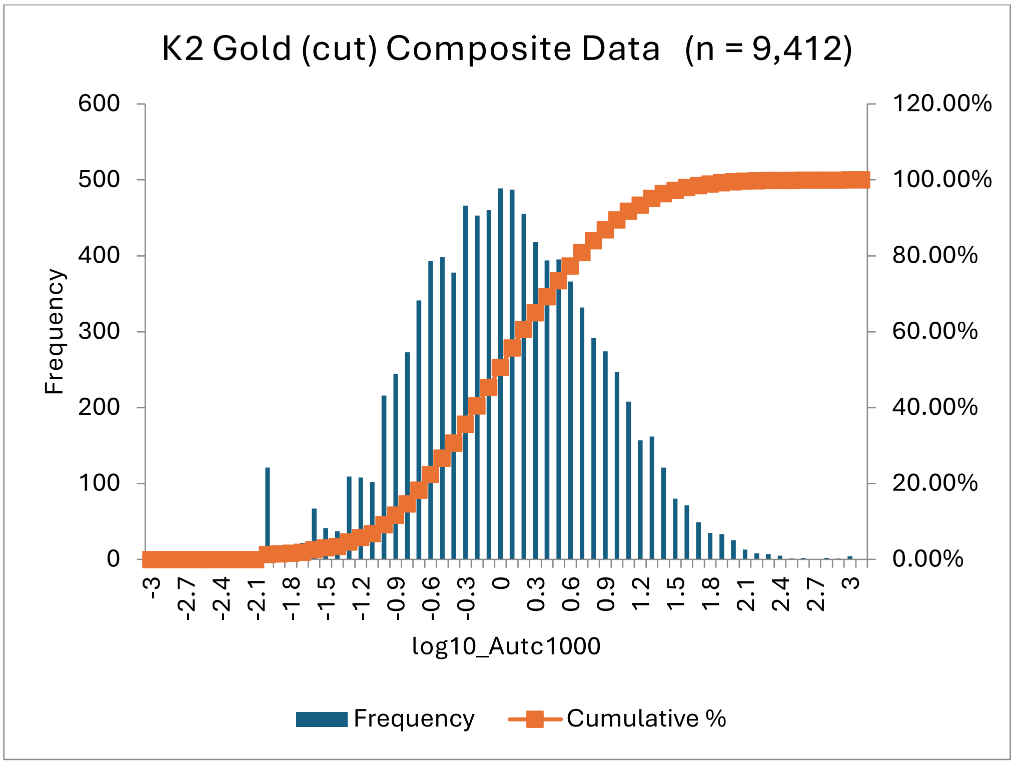


Figure 14-31 Histogram of K2 Gold Composite Data (H&SC)

Table 14-9 shows the summary statistics for the K2 lode. Previous work completed by H&SC indicated relatively little difference between the gold, silver and copper means for both datatypes (diamond core and face sampling), which suggest that the datatypes can be combined. This statement is still valid.

Table 14-9 Summary Statistics for the K2 Lode

	Gold	Gold (cut1000)	Gold (cut600)	Gold (cut100)	Copper	Silver	Silver (cut300)
Mean	5.94	5.25	5.07	4.47	1.20	23.99	21.82
Median	0.97	0.97	0.97	0.97	0.45	8.00	8.00
Std Dev	63.19	26.80	21.14	11.26	2.05	68.45	40.73
Coeff of Var	10.64	5.10	4.17	2.52	1.70	2.85	1.87
Minimum	0.01	0.01	0.01	0.01	0.0002	0.01	0.01
Maximum	4,605.511	1,000	600	100	26.5686	2,577.855	300
Count	9,412	9,412	9,412	9,412	9,412	9,412	9,412

The impact of applying the 1,000 g/t top-cut for gold is significant as whilst it affected only three samples, it generated an 11% drop in the overall mean value of the composites. Likewise, the CV for the uncut gold data is 10.6, with the 1,000 g/t top cut that CV is halved to 5.1. H&SC completed some experimentation work using a 600 g/t and a 100 g/t top cut which indicated a 15% and 25% drop respectively in the mean gold grade relative to the uncut data and reductions in CVs to 4.2 and 2.5 respectively.

The combined drill hole and face sampling data shows relatively low CVs for both copper and silver, which strongly suggests there is no need for top cutting. However, it was decided to use the 300 ppm silver top cut to lessen the impact of extreme grades.

There is no significant correlation between gold and the other two elements as shown in Table 14-10. There is a modest correlation between copper and silver.

Table 14-10 Correlation Coefficients for the K2 Lode Composite Data

	Gold	Gold_tc1000	Copper	Silver_tc300
Gold	1			
Gold_tc1000	0.83	1		
Copper	0.04	0.10	1	
Silver_tc300	0.19	0.29	0.65	1

Missing data for the Kora Link composites, i.e. zeros and blanks, were noted for copper only (Table 14-11). The missing copper assays are due to the same reasons as for K1. The blanks and zeroes were replaced with a low grade default value.

Table 14-11 Details of Insert Grades for the Kora Link Lode

		Sample Numbers	
Regressions		**Blanks**	**Zeroes**
Copper	n/a	5	29
Defaults			
Copper	0.0002		34
Silver	n/a		n/a

Figure 14-32 shows the cumulative frequency curve for the uncut Kora Link gold composite data. It indicates that possible top cuts to the data could be applied at 400 g/t (black solid line) or 100 g/t (black dash line). At K92ML's request no top cut was applied as their impact was likely to be small.

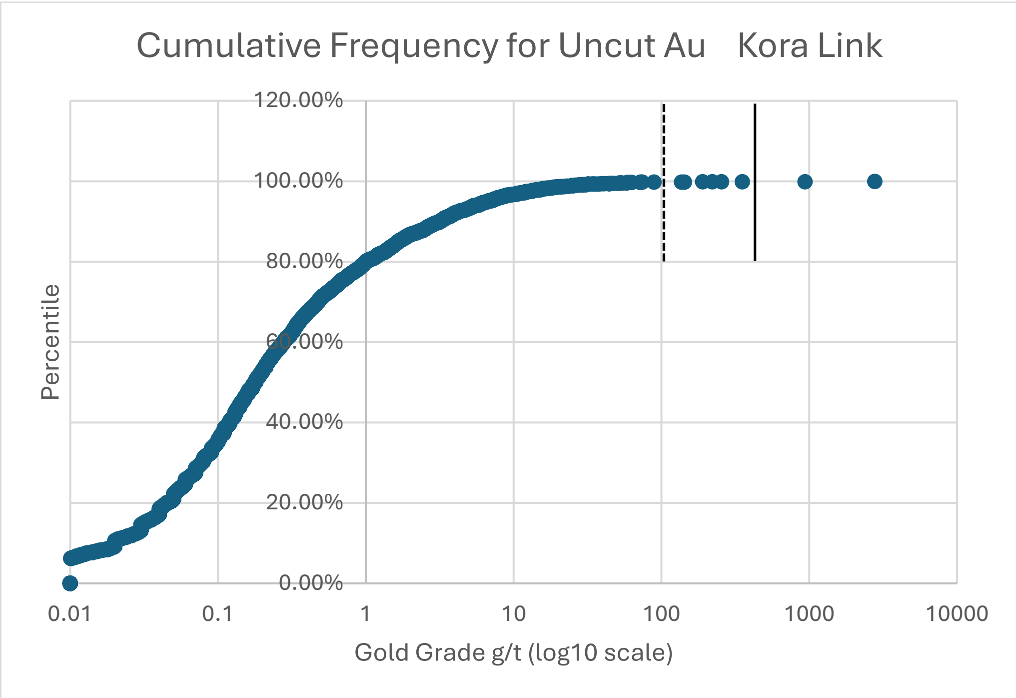


Figure 14-32 Kora Link Gold Composites Cumulative Frequency Curve (H&SC)

A histogram plot of the Kora Link cut gold data indicates a modest lognormal distribution with a positive skew but overall, the histogram resembles the expectation for a single population (Figure 14-33). The smooth cumulative frequency curve indicates a single population.

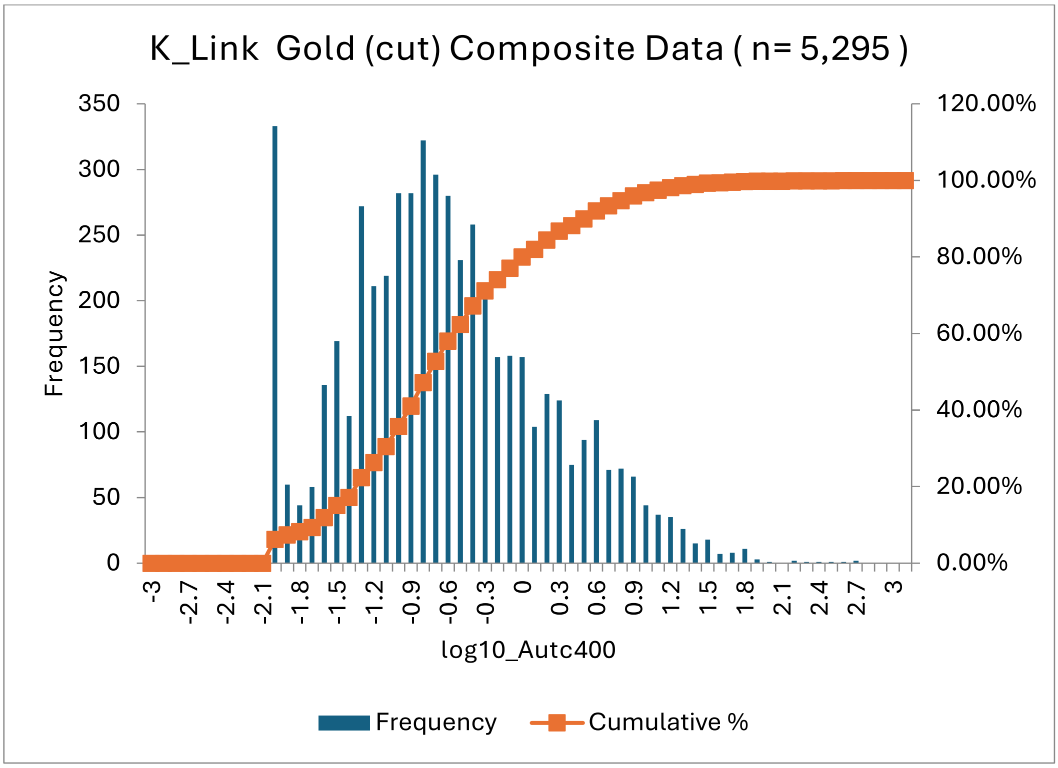


Figure 14-33 Histogram of Kora Link Cut Gold Composite Data (H&SC)

Table 14-12 shows the summary statistics for the Kora Link lode. Uncut gold data had a CV of 17.4 due to two extreme grades. Applying a 400 g/t top cut to the two extreme samples resulted in a 23% reduction in the composite mean and reduced the CV to 6.6, which is still considered quite high but because of the well-structured data and surrounding low grades the impact of the extreme values would be limited. The two samples were from drillholes and consisted of 2,777 g/t and 935 g/t.

Table 14-12 Summary Statistics for the Kora Link Lode

KLink	Gold	Gold_cut400	Gold_cut100	Copper	Silver	Silver_cut300
Mean	2.37	1.82	1.57	0.29	5.56	5.02
Median	0.18	0.18	0.18	0.09	2.00	2.00
Standard Deviation	41.25	11.94	6.24	0.71	34.67	13.57
Coeff of Variation	17.42	6.57	3.97	2.47	6.24	2.70
Minimum	0.01	0.01	0.01	0.0002	0.05	0.05
Maximum	2,777.07	400	100	17.4319	1,770	300
Count	5,295	5,295	5,295	5,295	5,295	5,295

A moderate CV was returned for copper which implies that no top cut is required but the relatively high CV for silver suggested the need for a top cut of 300 ppm.

It should be noted that the composite mean is considerably less that the 2021 model due to the removal this time of the mis-interpreted higher grade K1 face sampling data.

There is no significant correlation between gold and the other two elements as shown in Table 14-13. There is a weak correlation between copper and silver.

Table 14-13 Correlation Coefficients for the Kora Link Lode Composite Data

	Gold	Gold_cut400	Copper	Silver_cut300
Gold	1			
Gold_cut400	0.73	1		
Copper	0.15	0.32	1	
Silver_cut300	0.15	0.32	0.52	1

At Judd, missing data for the J1 lode composites, i.e. zeros and blanks, were noted for copper and silver (Table 14-14). The missing assays are due to the same reasons as for K1. The blanks and zeroes were replaced with low grade default values.

Table 14-14 Details of Insert Grades for the J1 Lode

		Sample Numbers	
Regressions	**Blanks**	**Zeroes**	
Copper	n/a	75	36
Silver	n/a	23	n/a
Defaults			
Copper	0.0001		111
Silver	0.01		23

Figure 14-34 shows the cumulative frequency curve for the uncut J1 lode gold composite data. The diagram shows three extreme samples which account for 13% of the total gold. These three samples comprise one face sample (1,327 g/t), one K92ML drilling sample (1,554 g/t) and a historic (Barrick -1,006 g/t) drilling sample. The three samples plus three other samples >400 g/t, indicate that a top cut to the data should be applied. It was agreed with K92ML that a top cut of 400 g/t was the most appropriate (black vertical line in the figure).

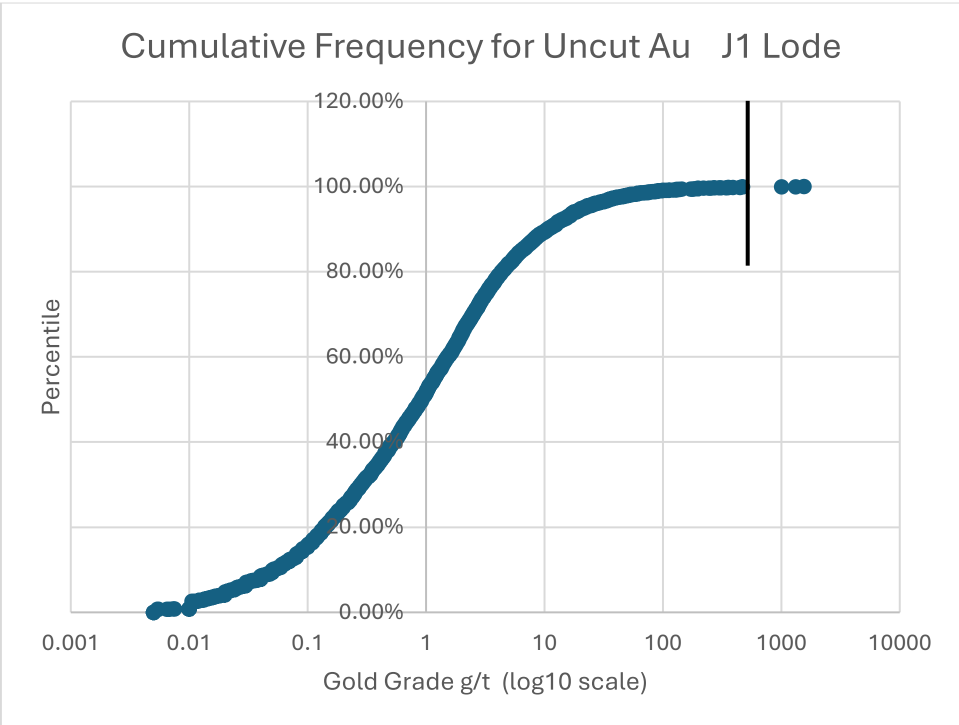


Figure 14-34 J1 Lode Gold Composites Cumulative Frequency Curve (H&SC)

A histogram plot of the J1 cut gold composite data indicates a modest lognormal distribution with a slight positive skew but overall, the histogram resembles the expectation for a single population (Figure 14-35). The smooth cumulative frequency curve indicates a single population.

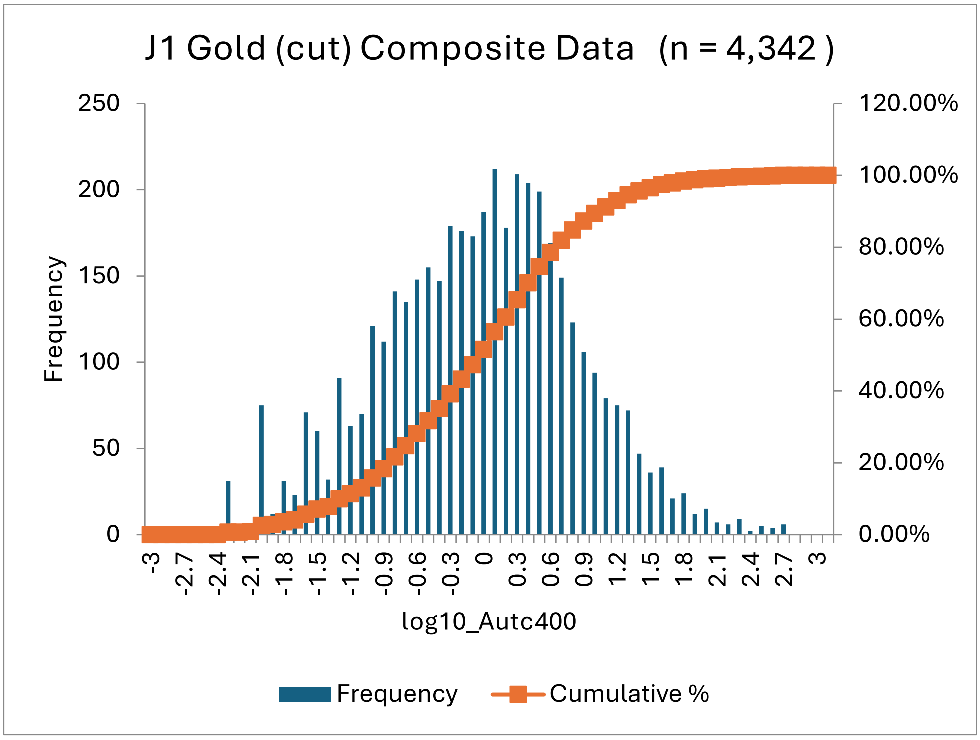


Figure 14-35 Histogram of J1 Gold Composite Data (H&SC)

Table 14-15 shows the summary statistics for the J1 lode. Application of the 400 g/t top cut resulted in a drop of 10% in the average gold grade for the composites. The uncut gold data had a relatively high CV of 6.3 due to the six extreme grades and whilst some of the grades are within the face sampling area, one sample sits in an area of limited drilling. With the limited amount of data in the peripheral areas this necessitates the application of a top cut. Applying the top cut saw a reduction in the CV to 4.2 which is still considered quite high but because of the good structure to the data a further reduction in the top cut was not considered necessary.

Table 14-15 Summary Statistics for the J1 Lode

J1	Gold	Gold_cut400	Gold_cut100	Copper	Silver	Silver_cut300
Mean	6.70	6.04	4.86	0.64	15.65	14.80
Median	0.92	0.92	0.92	0.19	4.48	4.48
Standard Dev	41.99	25.47	13.23	1.19	45.16	31.77
Coeff of Var	6.27	4.22	2.72	1.85	2.89	2.15
Minimum	0.005	0.005	0.005	0.0001	0.01	0.01
Maximum	1554.519	400	100	19.4186	1364.3551	300
Count	4,342	4,342	4,342	4,342	4,342	43,42

The 2023 mean grade for the cut gold (400 g/t) has dropped significantly from the 2021 figure and this is simply due to the additional drilling having intersected overall lower grade gold mineralization. Comparison of mean gold grades for the same area occupied by the drilling and face sampling indicates a similar gold grade for the uncut data of 8 g/t (drilling) and 8.45 g/t (face sampling). This justifies the combining of the face sample and drilling data in the composite file generation.

The moderate CV for copper suggests that a top cut is not needed, whereas the silver top cut of 300 ppm reduces the CV to a more acceptable level for grade interpolation.

There is a weak copper-silver correlation for the J1 Lode but unlike the other lodes there appears to be a weak correlation between gold and silver as shown in Table 14-16.

Table 14-16 Correlation Coefficients for the J1 Lode Composite Data

	Gold	Gold_cut400	Copper	Silver_cut300
Gold	1			
Gold_cut400	0.85	1		
Copper	0.11	0.18	1	
Silver_cut300	0.46	0.59	0.51	1

A histogram plot of the J2 cut gold composite data indicates a modest lognormal distribution with a slight positive skew but overall, the histogram resembles the expectation for a single population (Figure 14-36). The relatively smooth cumulative frequency curve indicates a single population.

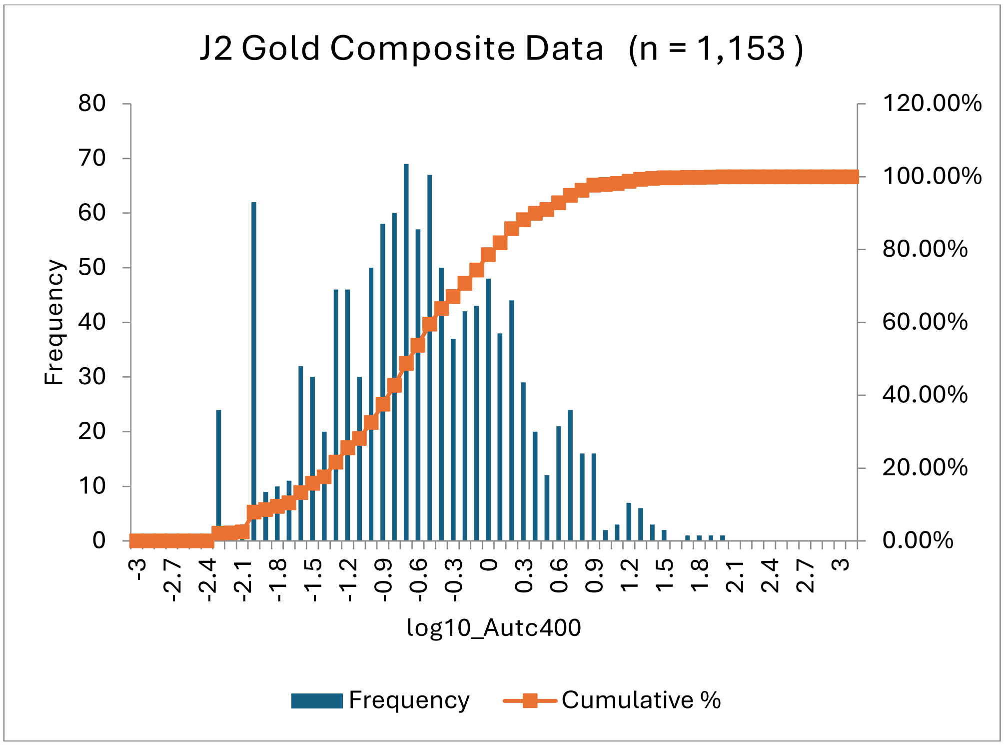


Figure 14-36 Histogram of J2 Gold Composite Data (H&SC)

Table 14-17 shows the summary statistics for the J2 lode. Despite the modestly high CV for gold and the overall much lower grades H&SC considered no top cut was necessary.

Table 14-17 Summary Statistics for the J2 Lode

J2	Gold	Copper	Silver	Silver_cut300
Mean	1.28	0.35	8.03	7.80
Median	0.22	0.07	2.71	2.71
Standard Deviation	4.90	0.84	22.05	17.00
Coeff of Variation	3.84	2.39	2.75	2.18
Minimum	0.005	0.0003	0.01	0.01
Maximum	99.07	9.98	567	300
Count	1,153	1,153	1,153	1,153

A histogram plot of the J3 gold composite data indicates a weak lognormal distribution with a slight positive skew but overall, the histogram resembles the expectation for a single population (Figure 14-37). The relatively smooth cumulative frequency curve indicates a single population.

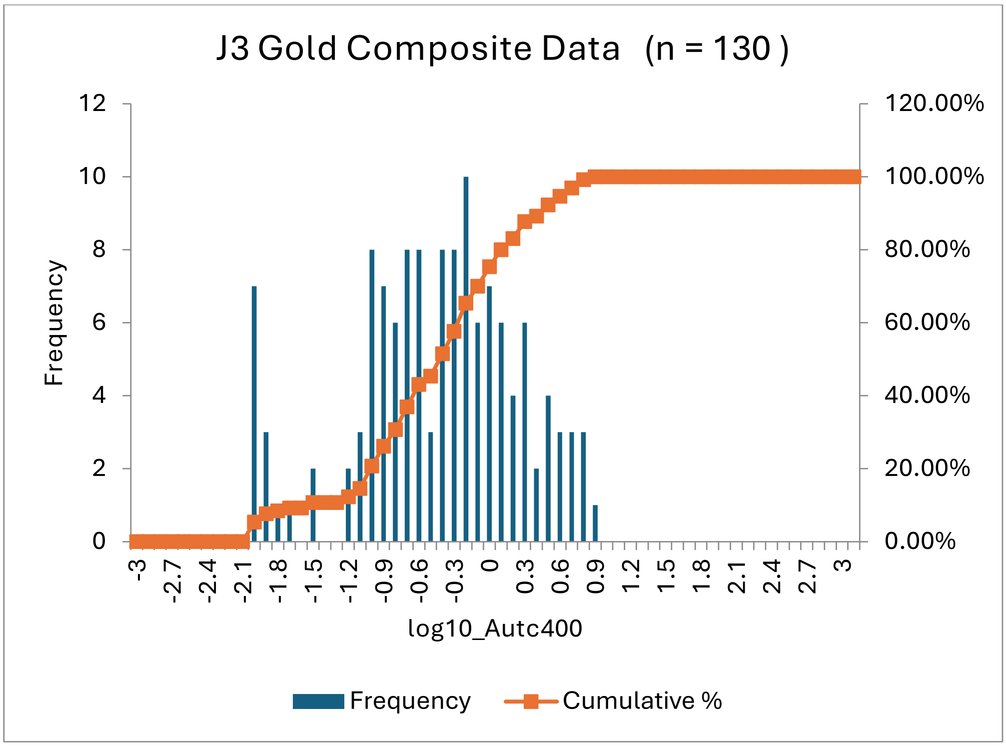


Figure 14-37 Histogram of J3 Gold Composite Data (H&SC)

Table 14-18 shows the summary statistics for the J3 lode. No top cut for gold was considered necessary on account of the low CV and the well-structured nature to the data.

Table 14-18 Summary Statistics for the J3 Lode

J2	Gold	Copper	Silver	Silver_cut300
Mean	0.90	0.57	14.91	14.86
Median	0.38	0.11	5.70	5.70
Standard Deviation	1.36	1.60	33.88	33.41
Coeff of Variation	1.51	2.78	2.27	2.25
Minimum	0.01	0.0057	1	1
Maximum	7.2277	14.47	307	300
Count	130	130	130	130

A histogram plot of the J1W gold composite data indicates a weak lognormal distribution with a moderately strong positive skew but overall, the histogram resembles the expectation for a single population (Figure 14-38). The cumulative frequency curve suggests possibly more than one population, but the small amount of data makes this difficult to tell.

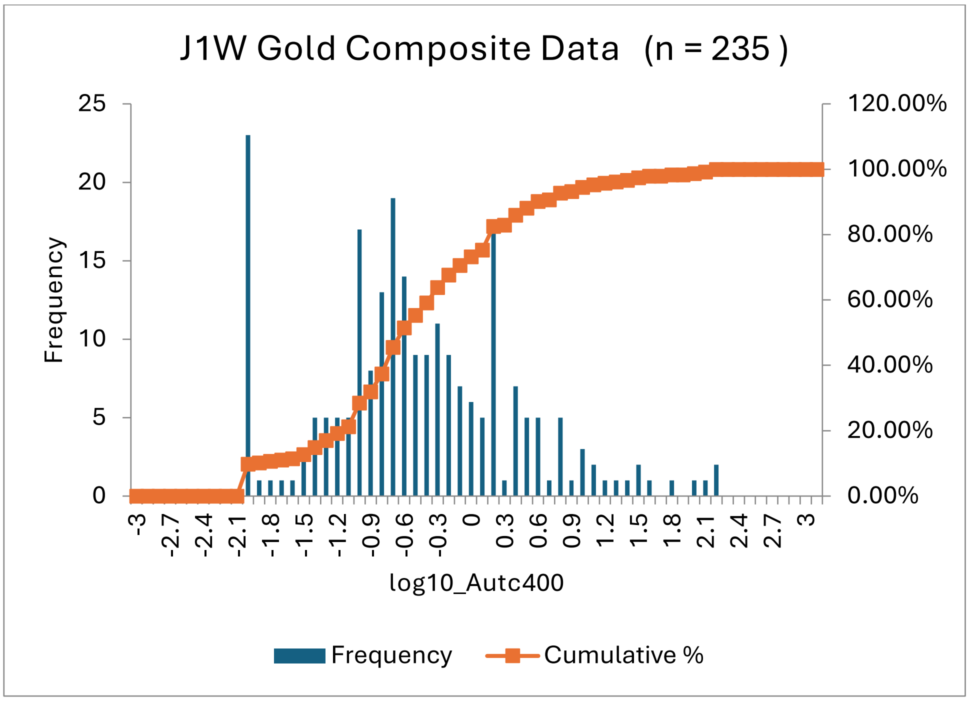


Figure 14-38 Histogram of J1W Gold Composite Data (H&SC)

Table 14-19 Summary Statistics for the J1W Lode

J2	Gold	Copper	Silver	Silver_cut300
Mean	3.84	0.51	11.57	11.57
Median	0.24	0.02	2.77	2.77
Standard Deviation	17.24	1.15	21.25	21.25
Coeff of Variation	4.49	2.26	1.84	1.84
Minimum	0.01	0.0004	0.05	0.05
Maximum	151.2	8.1313	174.1491	174.1491
Count	235	235	235	235

14.4 Variography

Variography was undertaken on the composite data of the individual lodes to ascertain spatial continuity of metal grades. The general comment is that the variography was weak to moderate, although K1 produced much better results than the other lodes. However, it should be noted that outside the drive development and stoping areas the drillhole spacing is relatively large and, in combination with the narrow lode structures and undulations in their dip and strike, good variography is difficult to achieve and is subject to compromises. The main implication from the variography is that more infill drilling is required in areas peripheral to the stopes and development.

Previously variography for Kora comprised a windowed out area of detailed drilling with no lode subdivisions that was used to develop 3D variogram models. With the increased amount of data this time it was possible to complete the variogram modelling using the most detailed subset of composite data (domain 1 in each case) for each lode (Figure 14-39 for K1 and Figure 14-40 for K2).

A similar composite data extraction process was applied to the Kora Link and J1 lodes and to the generation of variogram models (Figure 14-41). The J1 models were then used for the other three Judd lodes.

To further assist the variography and to smooth out outliers in the data, the modelling was undertaken on the 100 g/t top cut gold data. This is in recognition of a fundamental maxim for resource estimation in that the "higher the grade the shorter the continuity".

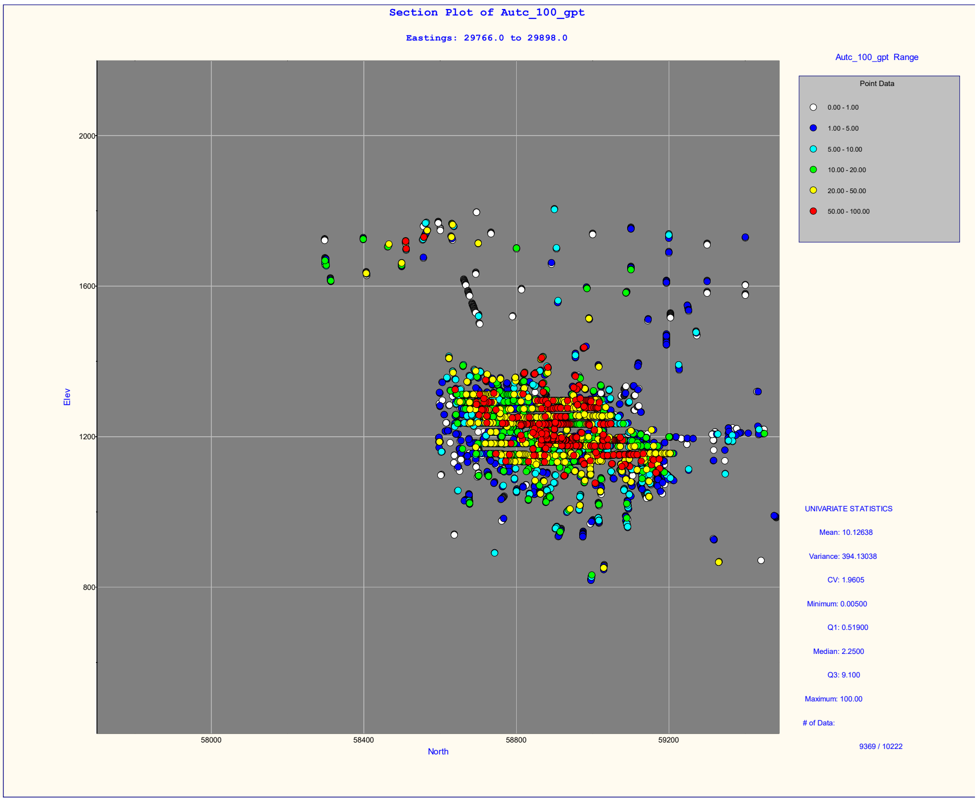


Figure 14-39 K1 Gold Composite Data Subset used for Variography Long Section View (H&SC)

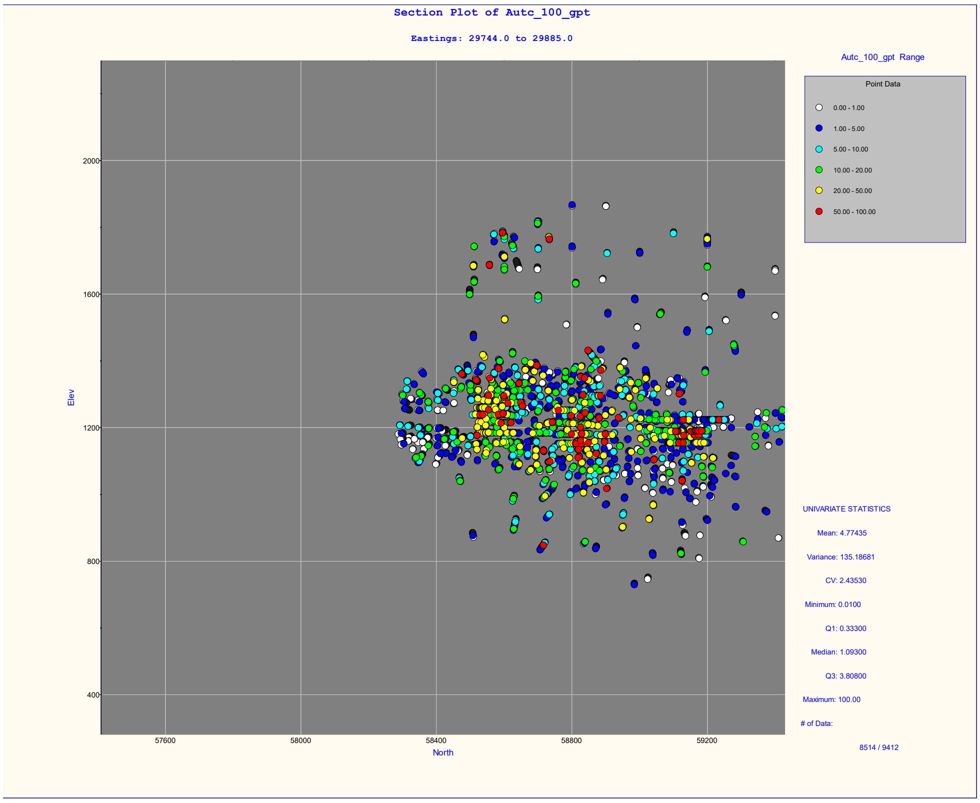


Figure 14-40 K2 Composite Data Subset used for Variography Long Section View (H&SC)

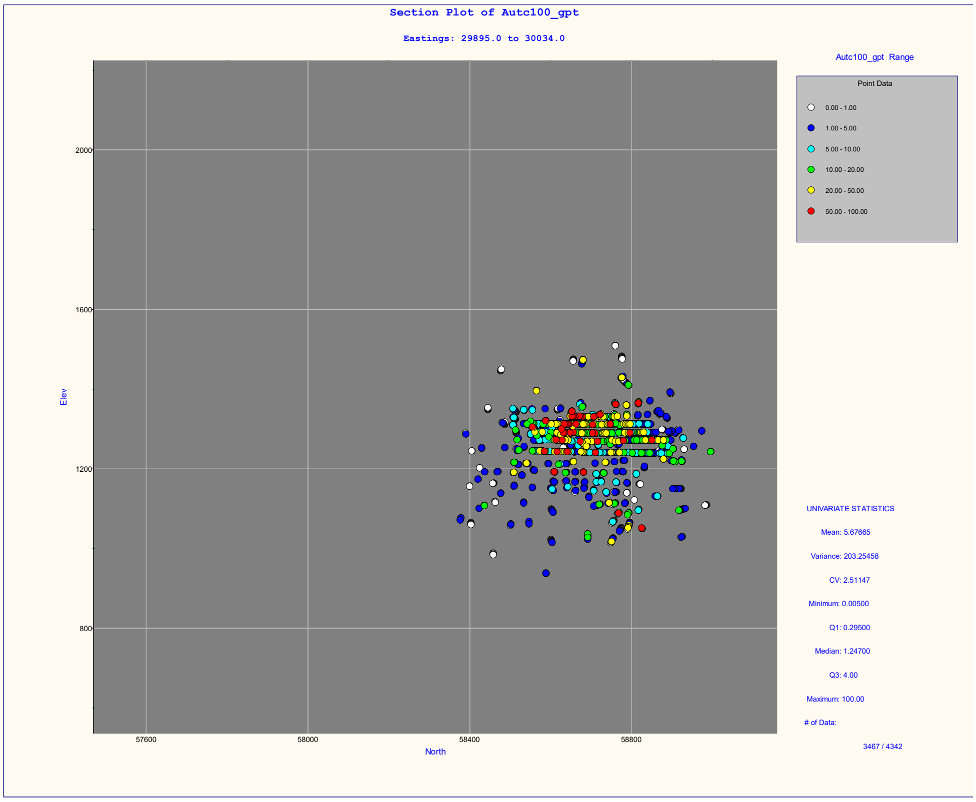


Figure 14-41 J1 Composite Data Subset used for Variography   Long Section View   (H&SC)

The asymmetry of the sampling will also impact on the variography, with 3 m spaced face sample lines along strike compared to 10-20 m spaced drive development for the down dip direction.

Figure 14-42 shows the strike and dip variogram maps for the K1 lode detailed drilling area. They are consistent with the geological understanding of the lode.

The strike image indicates a reasonable level of grade continuity over 20 m with a slightly better down dip continuity of 20-30 m (red contour in image). This is in contrast with the K2 images (Figure 14-43) which only indicate a strike continuity of 10 to 20 m with a slight smaller down dip continuity <10 m with but a longer lower grade range. The Kora Link lode has a similar pattern to the K2 lode.

K1 Lode

Strike **Dip**

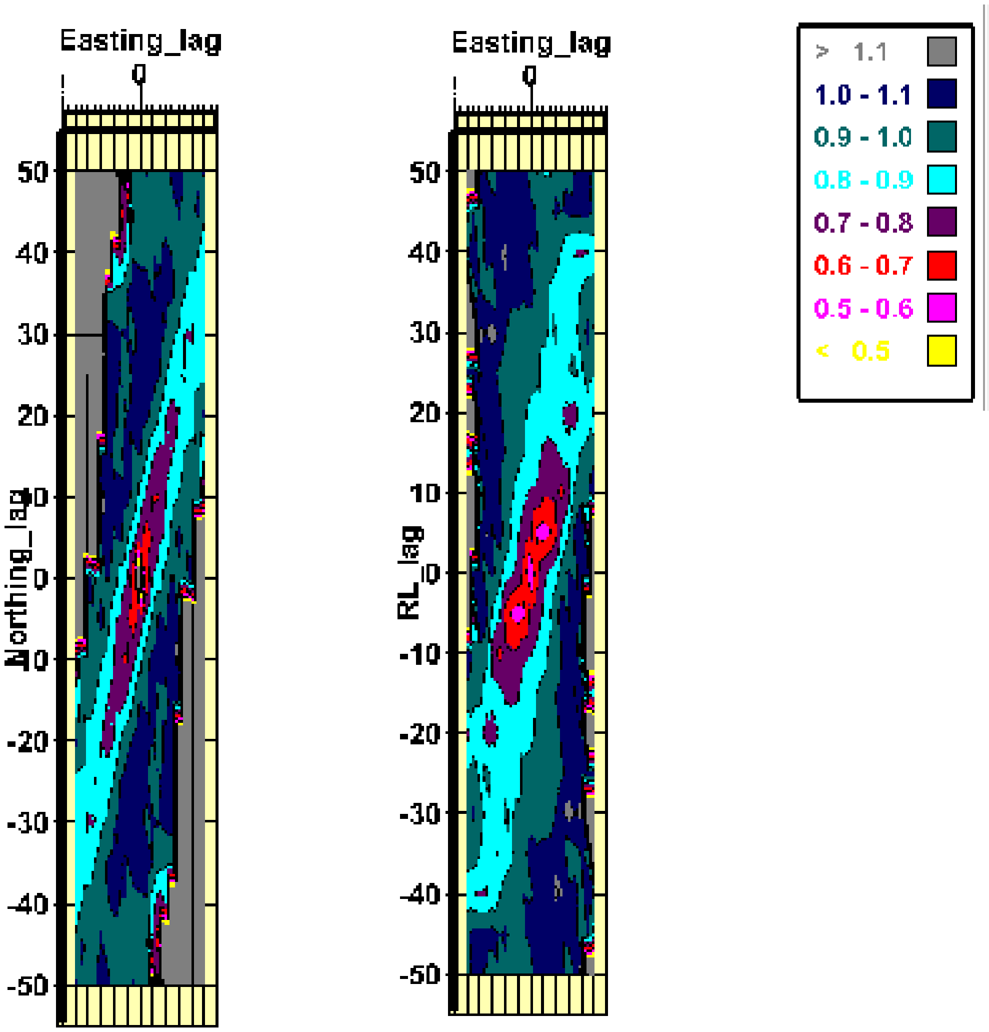


Figure 14-42 K1 Variogram Maps for Gold (H&SC)

K2 Lode

Strike **Dip**

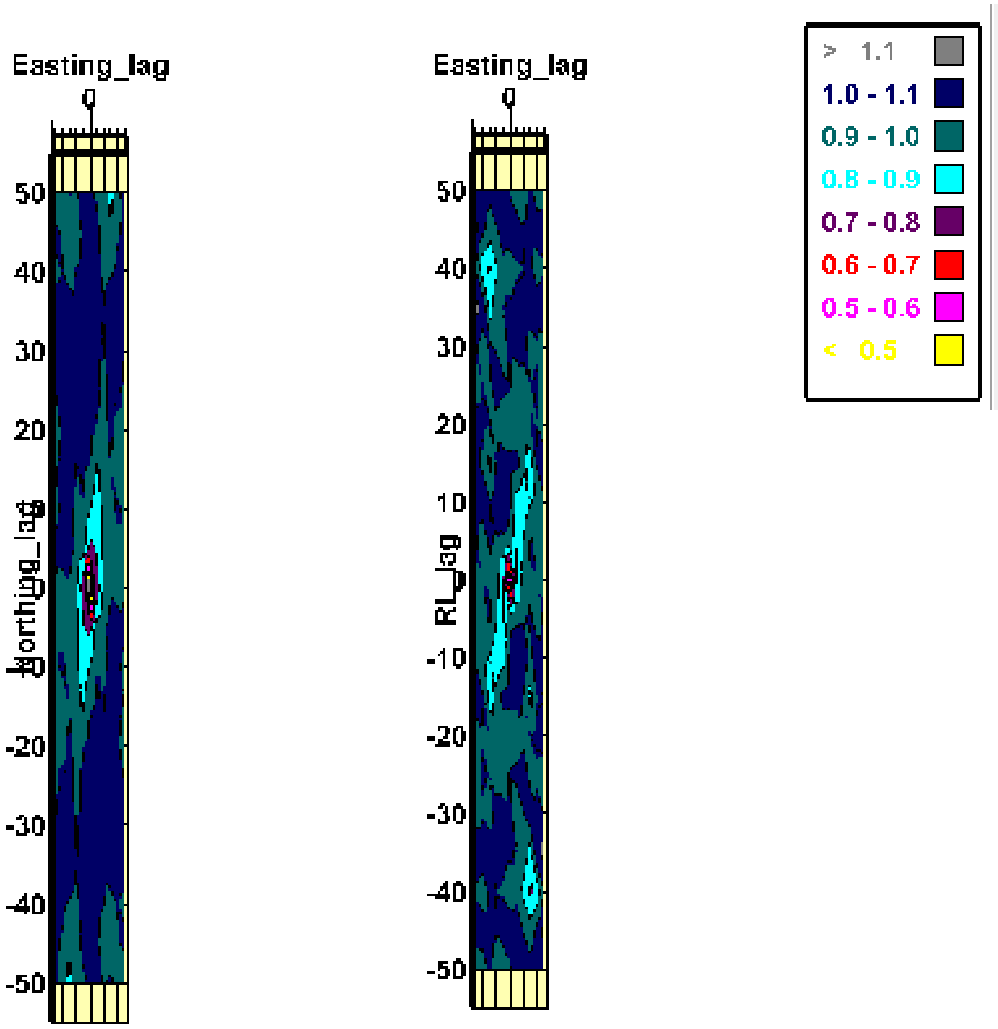


Figure 14-43 K2 Variogram Maps for Gold (H&SC)

The dip and strike images for the J1 lode are similar to the K2 lode indicating a strike continuity of 15-25 m and down dip continuity of 10-20 m (Figure 14-44).

These shorter distances for K2 and J1, compared to K1, are confirmed by visually inspecting the level plans for the face sampling for the two lodes. Visually, grade continuity is best measured by linking up higher grade zones using a nominal 5 g/t threshold along specific structural positions and not crossing structural lines e.g. adhering to the hanging wall contact, footwall contact or in a central zone.

Variogram maps were constructed in a similar way for copper and silver (top cut to 300 ppm). They tended to match the gold outcomes despite the lack of correlation between the elements.

J1 Lode

Strike **Dip**

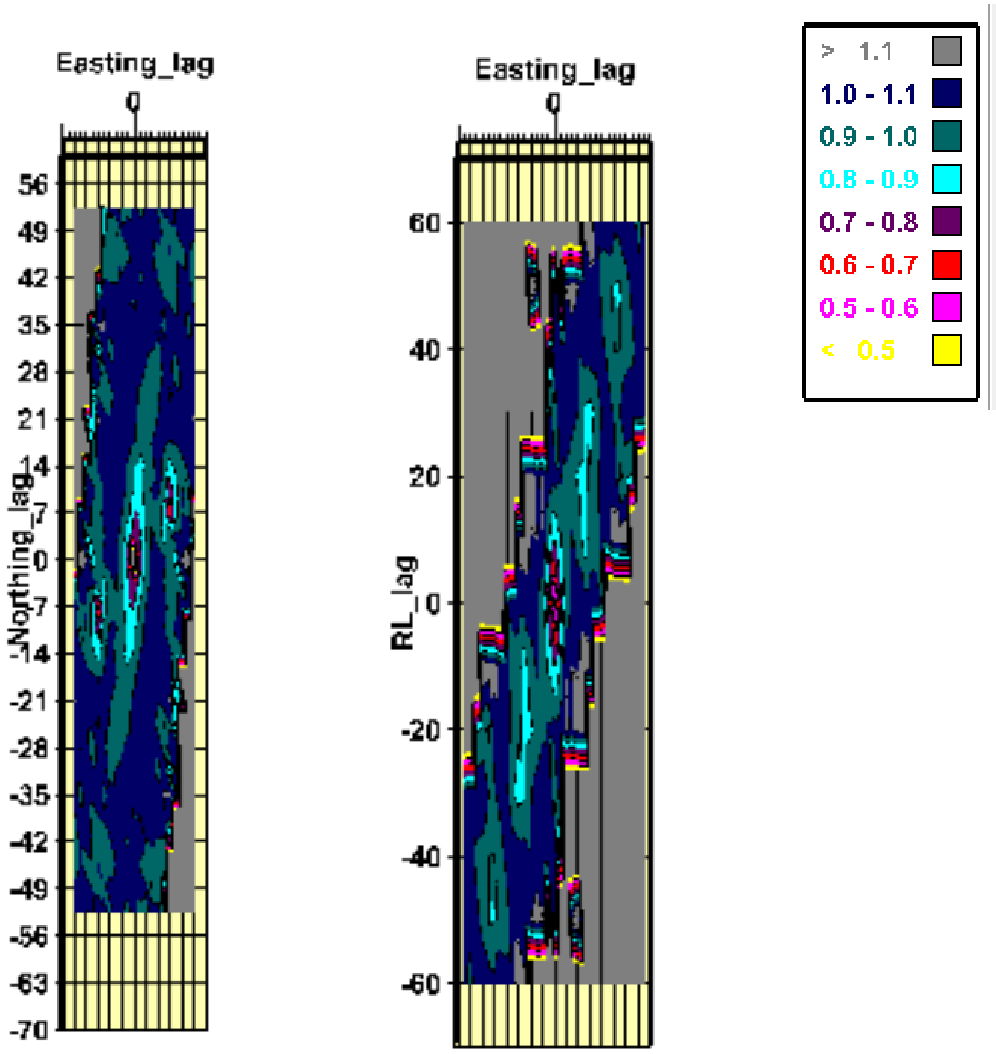


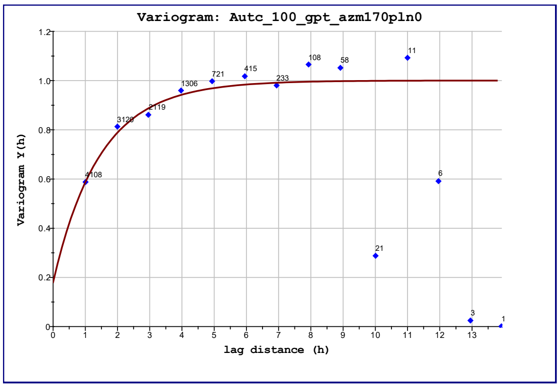
Figure 14-44 J1 Variogram Maps for Gold (H&SC)

Figure 14-45 shows the orthogonal variograms and the resulting 3D variogram model for the K1 gold composites. In an attempt to improve the variography an arbitrary top cut of 100 g/t was used. The variograms indicate a relative lack of grade continuity across strike (the down hole variogram), generally of the order of 2 or 3 m, as would be expected from the geology, whereas the dip and strike continuity of the composite gold grade is of the order of 20 to 40 m which is consistent with the variogram maps and the visual data review of the grade continuity seen with the face sampling.

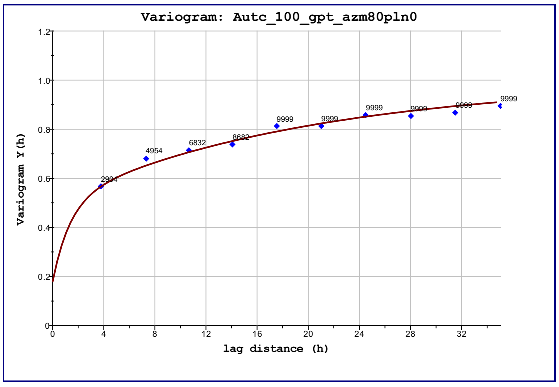
Also, in Figure 14-45 is a 3D representation of the variogram model for this subset of composite data (domain 1). It indicates greater grade continuity in the vertical direction which is plausible when looking at the composite grade distribution in long section.

K1 Lode

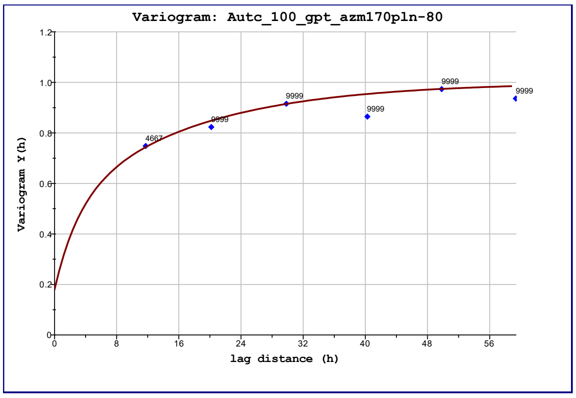
Downhole



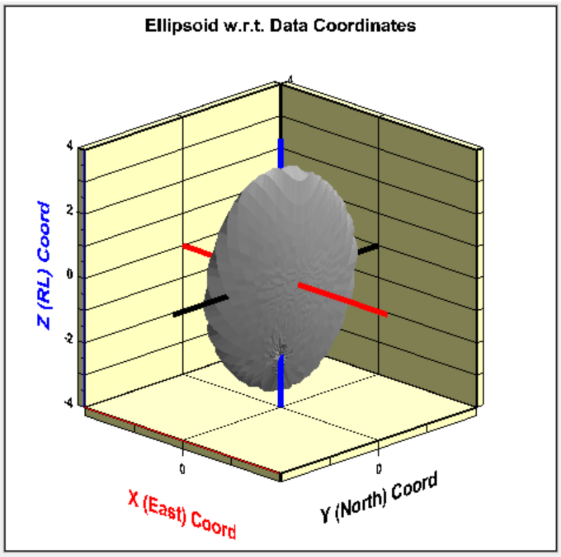
Strike



Dip



3D Variogram Model



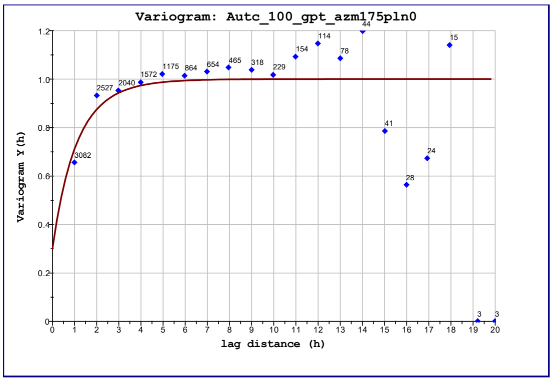
(trigonometrical convention for rotations)

Figure 14-45 K1 Variograms & Variogram Model for Gold (H&SC)

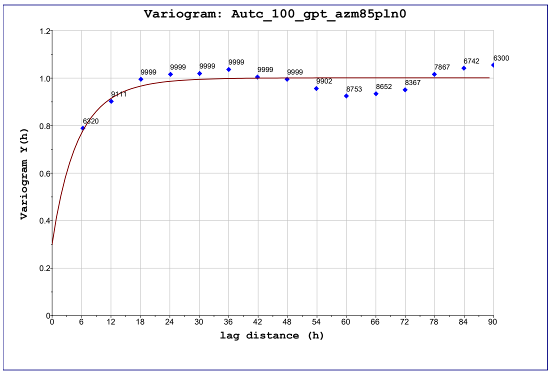
Figure 14-46 shows the orthogonal variograms and the resulting 3D variogram model for the K2 gold composites. In an attempt to improve the variography an arbitrary top cut of 100 g/t was used on the composite data. The variograms indicate a relative lack of grade continuity across strike (the down hole variogram), generally of the order of 1 or 2 m. whereas the dip and strike continuity of the composite gold grade is of the order of 10 to 20 m which is consistent with the visual data review of the grade continuity for the face sampling.

K2 Lode

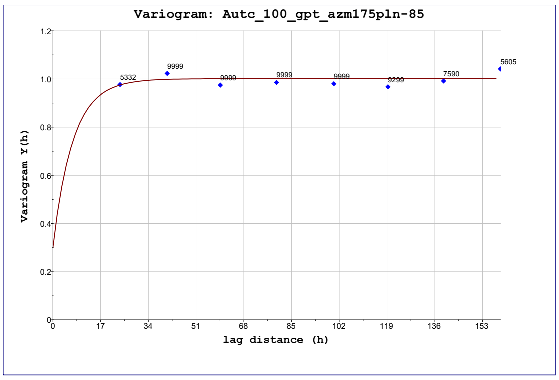
Downhole



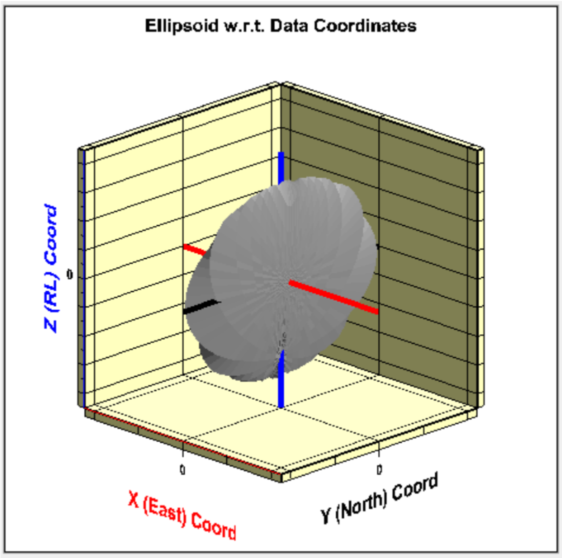
Strike



Dip



3D Variogram Model



(trigonometrical convention for rotations)

Figure 14-46 K2 Variograms & Variogram Model for Gold (H&SC)

Figure 14-47 shows the 3D variogram models for copper and silver for K2 which appear very similar and is consistent with the relative weak correlation noted for copper and silver at K2.

K2 Lode

Copper **Silver**

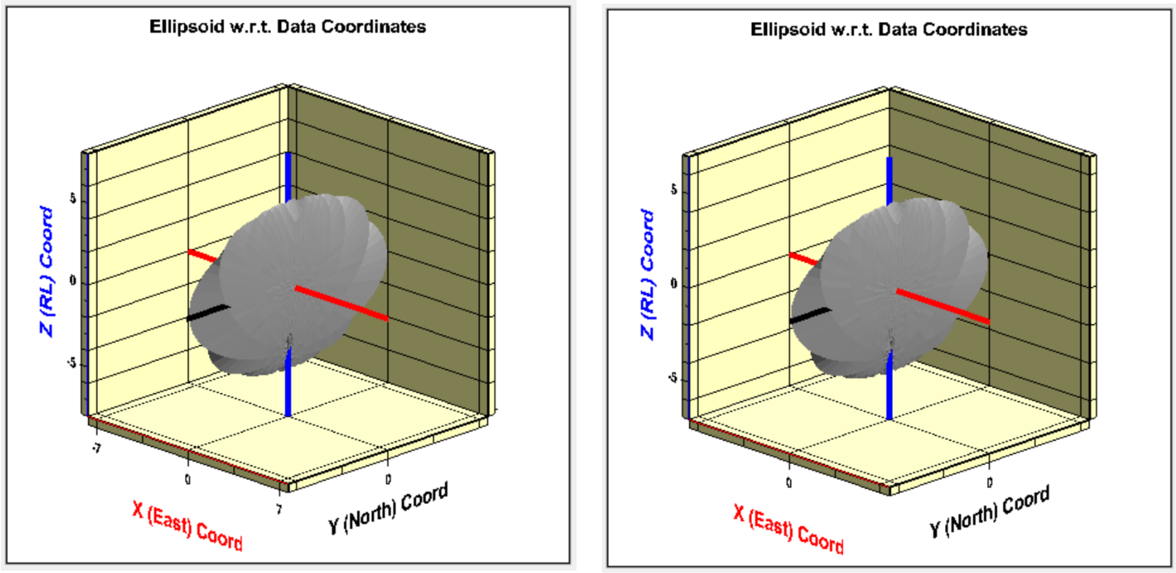
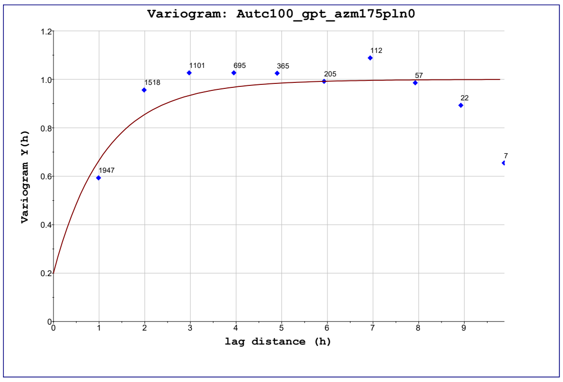


Figure 14-47 K2 Copper and Silver 3D Variogram Models (H&SC)

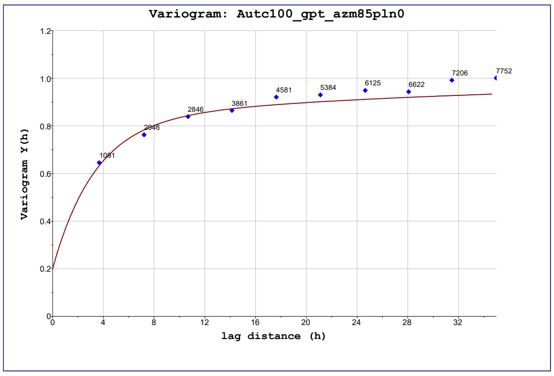
Figure 14-48 shows the orthogonal variograms and the resulting 3D variogram model for the gold composites for the J1 lode. To improve the variography an arbitrary top cut of 100 g/t was used on the composite data. The variograms indicate a relative lack of grade continuity across strike (the down hole variogram), generally of the order of 1 or 2 m. whereas the dip and strike continuity of the composite gold grade is of the order of 10 to 20 m which is consistent with the variogram maps and the visual data review of the grade continuity seen with the face sampling.

J1 Lode

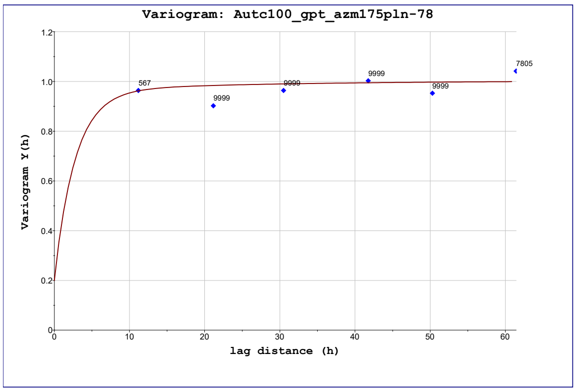
Downhole



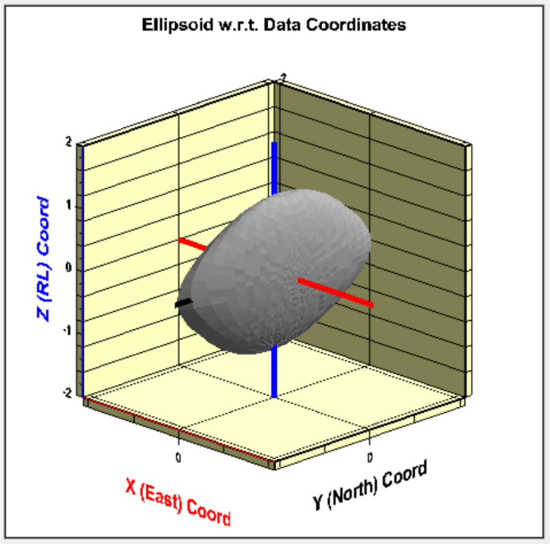
Strike



Dip



3D Variogram Model



(trigonometrical convention for rotations)

Figure 14-48 J1 Variograms & Variogram Model for Gold (H&SC)

Figure 14-49 shows the 3D variogram models for copper and silver for the J1 Lode. These are slightly different to each other noting the similarities observed with K2 and maybe due to the fact there is a slight gold/silver correlation at J1 not observable at Kora.

J1 Lode

Copper **Silver**

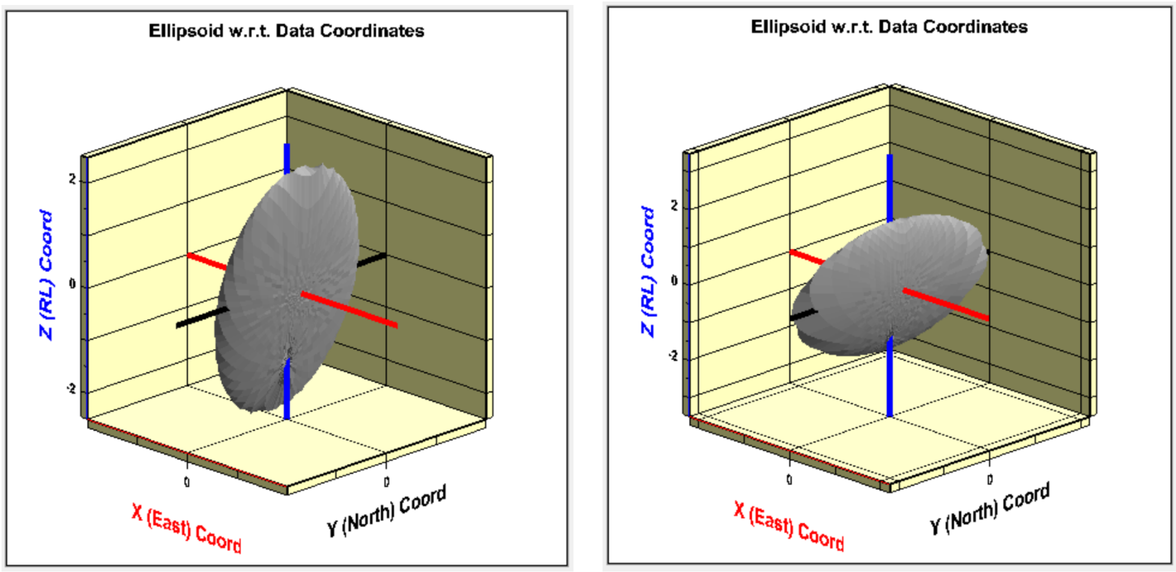


Figure 14-49 Judd Copper and Silver 3D Variogram Models (H&SC)

14.5 Block Model Details

Two separate 1 m (X) by 5 m (Y) by 5 m (Z) N-S oriented block models were created for Kora and Judd. The block model coordinates are the same as for previous models. The block size is mainly in deference to the close spaced drilling for the K1, K2 and J1 lodes and the face sampling. Details of the Surpac block models are included in Table 14-20.

Table 14-20 Block Models Details

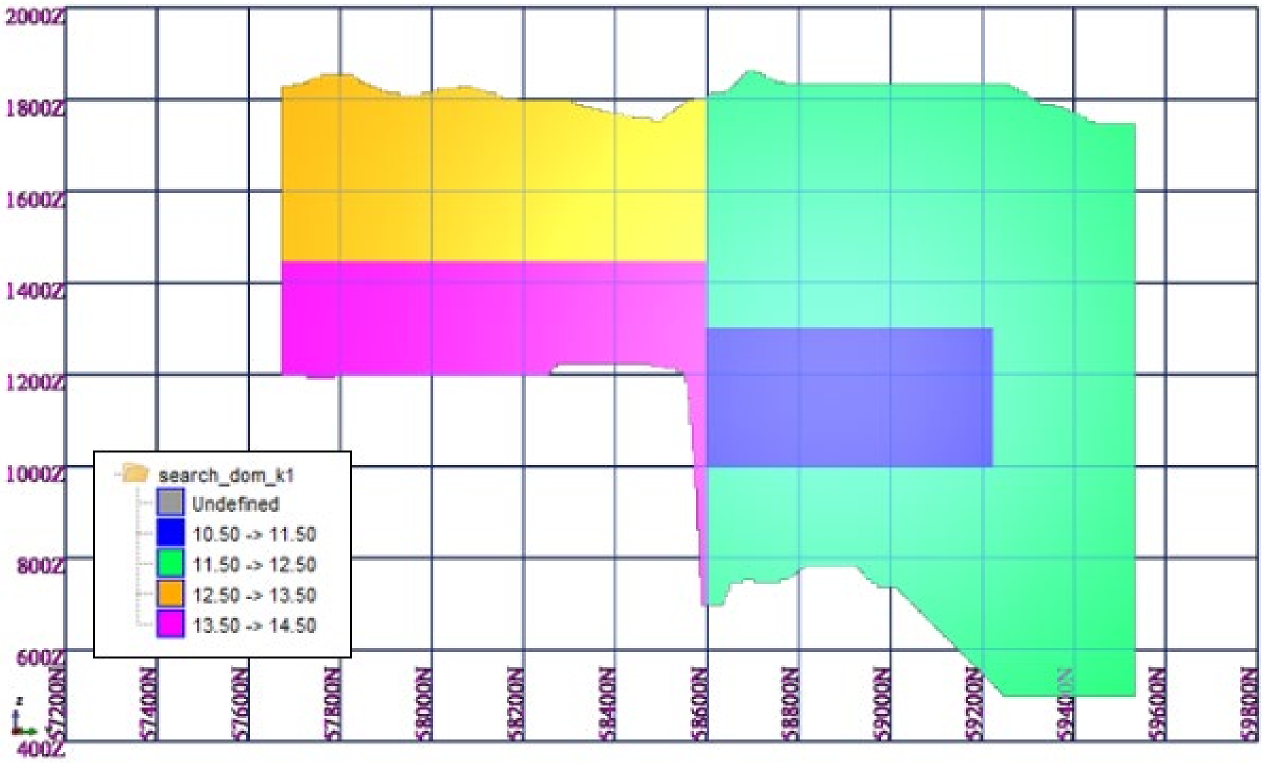
Block Model Summary			
Block model: Kora_ok_working_201123.mdl			
Type	**X**	**Y**	**Z**
Minimum Coordinates	29,715.5	57,322.5	472.5
Maximum Coordinates	29,920.5	59,592.5	1,972.5
User Block Size	1	5	5
Min. Block Size	1	5	5
Rotation	0	0	0
Discretisation	2	4	4
Block model Judd_ok_working_271123.mdl			
Type	**X**	**Y**	**Z**
Minimum Coordinates	29,800	57,300	550
Maximum Coordinates	30,100	59,400	2,000
User Block Size	1	5	5
Min. Block Size	1	5	5
Rotation	0	0	0
Discretisation	2	4	4

Grade interpolation was undertaken using the OK option from H&SC's in-house GS3 software. The resulting models for different search orientations were then loaded into a Surpac block model for post-modelling processing and resource reporting. The interpolation strategy consisted of three search passes each with increasing search radii and/or decreasing number of octants and/or decreasing minimum number of data. Details of the search pass parameters are listed in Table 14-21.

Table 14-21 Search Ellipse Parameters

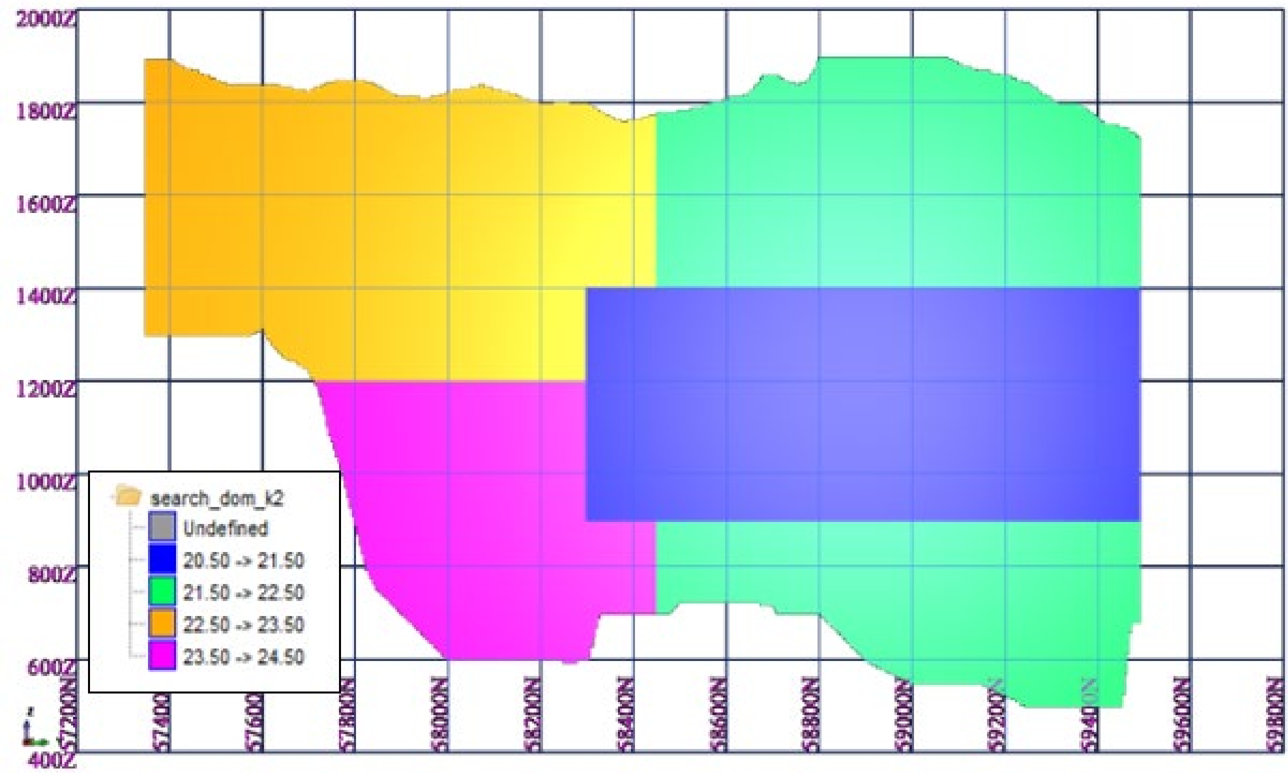
Pass No	X radius (m)	Y radius (m)	Z radius (m)	Min Data	Min Octants	Max Data
Kora						
1	2	25	25	12	4	32
2	4	50	50	12	4	32
3	12	125	125	6	2	32
Judd						
1	2	25	25	12	4	32
2	4	50	50	12	4	32
3	6	125	125	6	2	32

There are subtle variations in the dip and strike for both the Kora and Judd lodes that necessitated the use of search sub-domains with the block grade interpolation. These sub-domains are a product of the changes in the axes' orientations of the mineral wireframes. Four search sub-domains, labelled 11 to 14, were used for the K1 grade interpolation (Figure 14-50) and four sub-domains for K2 grade interpolation, labelled 21 to 24 (Figure 14-51). The Kora Link grade interpolation was completed using three search sub-domains, labelled 31 to 33 (Figure 14-52), whilst the J1 lode (and J2) used two search sub-domains labelled 41 and 42 (Figure 14-53). Generally speaking, the blue zones represent an inflexion in the relevant structure with a slightly shallower dip angle - the dilatant part of the overall sigmoidal structure.



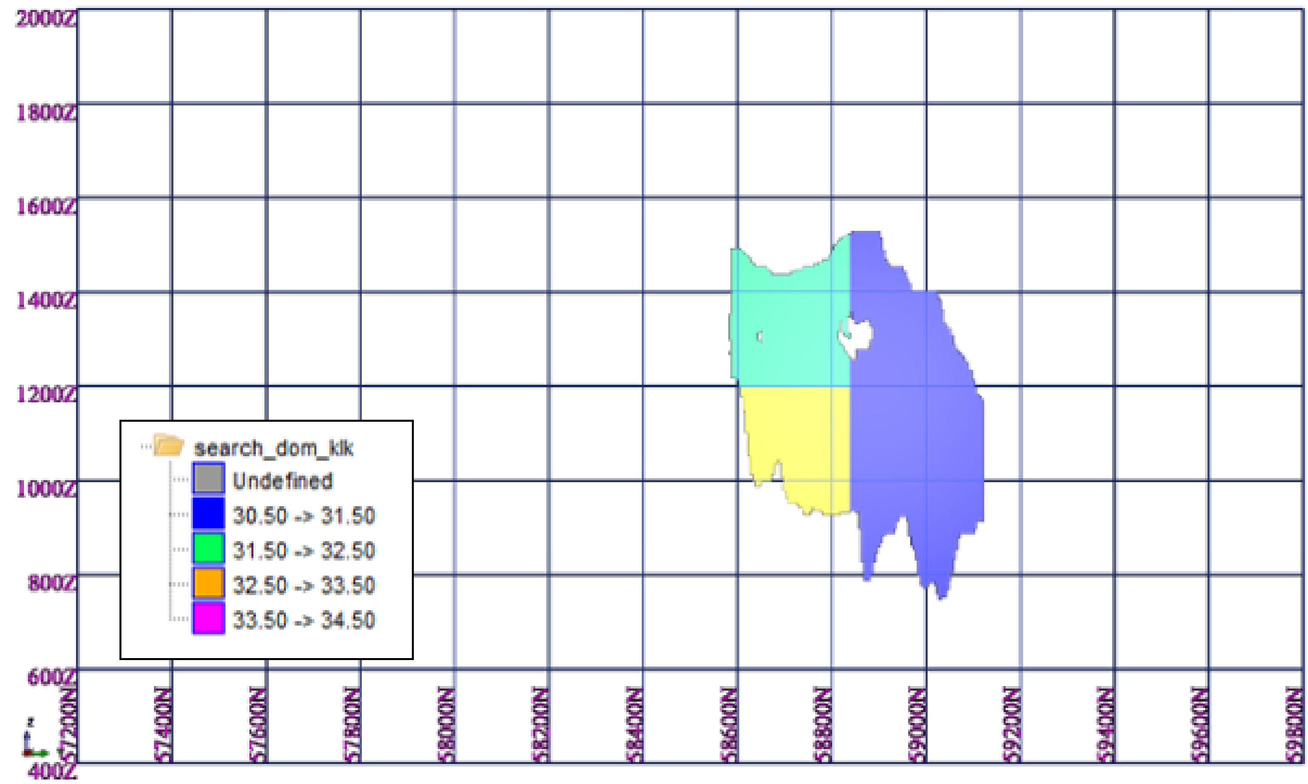
(view looking west)

Figure 14-50 Long Section for Search Sub-domains for K1 Lode (H&SC)



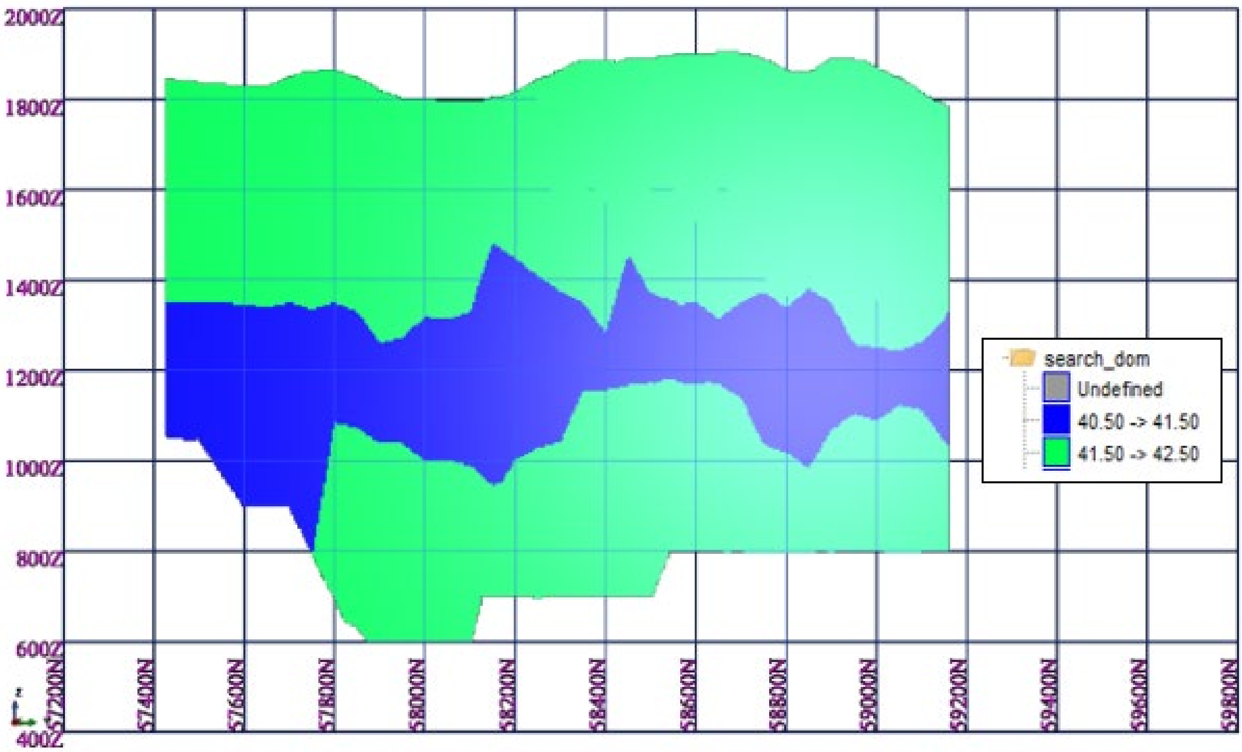
(view looking west)

Figure 14-51 Long Section for Search Sub-domains for Kora Link Lode (H&SC)



(view looking west)

Figure 14-52 Long Section for Search Sub-domains for J1 Lode (H&SC)



(view looking west)

Figure 14-53 Long Section for Search Sub-domains for J1 Lode (H&SC)

Details of the search ellipse rotations are in Table 14-22.

Table 14-22 Search Ellipse Rotations for Search Sub-domains

Pass No	Domain 1	Domain 2	Domain 3	Domain 4
K1				
X	0	0	0	0
Y	10	5	-10	-4
Z	-8	-1	3	0
K2				
X	0	0	0	0
Y	5	2	10	0
Z	-5	-5	3	3
Kora Link				
X	0	0	0	
Y	0	8	0	
Z	-12	-2	-8	
J1				
X	0	0		
Y	12	5		
Z	-5	0		

(angles in trigonometric convention)

Previously, density for the Mineral Resources comprised average values for each lode and an average value for the waste material. In a global sense this will probably be quite accurate but in a local sense it is perhaps unrealistic that every block will have the same density considering the nature of the mineral lodes with variable vugs, quartz and sulphide contents. H&SC felt that there was enough density sample data to undertake Ordinary Kriging for a density block grade interpolation. A common industry practice is to complete density modelling using the Inverse Distance Squared technique. H&SC prefers to use Ordinary Kriging as it is a more sophisticated method and allows for local interaction of sample data to more accurately estimate individual density block grades.

Density grade interpolation used the same general strategy as for the metal grade interpolation via the OK option from H&SC's in-house GS3 software. The resulting models for both Kora and Judd were then loaded into the Surpac block model for post-modelling processing and resource reporting.

Two search domains (as opposed to four domains for the metal grade interpolation) were used for density for Kora with details of the domains shown in Table 14-23. For Judd the same search domains as used for the metal grade interpolation were applied for the density modelling.

Table 14-23 Search Domains for Kora Density

Lode	Divide	Domain 1	Domain 2
K1	Northing	>58,850	<58,850
K2	Elevation	>1,200	<1,200
KLink	Northing	>58,850	<58,850

The interpolation strategy consisted of three search passes each with increasing search radii and/or decreasing number of octants and/or decreasing minimum number of data. Details of the search pass parameters are listed in Table 14-24 with the search domain rotations listed in Table 14-25.

Table 14-24 Search Ellipse Parameters for Density

Pass No	X radius (m)	Y radius (m)	Z radius (m)	Min Data	Min Octants	Max Data
Kora						
1	2	25	25	12	4	32
2	4	50	50	12	4	32
3	16	125	125	6	2	32
Judd						
1	2	25	25	12	4	32
2	4	50	50	12	4	32
3	16	125	125	6	2	32

J3 and J1W had very limited data and it was decided to use average default values for their density. These lodes make up a relatively small component of the overall Mineral Resources and have the lowest level of confidence.

Table 14-25 Search Ellipse Rotations for Density

Direction	Domain 1	Domain 2
K1		
X	0	0
Y	0	0
Z	-8	0

Direction	Domain 1	Domain 2
K2		
X	0	0
Y	6	0
Z	-7	-2
Kora Link		
X	0	0
Y	0	0
Z	-12	-3
J1 & J2		
X	0	0
Y	12	5
Z	-5	0

(angles in trigonometric convention)

The lack of density sample data in specific areas of the lodes meant that there were blocks without a density value. As a result, default density values based on the surrounding approximate average value were inserted into the gaps. The default density values and the areas affected are listed in Table 14-26.

Table 14-26 Summary of Default Density Values

Lode	Divide	Domain 1	Value t/m^3	Domain 2	Value t/m^3
K1	Elevation	<1,200mRL	2.77	>1,200mRL	2.61
K2	Elevation	<1,200mRL	2.73	>1,200mRL	2.59
KLink	Elevation	<925mRL	2.60	>1,450mRL	2.49
J1	3DM Surfaces	Sdom 41	2.62	Sdom 42	2.62
J2	3DM Surfaces	Sdom 41	2.54	Sdom 42	2.54
J3	3DM Surfaces	Sdom 41	2.68	Sdom 42	2.68
J1W	3DM Surfaces	Sdom 41	2.62	Sdom 42	2.62

Upon the late stage realisation in the resource estimation by K92ML that a substantial amount of the density data was suspect, a reallocation of the density grade was made whereby an average default value of 2.765 t/m^3 was applied to the upper southern area of both K1 and K2. This was done by using a defined shape (k1_reallocate_density_shape_1023.dtm) for blocks inside the lode wireframe with a density <2.5 t/m^3.

H&SC recommend a complete review of the density data including a check sample programme.

On completion of the gold grade interpolation using H&SC's in-house GS3 software, the modelled data was loaded into a Surpac block model. Each lode was modelled separately with several search domains and then loaded sequentially into the block model. Data for each lode was then combined into a single gold grade attribute for the deposit, with the same process being applied to copper, silver and density. The combination process relied on the lode attribute centroid as the control. As several different top cuts were applied to the modelling of the gold data, there were options to select the preferred modelled gold grade into the 'final' gold grade attribute. In this instance the uncut K1 data was used along with the 1,000 g/t cut K2 data and the uncut Kora Link data. The main reason for using the cut K2 gold data is that the majority of the extreme grades are in areas of limited drilling, away from the face sampling, and there is a need to try and limit the impact of those high grades, albeit that a substantial amount of the blocks will be classed as Inferred. With K1 the extreme grades are generally within the face sampling zone and their impact has been sufficiently controlled by the surrounding samples. There is a case for using the 400 g/t cut data for the Kora Link lode rather than the uncut data, due to two really extreme values, but the impact of these extreme values has been limited by the surrounding samples with the amount of material affected being relatively small in the overall scheme of things.

For the Judd lodes the 400 ppm top cut gold value was used in all instances.

The combined metal attributes are used to report the Mineral Resources.

14.6 Estimation Results

The new global estimation results for Kora with mining depletion removed (up to the end of September 2023) are reported for a range of different gold cut-off grades as shown in Table 14-27. Estimation results are reported for all block centroids inside the relevant mineral lode wireframe (in/out basis). The results are split by lode and demonstrate the impact of applying top cuts to the composite data. Classification of the estimates is included later in this chapter.

A gold equivalent average grade is provided based on using the preferred cut or uncut gold grade e.g. for K1 and Kora Link it is the uncut gold data, for K2 it is the 1,000 g/t cut data, for J1 (and other Judd lodes) it is the top cut 400 g/t data. K92ML supplied H&SC with price and recovery assumptions in order to produce a gold equivalent value to include the copper and silver grades. The Autc_100g/t cut is included to help K92ML's grade control work.

The Gold Equivalent (AuEq) g/t was calculated using the formula:

Au g/t + (Cu %*1.6481) + (Ag g/t*0.0114)

Assumptions include:

- Gold prices of USD $1,700/oz: Silver USD $22.5/oz; Copper USD $4.0/lb.
- Recoveries relative to gold of 93% for copper and 80% for silver.

Table 14-27 Estimation Results for Kora

Au Cut off g/t	Mt	Au g/t	Autc_600 g/t	Autc_100 g/t	Cu %	Ag g/t	Agtc_300 g/t	AuEq g/t	Density t/m^3
K1 Lode									
0.5	11.0	4.36	4.25	3.81	0.77	13.9	13.4	5.78	2.65
1	8.8	5.28	5.14	4.59	0.81	14.8	14.3	6.78	2.65
1.5	7.1	6.26	6.08	5.40	0.85	15.5	15.0	7.84	2.66
1.75	6.3	6.81	6.60	5.85	0.86	15.8	15.2	8.40	2.66
2	5.7	7.30	7.07	6.24	0.87	15.9	15.3	8.90	2.67
2.25	5.2	7.80	7.55	6.64	0.88	16.1	15.5	9.42	2.67
2.5	4.8	8.30	8.04	7.04	0.89	16.0	15.4	9.94	2.67
2.75	4.4	8.78	8.49	7.41	0.90	16.1	15.4	10.43	2.67
3	4.1	9.18	8.87	7.72	0.90	16.0	15.3	10.84	2.67
3.5	3.6	10.00	9.64	8.33	0.89	15.5	14.8	11.62	2.67
4	3.2	10.81	10.42	8.93	0.88	14.9	14.2	12.42	2.67
5	2.6	12.47	11.97	10.10	0.92	15.3	14.4	14.16	2.66
10	1.1	20.57	19.35	14.91	1.02	17.5	15.9	22.42	2.59

Au Cut off g/t	Mt	Au g/t	Autc 1000 g/t	Autc 400 g/t	Autc 100 g/t	Cu %	Ag g/t	Agtc 300 g/t	AuEq g/t	Density t/m³
K2 Lode										
0.5	17.3	5.02	4.66	4.22	3.52	1.39	28.9	26.2	7.24	2.67
1	14.0	6.03	5.50	5.04	4.18	1.49	31.3	28.2	8.27	2.68
1.5	11.3	7.15	6.30	5.94	4.87	1.55	32.6	29.4	9.19	2.70
1.75	10.2	7.75	6.78	6.41	5.22	1.58	33.2	30.0	9.72	2.70
2	9.2	8.44	7.32	6.93	5.61	1.61	33.9	30.6	10.32	2.70
2.25	8.2	9.20	7.92	7.52	6.03	1.66	34.7	31.2	11.02	2.71
2.5	7.3	9.97	8.52	8.09	6.44	1.70	35.1	31.7	11.68	2.71
2.75	6.7	10.72	9.11	8.65	6.84	1.72	35.3	32.0	12.31	2.71
3	6.0	11.54	9.74	9.26	7.25	1.74	35.2	32.0	12.97	2.71
3.5	5.1	13.14	10.96	10.42	8.02	1.75	34.9	32.3	14.21	2.71
4	4.3	14.84	12.25	11.63	8.80	1.72	34.8	32.3	15.45	2.70
5	3.2	18.20	14.81	13.94	10.20	1.67	34.6	32.4	17.93	2.68
10	1.4	32.85	25.68	23.08	14.61	1.42	34.5	32.0	28.38	2.60

Au Cut off g/t	Mt	Au g/t	Autc 400 g/t	Autc 100 g/t	Cu %	Ag g/t	Agtc 300 g/t	AuEq g/t	Density t/m³
Kora Link Lode									
0.5	3.31	3.44	2.50	2.24	0.32	6.5	6.3	4.05	2.63
1	2.17	4.87	3.44	3.04	0.36	7.3	7.2	5.55	2.65
1.5	1.52	6.42	4.38	3.81	0.40	8.1	8.0	7.17	2.66
1.75	1.29	7.28	4.88	4.20	0.42	8.4	8.3	8.07	2.66
2	1.10	8.20	5.39	4.60	0.44	8.6	8.6	9.01	2.66
2.25	0.95	9.19	5.92	5.00	0.45	8.9	8.8	10.03	2.66
2.5	0.83	10.16	6.43	5.38	0.47	9.1	9.0	11.03	2.66
2.75	0.73	11.20	6.95	5.76	0.48	9.3	9.2	12.10	2.66
3	0.65	12.25	7.47	6.12	0.49	9.5	9.4	13.17	2.66
3.5	0.52	14.45	8.50	6.83	0.52	9.8	9.7	15.42	2.65
4	0.43	16.80	9.55	7.51	0.55	10.1	10.0	17.82	2.65
5	0.30	22.01	11.71	8.81	0.60	10.6	10.5	23.12	2.65
10	0.11	48.07	20.11	12.51	0.63	11.9	11.7	49.24	2.61

Observations from the Kora estimation results are:

K1: There is a minor difference between uncut and cut(600g/t) gold data with respect to average gold grade (approximately 3% for the 2.5 g/t cut off). This is probably due to a lack of high grade samples in isolated zones peripheral to the general mined and well drilled area. The abundance of data including the face sampling has been able to control any high grade smearing.

K2: There is a significant difference between the uncut and cut (1,000 g/t) gold data, approximately 14% difference in gold grade for the 2.5 g/t cut off. There is a similar difference (approximately 19%) between the 1,000g/t cut data and the 400 g/t cut data, suggesting that a lower cut than 1,000 g/t might be more appropriate. It is important to note that the difference in gold grade for the different top cuts is most likely due to the dispersed nature of some of the high grades (unlike K1) into more isolated areas where the impact on the surrounding blocks is likely to be greater.

Kora Link: There is a dramatic impact with the 400 g/t top cut giving a 37% drop in gold grade for a 2.5 g/t cut off. Looking at the K1 and K2 results along with the spatial distribution of the high grade (confined to Measured and Indicated) and the relatively small volume of material involved, the overall impact of using uncut data for the Mineral Resources is likely to be small. K92ML has consistently argued that the top cuts suggested by H&SC are too severe.

In Figure 14-54 the relative impact of the top cuts can be seen as part of the grade tonnage curves for the gold estimates. The major discordant grade is the green coloured uncut Kora Link data, but it should be noted that very limited tonnages are associated with the higher grades. However, when plotting the Kora Link 400 g/t cut data (brown colour) the grades are similar to the K1 uncut data (fawn colour) and the grade patterns are similar to K2 1,000 g/t cut and 400 g/t cut.

The K2 400 g/t cut (dark blue) brings the high grades down to better match the K1 uncut data. However, there is a fundamental displacement in the K1 and K2 curves due to the spatial location of the higher grades whereby the majority of the K2 high grade data is in the peripheral zones to the general mine area where there is less control on the grade interpolation. H&SC's preferred position, in contrast to K92ML, on the K2 top cut is for the lower 400 g/t cut because of where these higher grades exist.

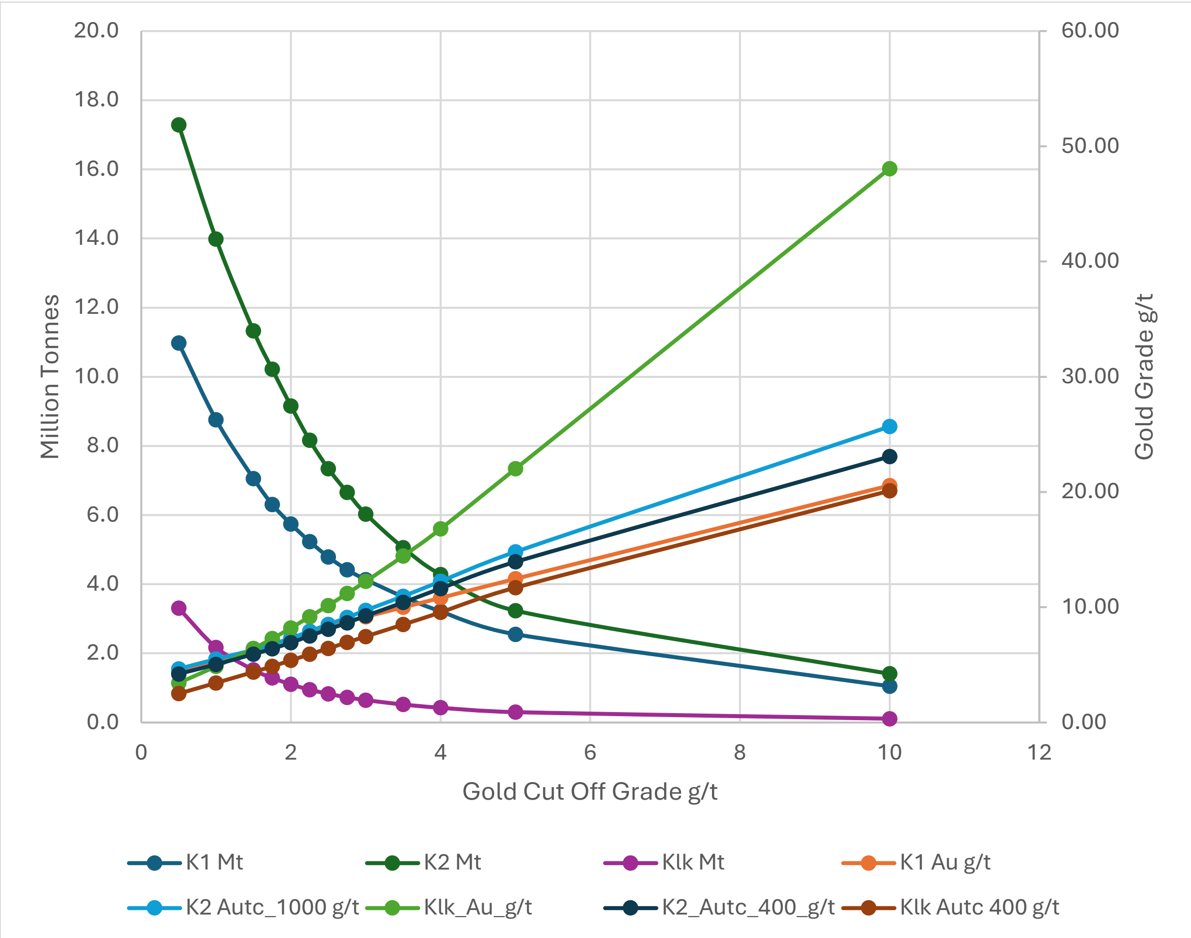
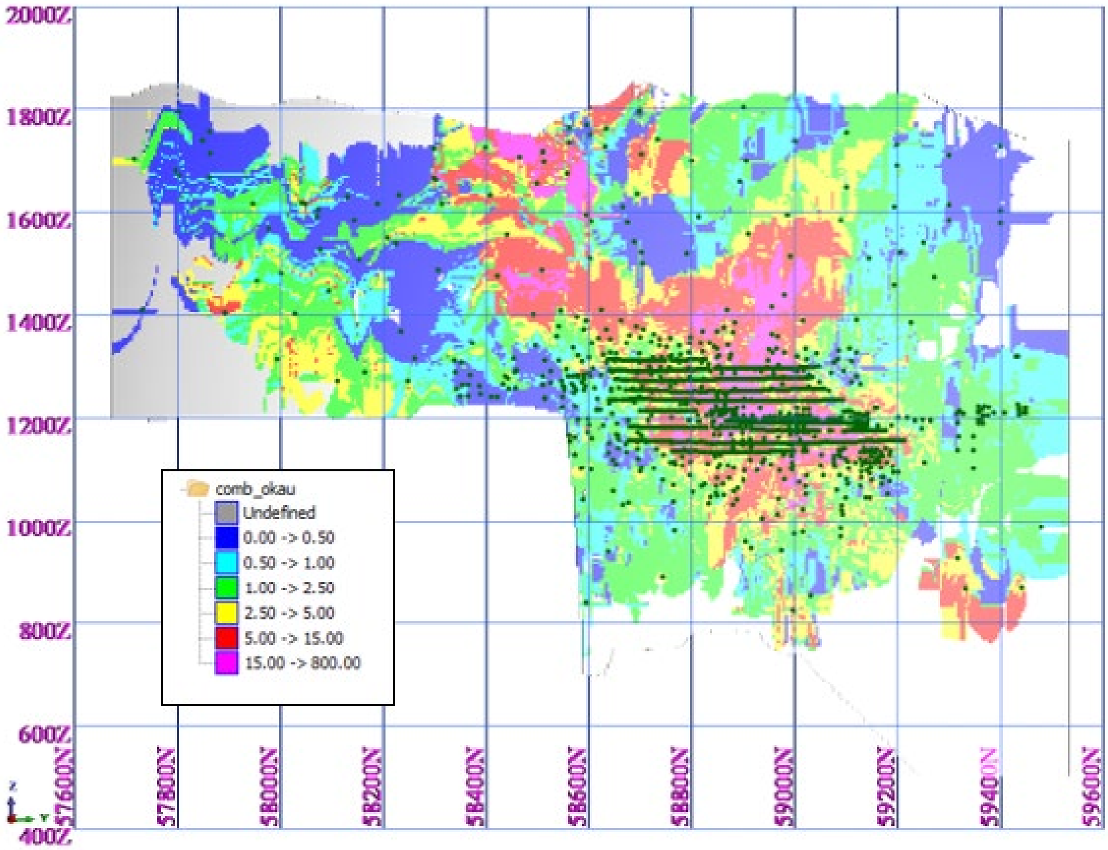


Figure 14-54 Grade Tonnage Curves for the Kora Lodes (H&SC)

Figure 14-55 shows the gold block grade distribution for the K1 lode for all search passes with no cut-off grade. The lode interpretation is shown as a faint grey line/shadow. Speculation suggests a central enriched zone plunging moderately to the north but with possible cross structures causing south plunging blow outs in grade. There is no proof of this concept from underground mapping. Mining depletion has not been removed for clarity purposes.

Other items of note are:

- Reconfiguration of the southern end of the lode equates to a relatively cleaner termination of the mineralization around the 58400mN line. It should be noted that there is a modest resurgence of gold mineralization further south.
- Potential improvements in the search domain design combined with the infill drilling have led to less 'holes' in the block grade interpolation compared to the 2021 model.
- Similar block grade distribution for the most part to the 2021 model.

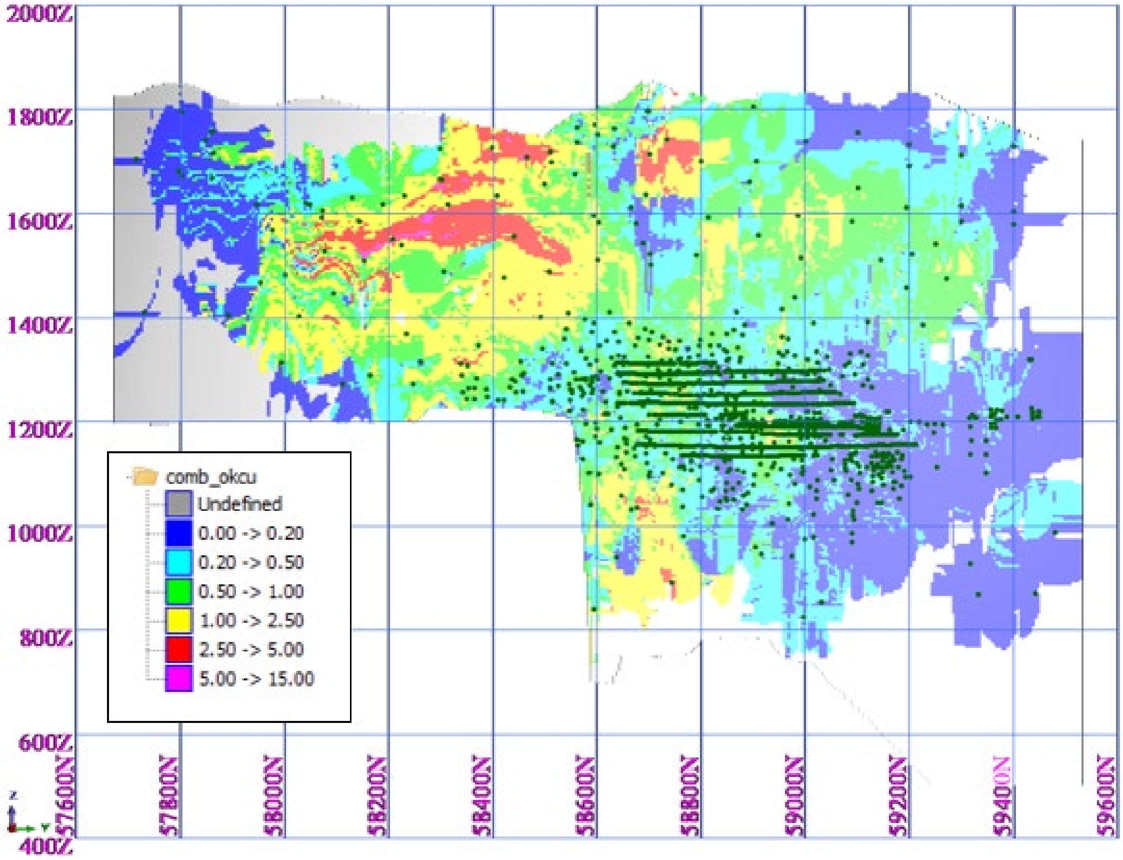


(view : looking west)(green dots = drillhole and face samples pierce points)

Figure 14-55 K1 Lode Gold Block Grade Distribution All Passes Long Section (H&SC)

Figure 14-56 shows the copper block grade distribution for the K1 lode for a zero gold cut-off grade. It shows marked zones of enrichment associated with the old Kora and Eutompi deposits relative to the general Kora mine area. However, it should be noted that there is a possible resurgence of copper grades at the central base of K1. Other items of note are:

- Marked truncation of copper mineralization at the extreme southern end (58000mN) suggests a possible fault truncation. The geological interpretation may need a review.
- Possible flat-lying nature to some of the copper mineralization in the southern upper zone. This trend in mineralization has not been noted before and may be an artefact of the modelling or is indicative of some 'boiling point' zonation associated with the mineral introduction.

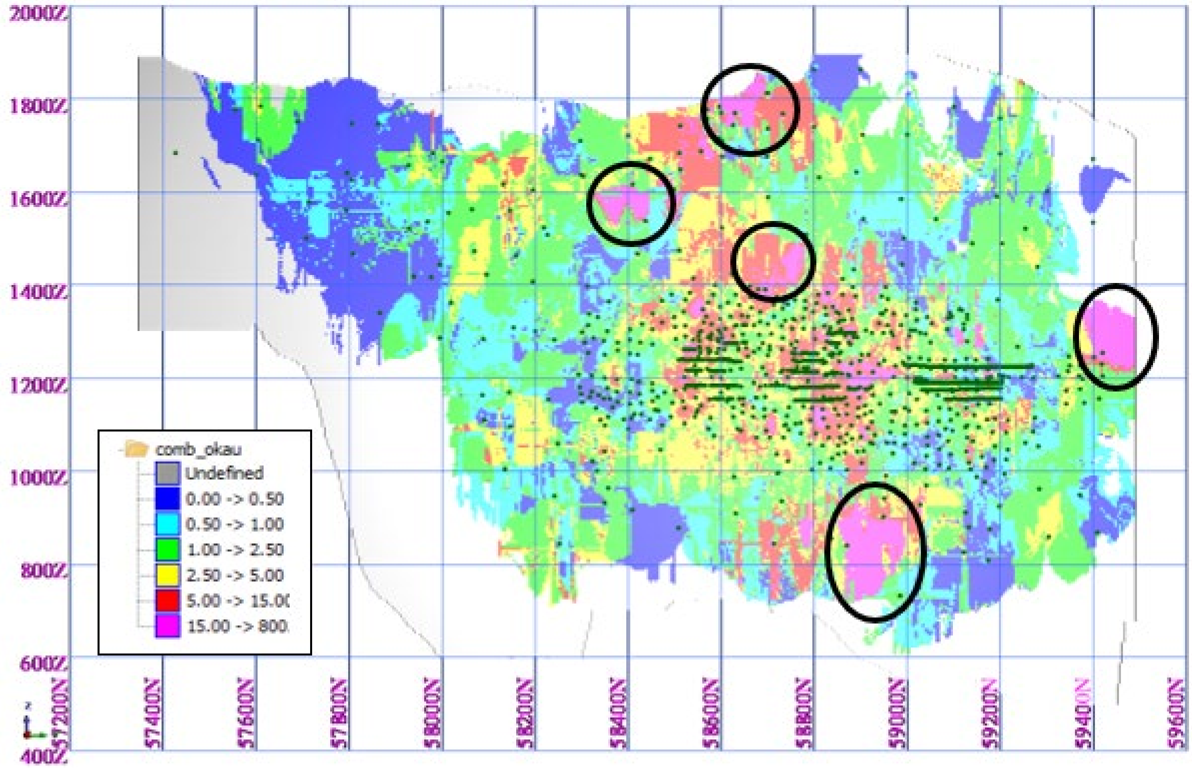


(view : looking west)(green dots = drillhole and face sample pierce points)

Figure 14-56 K1 Copper Block Grade Distribution All Passes Long Section (H&SC)

Figure 14-57 shows the gold block grade distribution for the K2 lode for all search passes with no cut-off grade. Higher grade zones seem to be scattered about as localised lenses, which may be a real feature and not an artifact of the modelling. Other items of note are:

- There is an increase in size of the lode at its southern end with mineralization being open to the south at depth but appears to be lower grade.
- The poddy nature to the mineralization represents a grade continuity risk.
- Additional drilling and refinements to the search domaining improved modelling resulting in a more coherent mineral zone.
- There are at least 5 suspect high grade zones due to isolated high grades in the drilling (black circles). These zones do represent a higher risk in which follow up drilling may not match the expected grades.
- There appears to be a southern upper termination to the gold mineralization that is relatively abrupt suggesting a possible fault truncation to mineralization. This truncation is in a similar position for the copper zone in K1, but not necessarily the gold termination point in K1. More study of the model by K92ML is recommended.

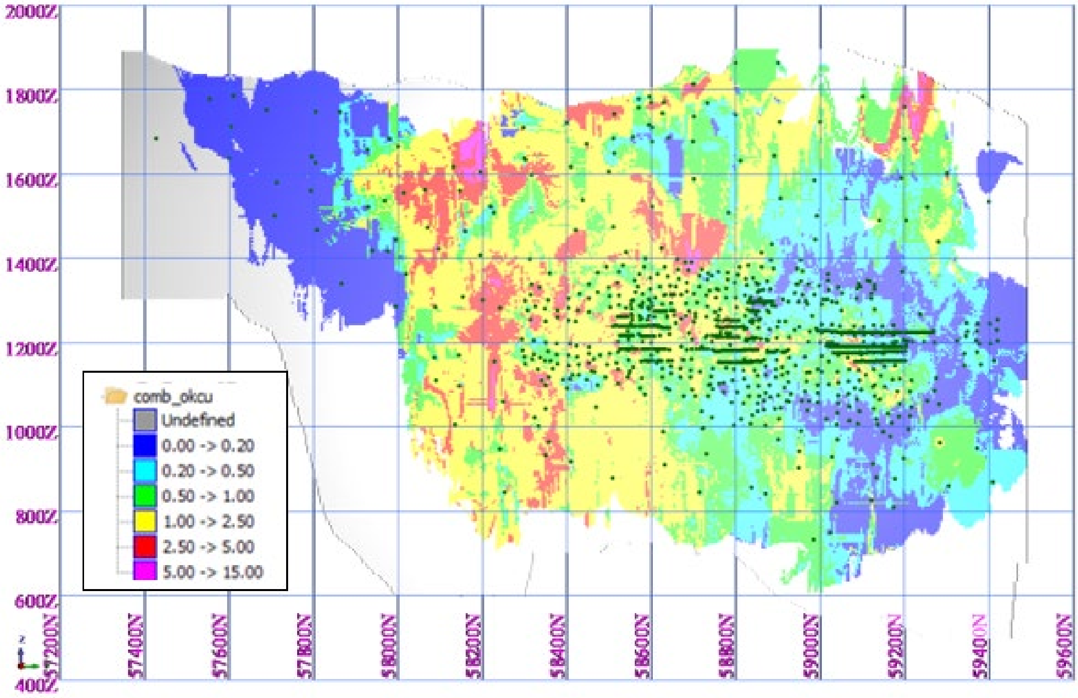


(view : looking west)(green dots = drillhole and face sample pierce points)

Figure 14-57 K2 Block Grade Distribution All Passes Long Section (H&SC)

Figure 14-58 shows the copper block grade distribution for the K2 lode for a zero gold cut-off grade. It shows an overall elevation in copper values relative to K1 and has a higher grade in parts of the Kora area as per the K1 lode. Other items of note are:

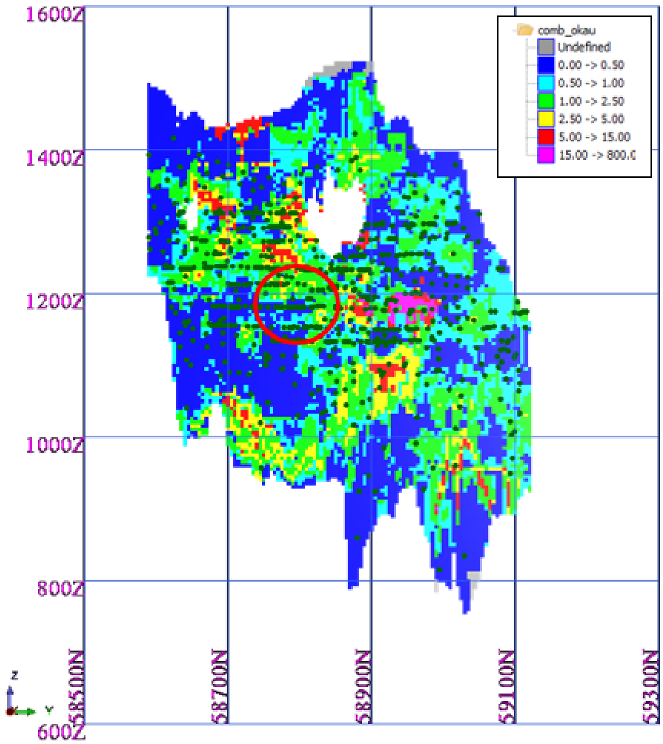
- There is an expansion of the copper mineralization at the southern end of the lode and it remains open down plunge to the south (a potential lack of faulting permitting).
- A severe truncation of the lode occurs in the southernmost upper quarter (5800mN line) that is reasonably supported by the K1 copper image and the K2 gold image. This looks like a fault termination to the mineralization. H&SC has speculated that the mineral intercepts in holes deemed part of the Judd Lode system were not part of the Judd but actually were fault off-setted K1 and K2 lodes.



(view : looking west)(green dots = drillhole and face sample pierce points)

Figure 14-58 K2 Copper Block Grade Distribution All Passes Long Section (H&SC)

Figure 14-59 shows the gold grade distribution for the Kora Link lode for all search passes. The main difference from the 2021 resource model is the removal of what transpired to be K1 material from the upper southern part of the Kora Link. Elsewhere the rather patchy nature to the higher grade mineralization is reminiscent of the K2 lode. The red circle highlights a small zone of very high grade material which has produced the significant kick in gold grades for the lode relative to K1 and K2.



(view : looking west)(green dots = drillhole and face sample pierce points)

Figure 14-59 Kora Link Lode Gold Block Grade Distribution All Passes Long Section (H&SC)

The global estimation results for Judd with mining depletion removed (up to the September 2023) are reported for a range of different gold cut-off grades as shown in Table 14-28. Estimation results are reported for all block centroids inside the relevant mineral wireframe (in/out basis). Classification of the estimates is included later in this chapter.

Table 14-28 Estimation Results for Judd

Au Cut off g/t	Mt	Au g/t	Autc_400 g/t	Autc_100 g/t	Cu %	Ag g/t	Agtc_300 g/t	AuEq g/t	Density t/m^3
J1 Lode									
0.5	5.01	5.16	4.43	3.55	0.52	11.2	11.0	5.42	2.64
1	3.62	6.86	5.85	4.63	0.60	12.5	12.5	6.98	2.64
1.5	2.79	8.53	7.22	5.65	0.64	13.5	13.4	8.43	2.64
1.75	2.57	9.13	7.71	6.00	0.66	13.9	13.8	8.95	2.63
2	2.36	9.76	8.21	6.35	0.67	14.3	14.3	9.48	2.63
2.25	2.19	10.37	8.70	6.69	0.68	14.7	14.6	9.98	2.63
2.5	2.02	11.02	9.21	7.04	0.67	15.0	14.9	10.49	2.62
2.75	1.86	11.76	9.79	7.43	0.67	15.4	15.3	11.07	2.62
3	1.73	12.43	10.31	7.78	0.67	15.7	15.6	11.59	2.61
3.5	1.51	13.76	11.34	8.43	0.66	16.2	16.0	12.61	2.61
4	1.35	14.95	12.24	8.99	0.66	16.5	16.4	13.52	2.61
5	1.09	17.41	14.07	10.05	0.66	17.2	17.1	15.35	2.62
10	0.50	29.98	22.65	13.94	0.66	18.9	18.5	23.95	2.64

Au Cut off g/t	Mt	Au g/t	Autc_400 g/t	Autc_100 g/t	Cu %	Ag g/t	Agtc_300 g/t	AuEq g/t	Density t/m^3
J2 Lode									
0.5	2.93	1.81	1.81	1.81	0.39	9.0	8.9	2.56	2.54
1	1.45	2.92	2.92	2.92	0.41	9.9	9.9	3.71	2.54
1.5	0.82	4.24	4.24	4.24	0.38	9.4	9.4	4.98	2.54
1.75	0.67	4.82	4.82	4.82	0.37	9.4	9.4	5.53	2.54
2	0.58	5.26	5.26	5.26	0.37	9.5	9.5	5.97	2.54
2.25	0.51	5.69	5.69	5.69	0.35	9.5	9.5	6.38	2.54
2.5	0.46	6.08	6.08	6.08	0.35	9.4	9.4	6.76	2.54
2.75	0.41	6.50	6.50	6.50	0.33	9.4	9.4	7.16	2.54
3	0.36	7.02	7.02	7.02	0.33	9.5	9.4	7.67	2.54
3.5	0.28	8.10	8.10	8.10	0.32	9.5	9.5	8.74	2.54
4	0.23	8.99	8.99	8.99	0.32	9.5	9.5	9.62	2.54
5	0.15	11.33	11.33	11.33	0.33	9.8	9.8	11.99	2.54
10	0.08	15.54	15.54	15.54	0.33	10.5	10.5	16.21	2.54

Au Cut off g/t	Mt	Au g/t	Autc_400 g/t	Autc_100 g/t	Cu %	Ag g/t	Agtc_300 g/t	AuEq g/t	Density t/m^3
J3 Lode									
0.5	0.34	1.14	1.14	1.14	0.52	12.3	12.3	2.13	2.65
1	0.15	1.65	1.65	1.65	0.55	14.0	14.0	2.73	2.65
1.5	0.07	2.08	2.08	2.08	0.39	11.3	11.3	2.86	2.65
1.75	0.05	2.31	2.31	2.31	0.18	6.8	6.8	2.69	2.65
2	0.03	2.47	2.47	2.47	0.17	6.3	6.3	2.82	2.65
2.25	0.03	2.53	2.53	2.53	0.17	6.2	6.2	2.87	2.65
2.5	0.02	2.60	2.60	2.60	0.16	6.4	6.4	2.94	2.65
2.75	0.002	3.07	3.07	3.07	0.16	7.0	7.0	3.41	2.65
3	0.002	3.17	3.17	3.17	0.16	7.0	7.0	3.52	2.65

Au Cut off g/t	Mt	Au g/t	Autc 400 g/t	Autc 100 g/t	Cu %	Ag g/t	Agtc 300 g/t	AuEq g/t	Density t/ m3
J1W Lode									
0.5	0.67	5.43	5.43	4.81	0.57	13.4	13.4	6.52	2.65
1	0.52	6.89	6.89	6.08	0.64	15.5	15.5	8.11	2.65
1.5	0.42	8.11	8.11	7.13	0.62	16.1	16.1	9.31	2.65
1.75	0.40	8.48	8.48	7.44	0.59	15.9	15.9	9.63	2.65
2	0.36	9.16	9.16	8.02	0.52	14.6	14.6	10.19	2.65
2.25	0.34	9.74	9.74	8.51	0.49	14.3	14.3	10.71	2.65
2.5	0.32	10.15	10.15	8.84	0.46	14.1	14.1	11.07	2.65
2.75	0.30	10.57	10.57	9.19	0.46	14.3	14.3	11.49	2.65
3	0.29	10.95	10.95	9.50	0.46	14.5	14.5	11.87	2.65
3.5	0.25	12.10	12.10	10.44	0.47	15.5	15.5	13.06	2.65
4	0.23	12.88	12.88	11.06	0.48	16.0	16.0	13.85	2.65
5	0.19	14.42	14.42	12.27	0.48	16.5	16.5	15.39	2.65
10	0.10	21.24	21.24	17.32	0.36	18.6	18.6	22.05	2.65

Observations from the estimation results are:

- J1: There is a significant drop in grade with the application of a 400 g/t top cut and a similar drop to the 100 g/t cut. From inspection of detailed data it is noticeable that higher grades are associated with the Pass 4 and 5 searches, suggesting that there are isolated high grade samples in peripheral areas having an undue effect on the block grades.
- J2 is a relatively low grade lode with no top cutting considered necessary.
- J3: the same can be said for J3 as for J2.
- J1W: Interestingly the gold grades for J1W are similar to J1 and suggest some possible link. K92ML originally supplied a wireframe for a new lode called K1S, which was interpreted to be a linking structure between K1 and J1. The interpretation lacked geological support in places with H&SC using the northern part of the lode to better define K1 and the spatially separate (geologically) southern part delineated as the J1W lode.
- Figure 14-60 shows the gold grades for the Judd lodes for different gold cut off grades. Whilst overall tonnages are different there appears to be some similarity between the gold grades for J1 and J1W lodes. This supports both the supposition that they are related and that the splitting of K1S was justified.

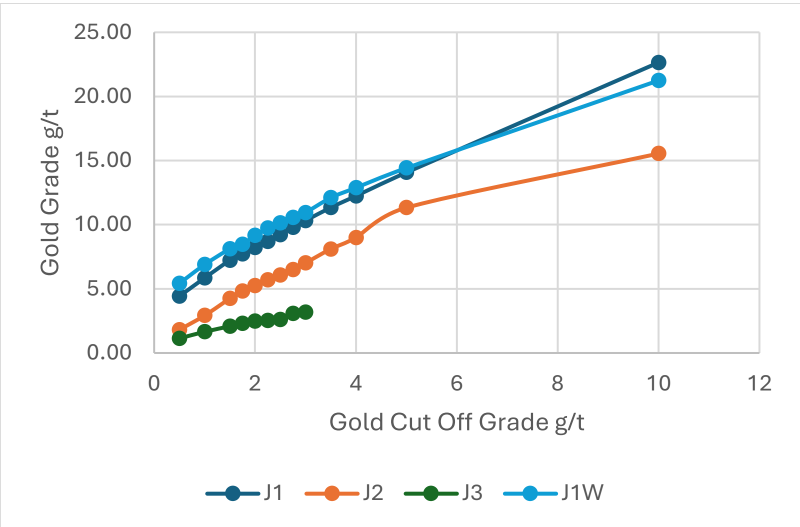
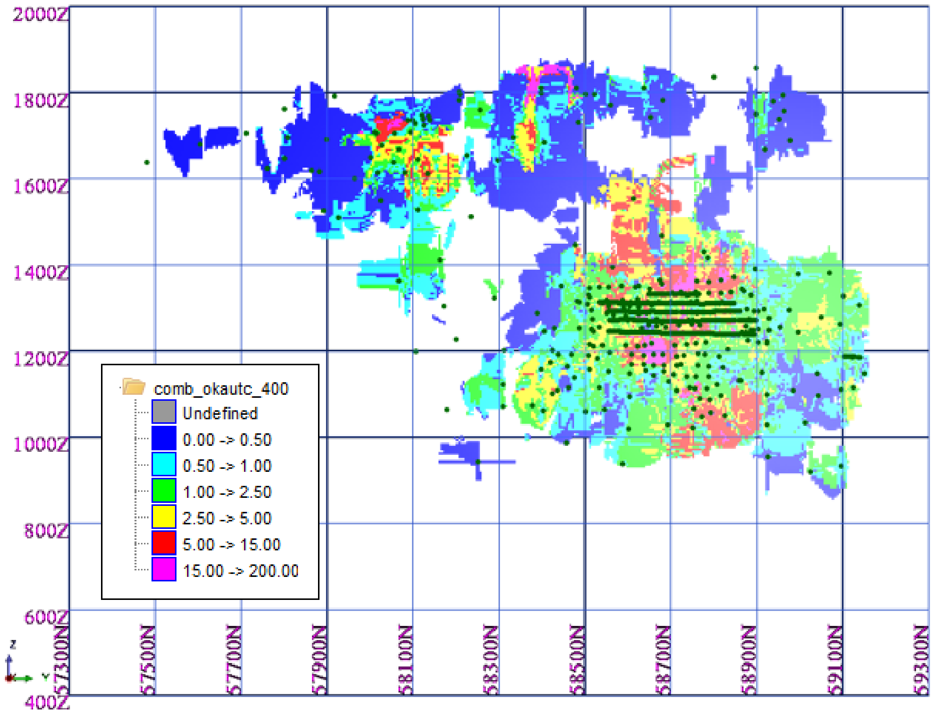


Figure 14-60 Gold Grade Curves for the Combined Judd Lodes (H&SC)

Figure 14-61 shows the gold block grade distribution for the J1 lode for all search passes with a zero gold cut-off grade. Figure 14-62 shows the copper block grade distribution for the J1 lode for all search passes with a zero gold cut-off grade. Mining depletion has not been removed for clarity purposes.

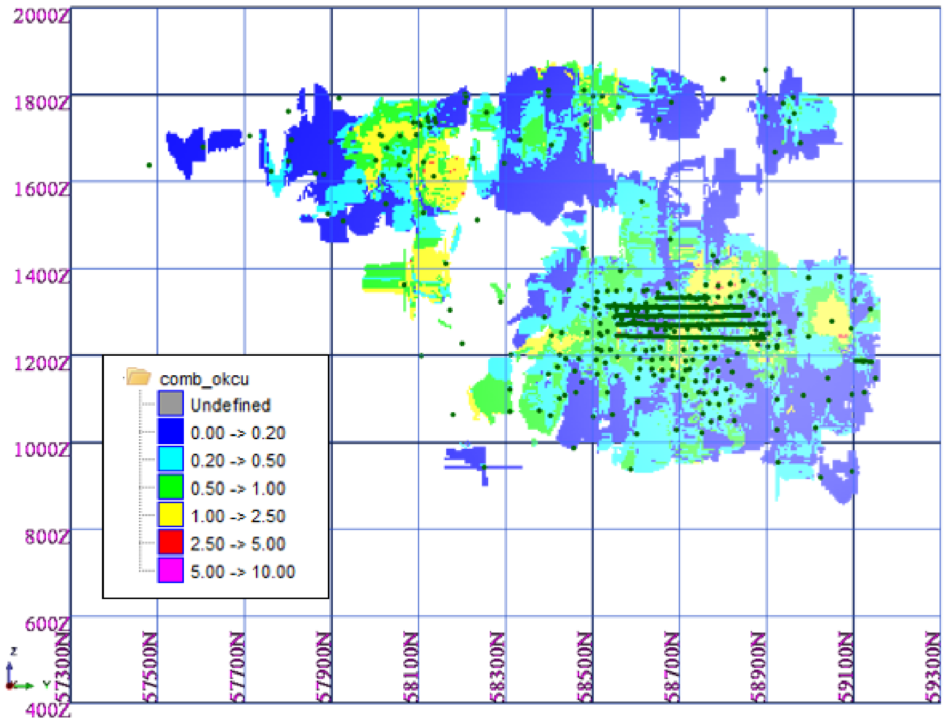
Figure 14-63 shows the gold block grade distribution for the J2 lode for all search passes with a zero gold cut-off grade.

Figure 14-64 shows the copper block grade distribution for the J2lode for all search passes with a zero gold cut-off grade.



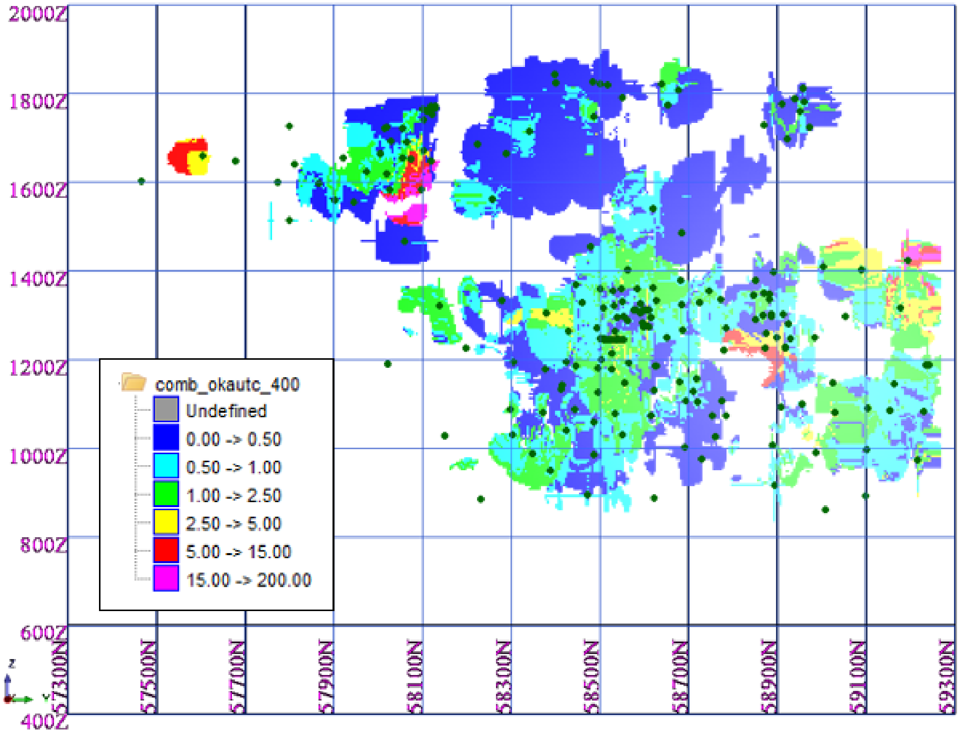
(view : looking west)(green dots = drillhole and face sample pierce points)

Figure 14-61 J1 Lode Gold Block Grade Distribution All Passes Long Section (H&SC)



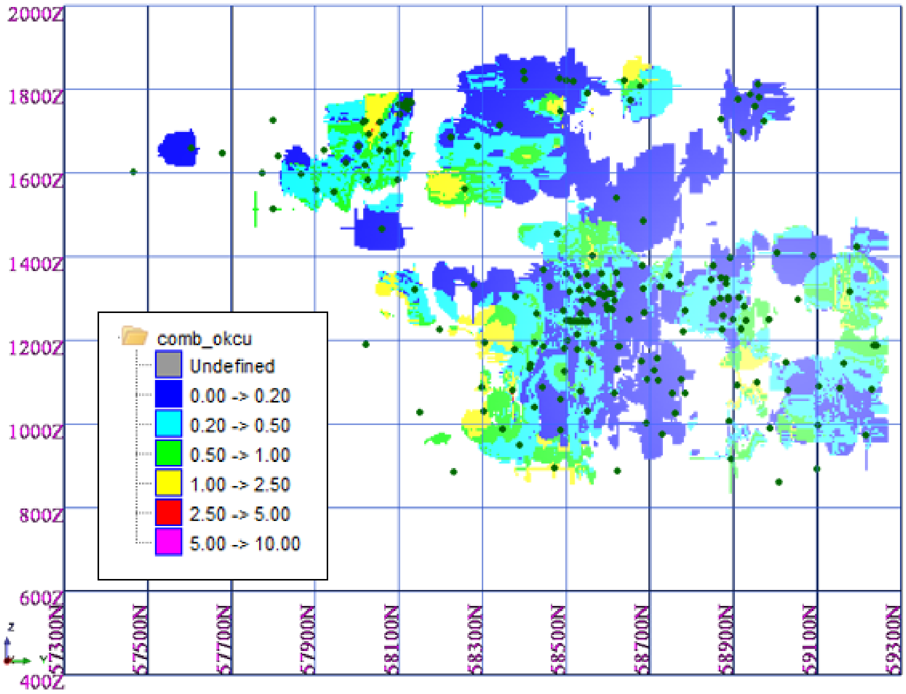
(view : looking west)(green dots = drillhole and face sample pierce points)

Figure 14-62 J1 Lode Copper Block Grade Distribution   All Passes   Long Section   (H&SC)



(view : looking west)(green dots = drillhole and face sample pierce points)

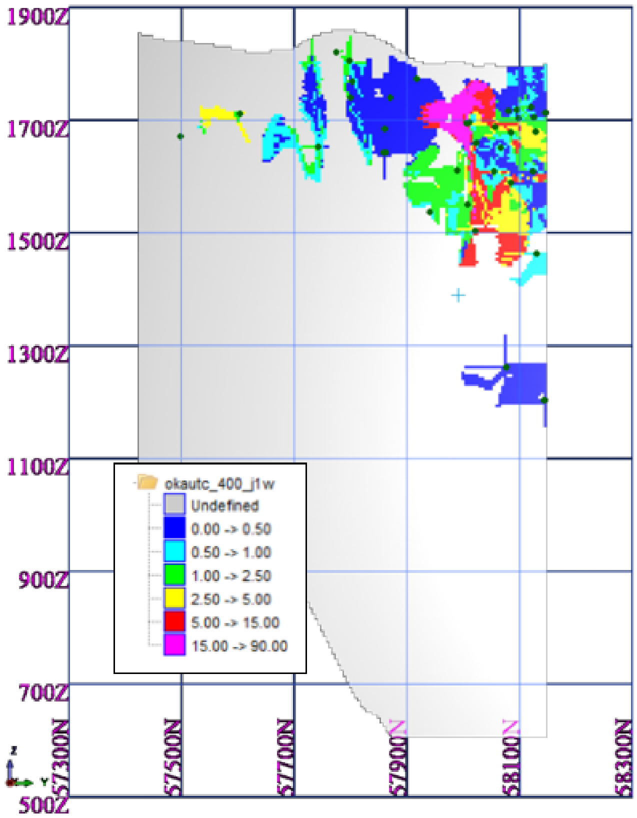
Figure 14-63 J2 Lode Gold Block Grade Distribution   All Passes   Long Section   (H&SC)



(view : looking west)(green dots = drillhole and face sample pierce points)

Figure 14-64 J2 Lode Copper Block Grade Distribution   All Passes   Long Section   (H&SC)

Gold mineralization in J3 is low grade whilst the higher grade mineralization in J1W is located in its upper northern sector (Figure 14-65).



(view : looking west)(green dots = drillhole and face sample pierce points)

Figure 14-65 J1W Lode Gold Block Grade Distribution All Passes Long Section (H&SC)

14.7 Block Model Validation

Block model validation has consisted of visual inspection of block grades against drillhole assay grades and composite values, comparison of summary statistics for block grades and composite values, cumulative frequency curves for global block grades and composites, check models and reconciliation with mill feed.

Block grades were viewed in section against both drillhole assays and composite values. Minor discrepancies were noted for the more peripheral drillholes. The narrowness of the sample intervals and the block size make visual representations difficult to show clearly.

Figure 14-66 shows a cross section example of gold block grades versus drillhole assays for the K1 (red dash), K2 (green dash) and Kora Link (light brown dash) lodes. The comparison is reasonable considering the section window is 10 m and the tenor of the surrounding drillhole and face sampling data. Note how K1 and K2 are discrete lodes whereas the Kora Link is a mixture of limited length lenses of variable grade. Brown solid lines represent development.

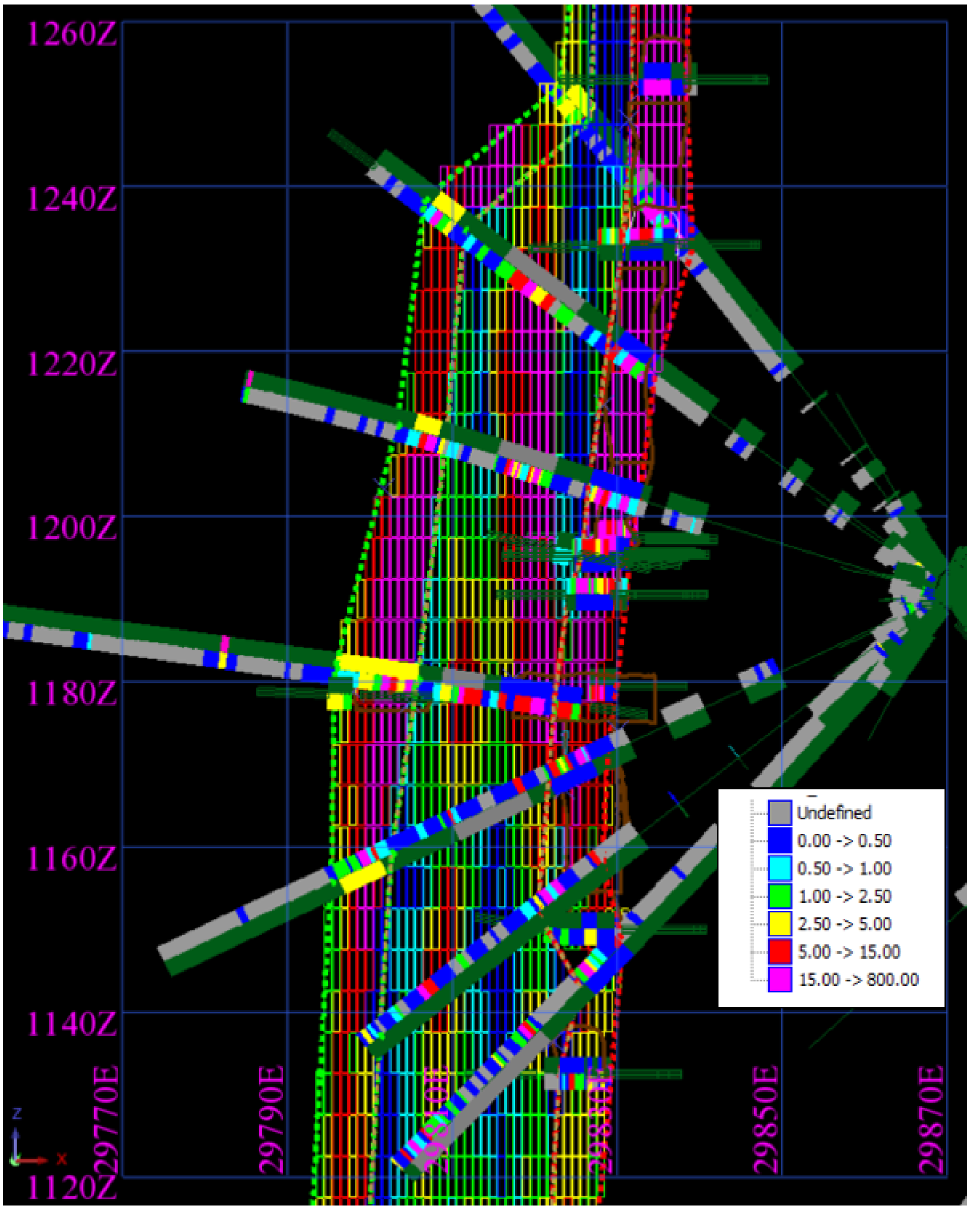


Figure 14-66 Kora Gold Block Grade & Drillhole Assay Comparison Cross Section 58900mN (H&SC)

Figure 14-67shows a cross section example of gold block grades versus drillhole assays for the J1 (green dash) and J2 (red dash) lodes. Again, the results are reasonable considering the 10 m section window and the tenor and limits of the surrounding face sampling and drillhole data.

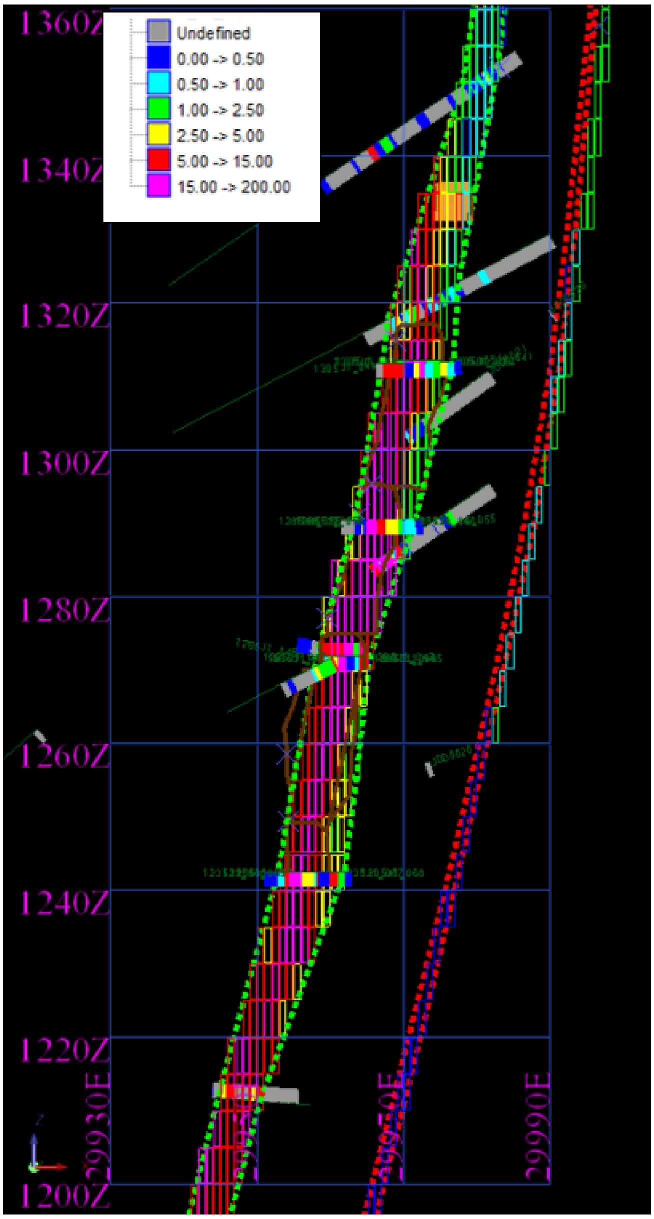


Figure 14-67 J1 & J2 Gold Block Grade & Drillhole Assay Comparison Cross Section 58620mN (H&SC)

Figure 14-68 and Figure 14-69 are cross section examples of gold block grades (drillholes and face sampling) against composite values for the K1 and K2 lodes respectively (zoom on image for better resolution).

The main item to note in Figure 14-68 is the slightly oblique orientation of the mineralization to the general geological dip in the upper half of the section. This may be a function of the grade interpolation or it signifies a change in the grade continuity direction via oblique angled faulting or ductile structural (sigmoidal) deformation e.g. a reidel shear.

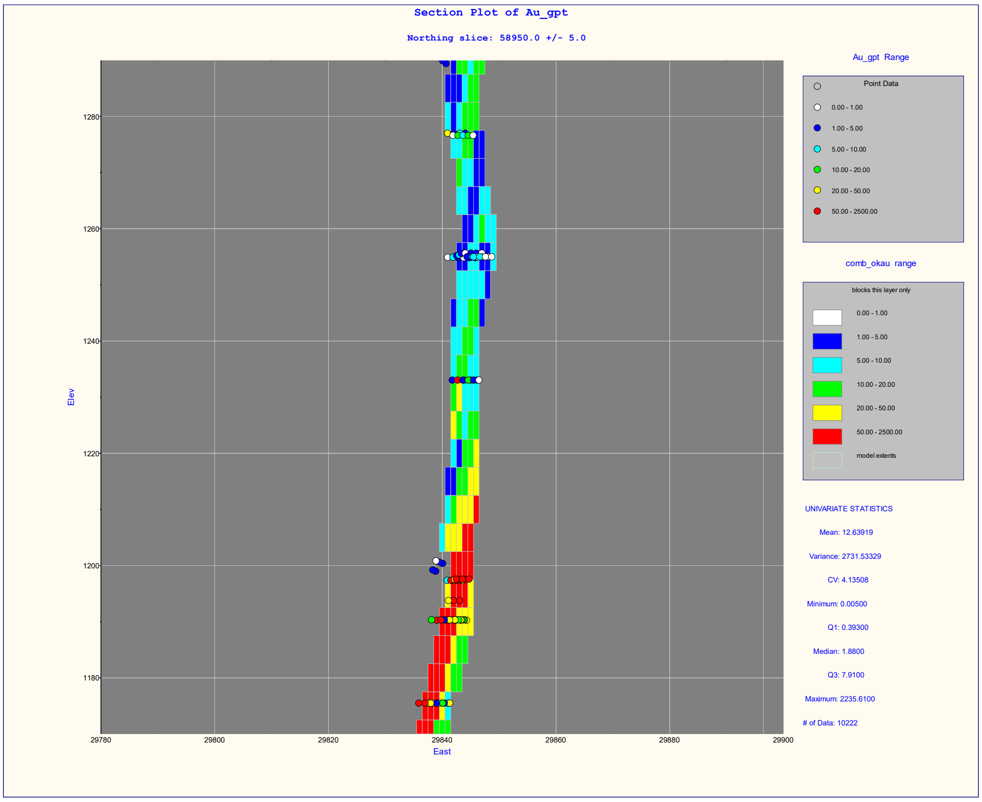


Figure 14-68 Au Block Grade and Composite Value Comparison for the K1 Lode Cross Section 58950mN (H&SC)

In Figure 14-69 the main item to note for K2 is the dogleg nature of the lode in the upper half of the section with the associated change in dip angle which may indicate an off-setting fault. Also of note is the general lensoid nature to the gold mineralization which is typical of K2.

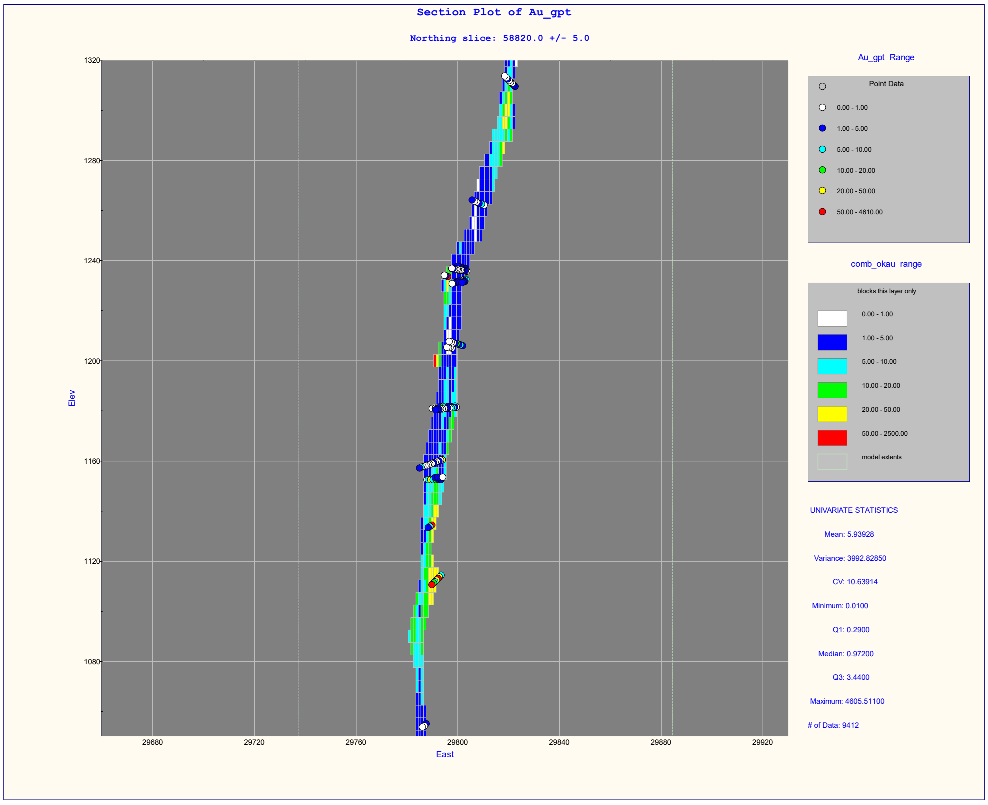


Figure 14-69 Au Block Grade and Composite Value Comparison for the K2 Lode Cross Section 58820mN (H&SC)

Figure 14-70 is a plan view of the K1 face sampling at the 1230mRL. There are two items to note, with the first being the dogleg in strike coinciding with a drop in gold grade, possibly indicating a fault impacted zone with modest dextral movement. Secondly, in the northern half of the image there is the delineation of a narrower higher grade zone down the central axis of the lode, which is a common feature for K1. This high grade zone then seems to migrate to the footwall structural position confirming the observed grade continuity range of 20-40 m in the strike direction.

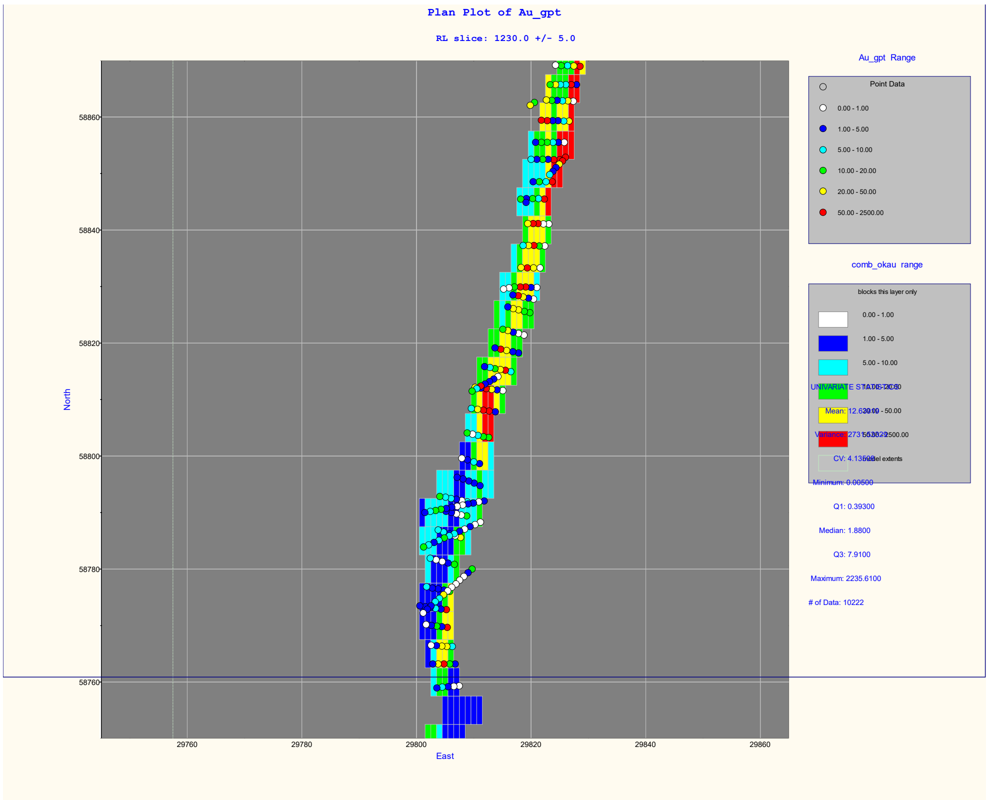


Figure 14-70 Au Block Grade and Composite Value Comparison for the K1 Lode Plan 1230mRL (H&SC)

Figure 14-71 is a plan view of the K2 drilling and face sampling at the 1260mRL. A distinct dogleg exists in the centre of the view which may be fault related. Also prevalent is the lensoid and variability nature of the mineralization.

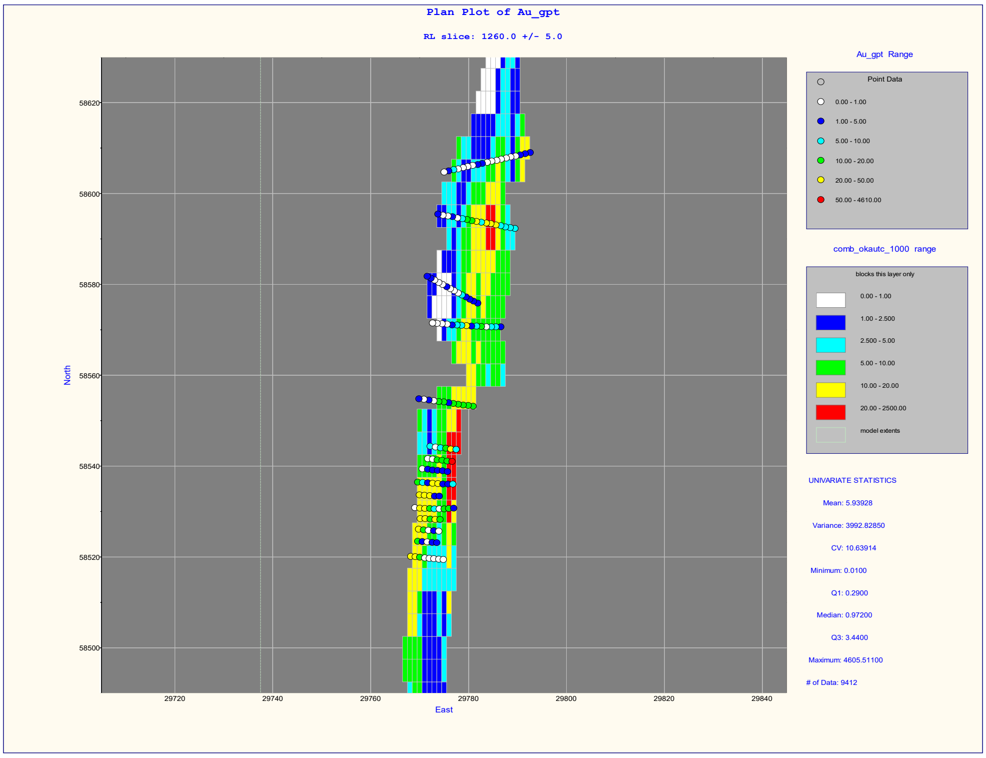


Figure 14-71 Au Block Grade and Composite Value Comparison for the K2 Lode   Plan 1260mRL   (H&SC)

Figure 14-72 is a cross section example of gold block grades against drillhole composite values for the Kora Link lode. Clearly there is an implication of a fault structure causing the dogleg in the centre of the section with a resulting change in grade pattern across that point.

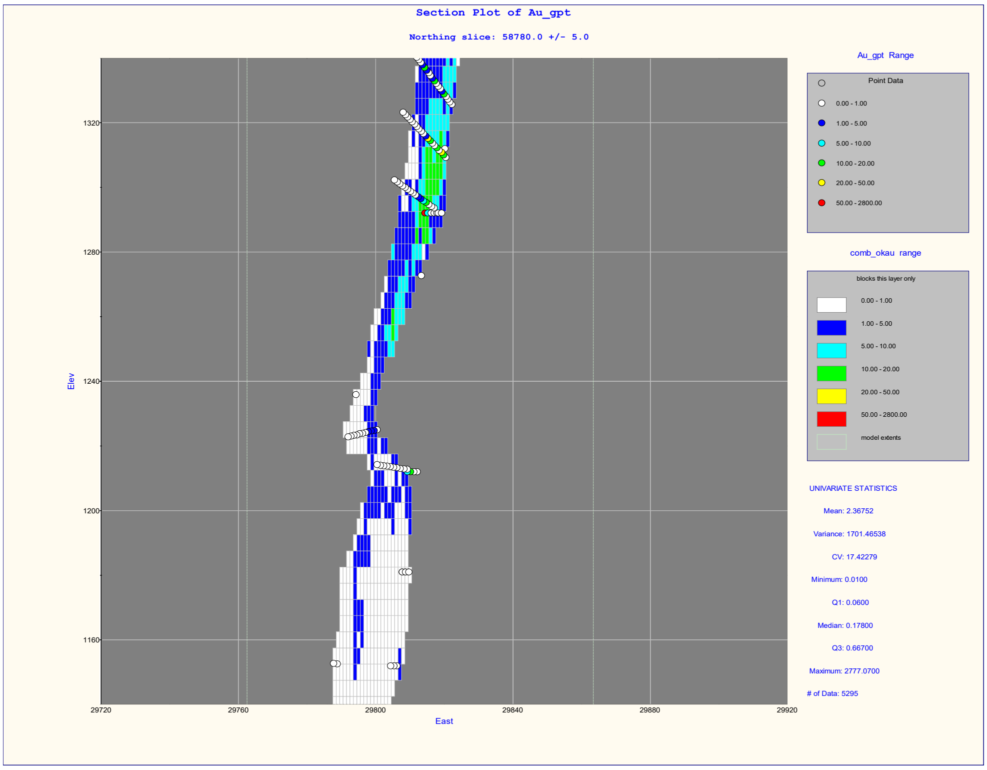


Figure 14-72 Au Block Grade and Composite Value Comparison for the Kora Link Lode   Cross Section 58780mN (H&SC)

Figure 14-73 is a plan view of the Kora Link drilling and face sampling at the 1180mRL level. In contrast to other parts of the lode the grade continuity is good albeit that there are still high grade lenses at different structural positions within the broader zone.

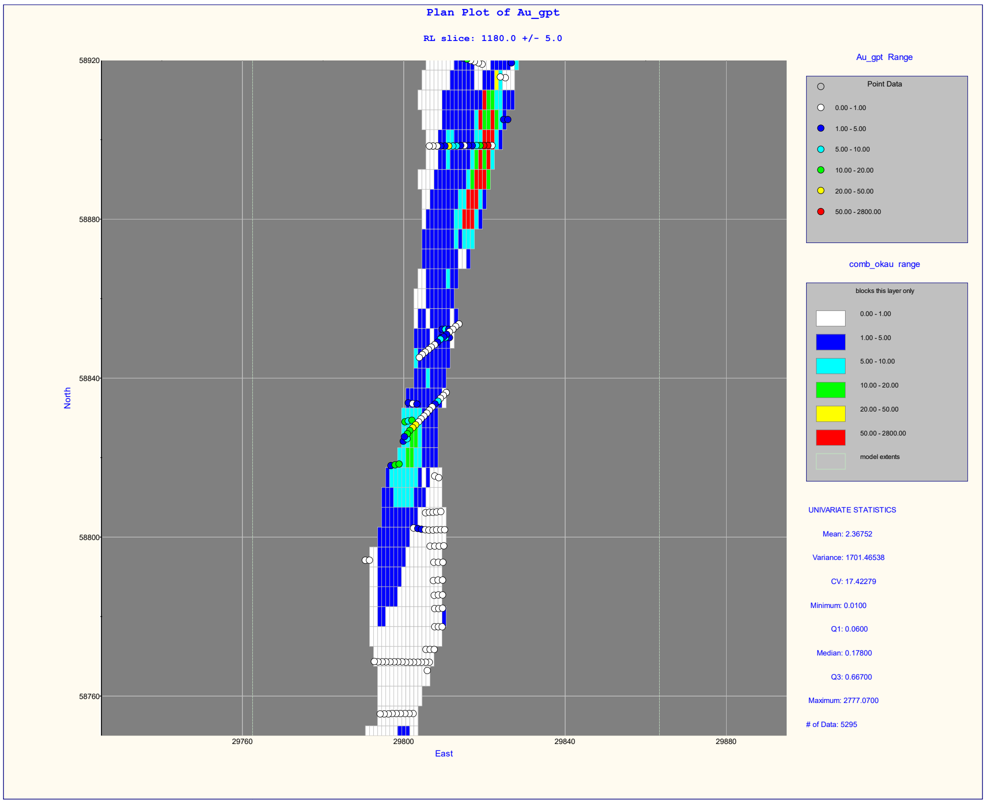


Figure 14-73 Au Block Grade and Composite Value Comparison for the Kora Link Lode Plan 1180mRL (H&SC)

Figure 14-74 shows a cross section example of cut gold block grades against composite values for the J1 Lode. It indicates the lensoid nature to the higher grade mineralization.

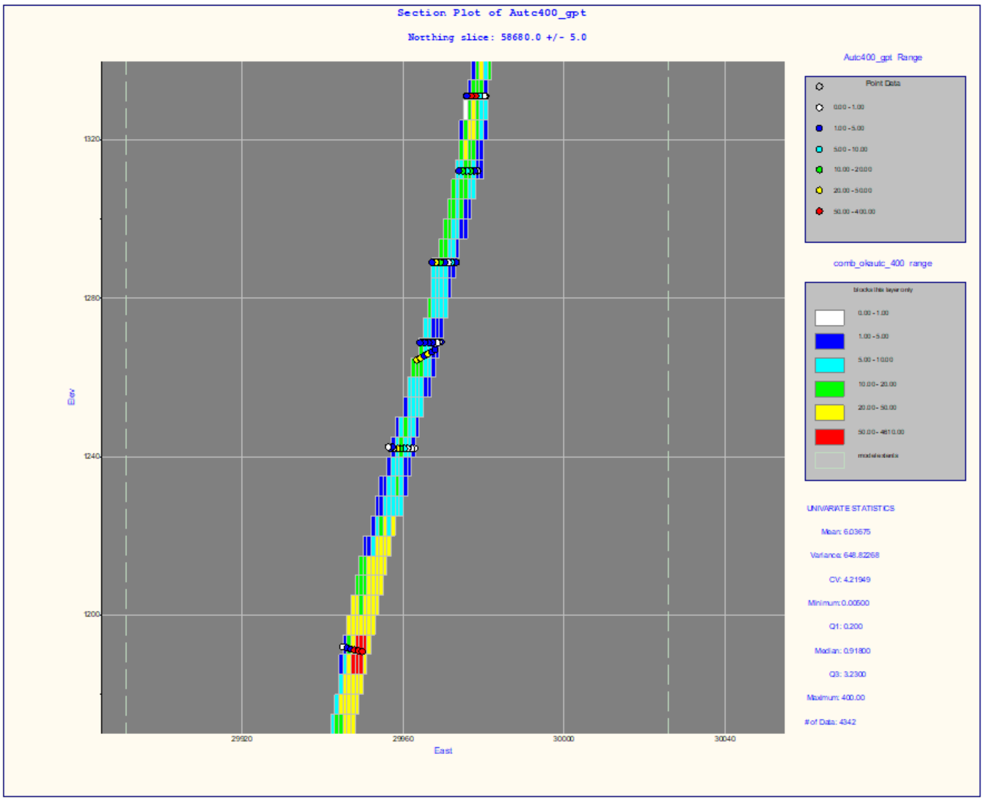


Figure 14-74 Au Block Grade and Composite Value Comparison for the J1 Lode Cross Section 58680mN (H&SC)

Figure 14-75 shows a plan view of cut gold block grades against composite values (face samples and drillholes) for the J1 Lode. It again indicates the lensoid nature to the higher grade mineralization. It also provides an indication of the narrowness of the main mineral zone within the overall lode system.

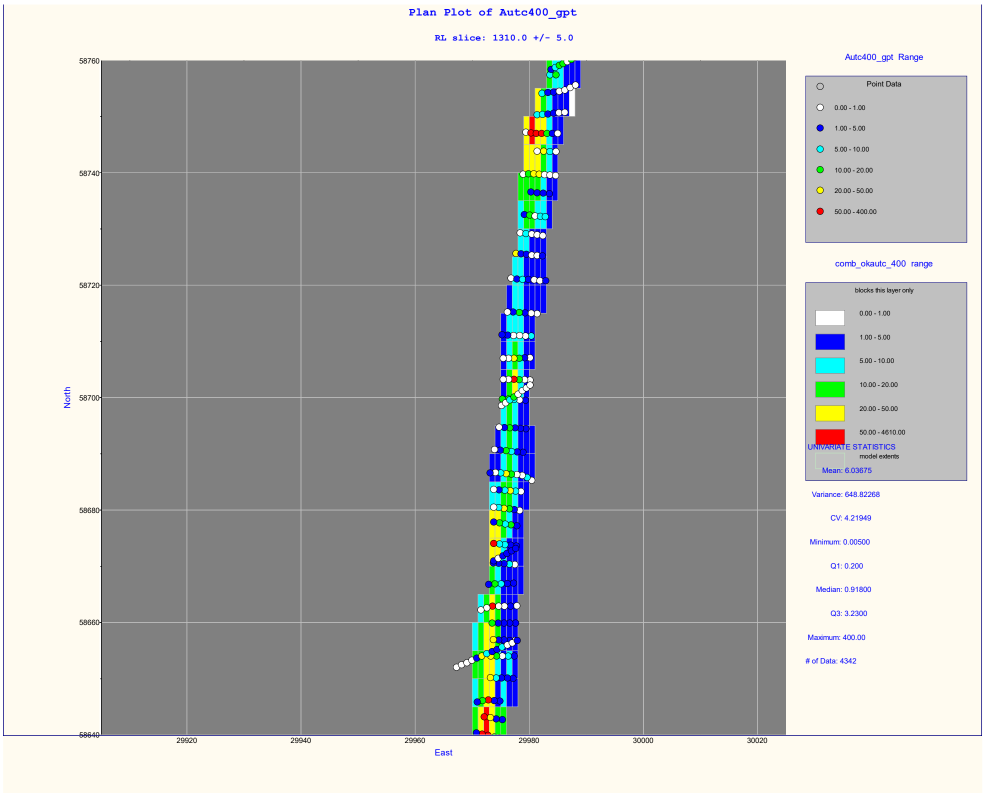


Figure 14-75 Au Block Grade and Composite Value Comparison for the J1 Lode Plan 1310mRL (H&SC)

A comparison of composite and global block grade means would normally be expected to show the composite mean being higher than the block grade mean. This is clearly the case for gold for the K1 lode as shown in Table 14-29. It indicates no obvious issues with the grade interpolation and suggests that the uncut high gold grades, particularly from the face sampling, have been managed by the modelling. However, copper and silver show higher block means than the composite averages, which is attributed to higher copper and silver composite values in areas of widely spaced drilling, where the higher values are likely to have a greater impact on the magnitude of the block grades and the number of higher block grades. In addition, a large majority of the composites, mainly from the face sampling, are in an area of lower copper grades and thus would tend to have the effect of lowering the average copper composite grade (and silver-noting its weak correlation with copper). It should also be noted that the pass 1 and pass 2 mean copper and silver grades are much lower than the means for the remaining pass 3.

Table 14-29 Comparison of Summary Statistics for Composites & Block Grades for the K1 Lode

K1 lode	Gold		Copper		Silver	
	Comp	Block	Comp	Block	Comp	Block
Mean	12.64	4.21	0.59	0.73	9.55	12.88
Median	1.88	1.69	0.19	0.42	3.83	7.20
Standard Deviation	52.27	10.07	1.29	0.85	22.54	15.34
Coeff of Variation	4.14	2.39	2.20	1.16	2.36	1.19
Minimum	0.005	0	0.0005	0.01	0.01	0.02
Maximum	2,235.61	816.13	22.5	9.71	300	174.29
Count	10,222	21,3454	10,222	213,454	10,222	213,454

Table 14-30 shows the summary statistics comparison for the K2 lode composites and global block grades. The gold means behave according to expectation but for both silver and copper the block means are slightly higher than the respective composite means. In the case of copper this is put down to a large number of relatively higher grade blocks on the periphery of the deposit having the significant impact of raising the block mean. It should also be noted that the pass 1 and pass 2 mean copper grades are lower than the means for the remaining pass 3, which implies that the face sampling, which accounts for a substantial portion of the composite data, is in an area of relatively lower copper grade. The difference in the means for silver is likely due to the same reasons as for copper.

Table 14-30 Comparison of Summary Statistics for Composites & Block Grades for the K2 Lode

K2 lode	Gold		Copper		Silver	
	Comp	Block	Comp	Block	Comp	Block
Mean	5.25	4.16	1.20	1.24	21.82	23.58
Median	0.97	1.75	0.45	0.93	8.00	17.55
Standard Deviation	26.80	10.72	2.05	1.22	40.73	21.58
Coeff of Variation	5.10	2.57	1.70	0.98	1.87	0.92
Minimum	0.01	0.01	0.0002	0	0.01	0.05
Maximum	1,000	431.88	26.5686	15.92	300	237.04
Count	9,412	307,149	9,412	307,149	9,412	307,149

Table 14-31 shows the summary statistics comparison for the Kora Link lode. The gold, copper and silver composite means are higher than the block grade means which would be a normal expectation and helps to validate the geological interpretation and the grade interpolation method.

Table 14-31 Comparison of Summary Statistics for Composites & Block Grades for the Kora Link Lode

Kora Link lode	Gold		Copper		Silver	
	Comp	Block	Comp	Block	Comp	Block
Mean	2.37	2.02	0.29	0.26	5.02	4.84
Median	0.18	0.62	0.09	0.18	2.00	3.34
Standard Deviation	41.25	12.45	0.71	0.25	13.57	4.65
Coeff of Variation	17.42	6.16	2.47	0.99	2.70	0.96
Minimum	0.01	0.01	0.0002	0	0.05	0.67
Maximum	2,777.07	850.94	17.4319	4.98	300	115.19
Count	5,295	91,457	5,295	91,457	5,295	91,457

Table 14-32 shows the summary statistics comparison for the J1 lode. The gold, copper and silver composite means are higher than the block grade means which would be a normal expectation and validates the geological interpretation and the grade interpolation method including any top cutting.

Table 14-32 Comparison of Summary Statistics for Composites & Block Grades for the J1 Lode

Judd lode	Gold		Copper		Silver	
	Comp	Block	Comp	Block	Comp	Block
Mean	6.04	3.12	0.64	0.45	14.80	10.07
Median	0.92	1.07	0.19	0.31	4.48	6.48
Standard Deviation	25.47	6.45	1.19	0.47	31.77	11.79
Coeff of Variation	4.22	2.07	1.85	1.04	2.15	1.17
Minimum	0.005	0.01	0.0001	0	0.01	0.01
Maximum	400	189.41	19.42	5.84	300	175.14
Count	4,342	108,357	4,342	108,357	4,342	108,357

The conclusion from the comparisons of drillhole samples, the composites values and the block grades is that there appears to be no obvious issues with the grade interpolation for all elements.

Another grade interpolation check is to compare cumulative frequency curves for the composites and block grades for the different elements for the different lodes. Figure 14-76 shows the cumulative frequency comparison for uncut gold in the K1 lode and exhibits a very acceptable pattern. The comparison was completed for copper and silver and indicated no significant issues with the grade interpolation or post-modelling processing.

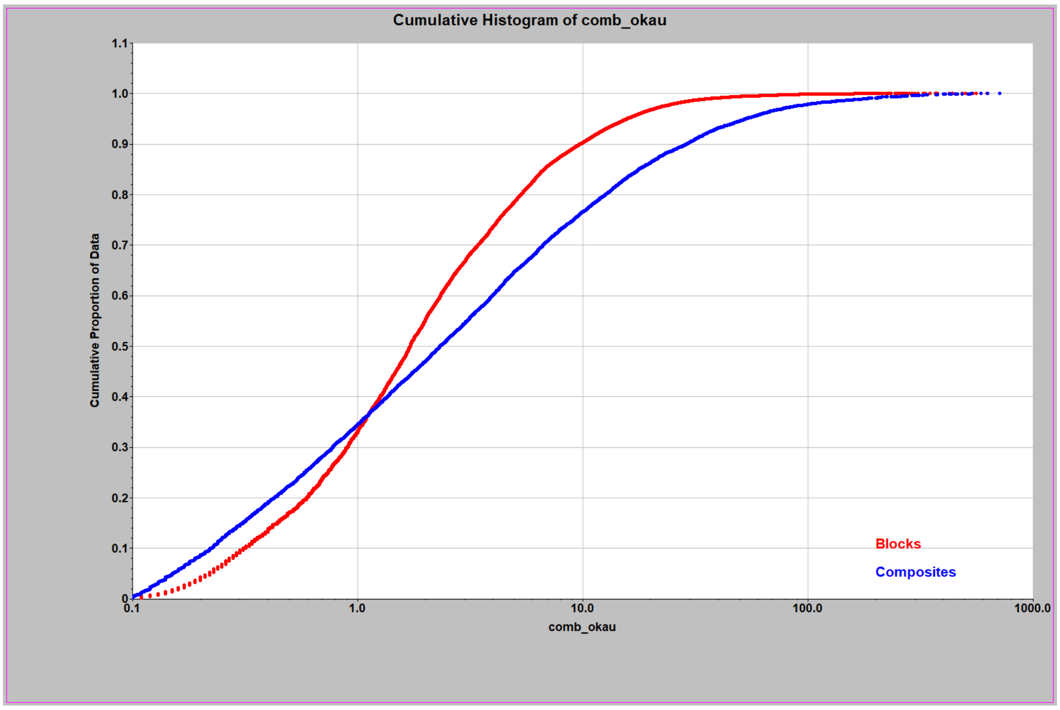


Figure 14-76 Gold Block Grade & Composite Cumulative Frequency Curves for the K1 Lode (H&SC)

Figure 14-77 shows the cumulative frequency curves for the gold composite and block grade data for K2. Similar curve patterns were produced for copper and silver.

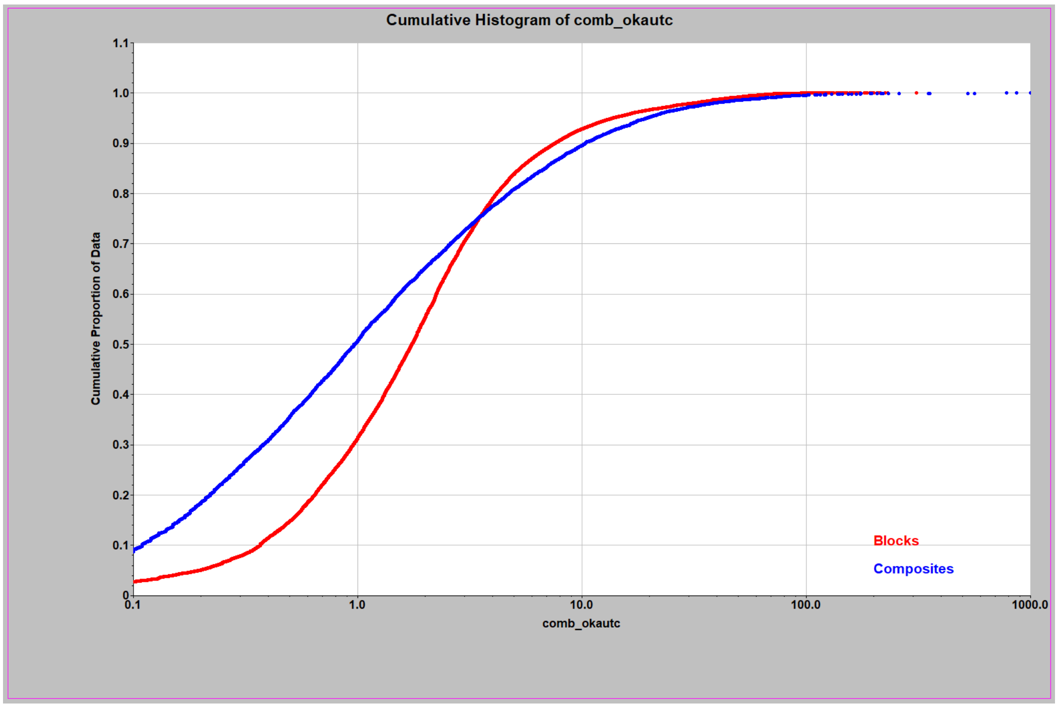


Figure 14-77 Gold Block Grade & Composite Cumulative Frequency Curves for the K2 Lode (H&SC)

Figure 14-78 shows the cumulative frequency curves for the gold composite and block grade data for the Kora Link lode. Similar curve patterns were produced for copper and silver.

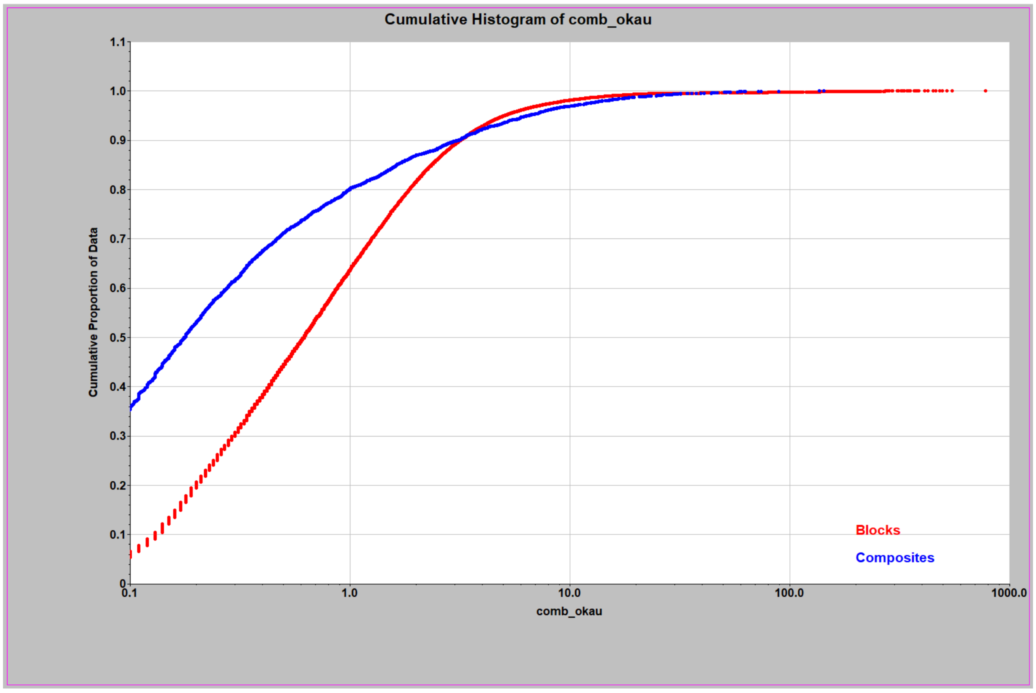


Figure 14-78 Gold Block Grade & Composite Cumulative Frequency Curves for the Kora Link Lode (H&SC)

Figure 14-79 shows the cumulative frequency curves for the gold composite and block grade data for J1 lode. Similar curve patterns were produced for copper and silver.

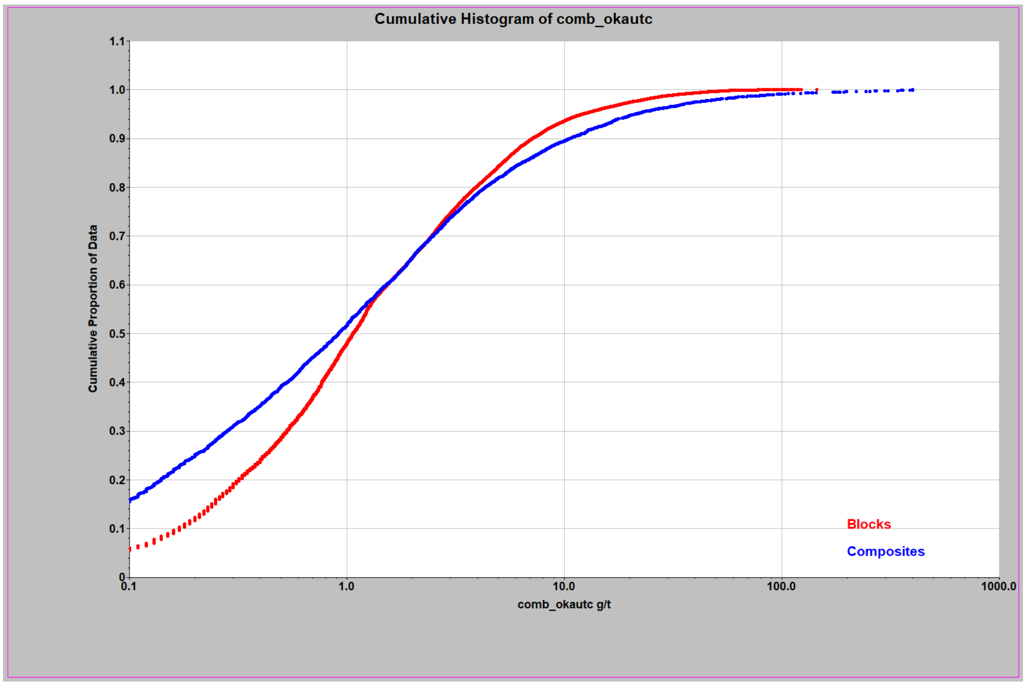


Figure 14-79 Gold Block Grade & Composite Cumulative Frequency Curves for the J1 Lode (H&SC)

The conclusion from the cumulative frequency curves is that there appears to be no obvious issues with the grade interpolation for all elements.

The 2018 work comprised a number of check models with the following conclusions:

- The use of an expanded wireframe to represent the mining widths appeared to be appropriate especially in respect to the production reconciliation (narrow wireframe vs broader wireframe).
- The relatively small block size does not seem to result in an over-statement of grades via over-smoothing.

- Using the combined drilling and face sampling data for grade interpolation is acceptable.

For 2020 rudimentary check Multiple Indicator Kriged ("MIK") models for the K1 and K2 lodes were completed by H&SC's Sydney office using a dynamic interpolation search method. The results for each lode from this model were reported for the OK model pass categories. The MIK K1 outcomes showed a very good match with the OK model estimates (<0.3% difference between gold equivalent ounces), whilst the K2 results were acceptable; the MIK estimates had just under 6% more gold equivalent ounces although the increase was associated with the more peripheral search pass and therefore of higher risk. A comparison of the results for the mined material showed very similar overall gold ounces for both K1 and K2, although the OK model had slightly more gold equivalent ounces from K1 which was offset by more ounces from the MIK model for K2. The check provides added confidence in the estimation results generated by the OK modelling. Whilst the MIK modelling method is a more sophisticated method than OK, it is generally better suited to open pit operations and requires expert knowledge when used for underground stope design. The MIK model with the dynamic interpolation was only intended as a check model in order to confirm that the OK approach with multiple search domains was an acceptable technique.

For this report a check OK model using composite data without the face sampling was completed by H&SC. The study only looked at a nominal 'general mine' area for Kora where there was detailed drilling and face sampling. The results confirmed the 2018 conclusions in that for K2 the tonnes and grade were similar with or without the face sampling but for K1 the gold, copper and silver grades were between 10-15% lower without the face sampling. The good reconciliation (see below) indicates that the face sampling is needed to replicate the mill feed grades especially for gold.

Reconciliation data was supplied by K92ML, comprising tonnes and grade of material processed by the mill from Kora and Judd for the end of September 2023, covering approximately five and a half years of mining. The mill processed material has included the K1, K2 and J1 development drives and stopes plus 'other' material derived from reprocessed material and small amounts of material from dilution from CAF mining operations as well as from mining of Irumafimpa and other mineralized structures.

The block model was flagged using the supplied development and stope wireframes for Kora and Judd on a centroid in/out basis. Blocks with a centroid inside the mineral wireframes were reported for a zero gold cut off using the combined gold data (most of the data was from the K1 lode). For Judd the data was reported from the top cutted combined gold attribute. This is the same technique as for previous resource estimates. Previous reporting had tested the use of a partial percent volume adjustment to the development and stope wireframes with a minor difference (not significant) compared to the centroid in/out basis. The partial percent volume adjustment method for the stopes and development was not an option for reporting mined material due to the supplied wireframes not being suitable for the technique.

Over the years it has become apparent that small volumes of 'other' material have been placed on the ROM for processing. The overall gold grade of the 'other' material was derived from previous estimates as being the grade of material needed to equate with the processed mill feed material (Table 14-33). The gold grade of this 'other' material is not actually known but K92ML confirm that the numbers in Table 14-33 are reasonable considering the likely sources of the material e.g. Irumafimpa.

Table 14-33 Details of Sources for 'Other' Material

Source	Tonnes	Au (g/t)	Cu (%)	Ag (g/t)
2017-2020 Period	52,291	8.6	0.40	8.0
2020_2023 Back Fill lifting with ore	65,000	2.8	0.35	4.0
2021-2023 Period MW Processed Unsurveyed self-stope, collapsing unsurveyed stopes	62,757	1.6	0.30	5.0
Total	**180,049**	**4.1**	**0.35**	**5.5**

(the use of significant figures does not imply accuracy)(minor rounding errors)

Reporting of the estimates from the H&SC model used the development and stope centroids and detailed approximately 1.37 Mt of lode material (including 0.16 Mt of waste) had been extracted for 509Kozs of gold (Table 14-34). The reported tonnage from the resource model is moderately less than the ROM figures which comprised 1.59 Mt for 539 kozs of gold. With the inclusion of the 'other' material the match-up for gold ounces between the new resource model and the mill feed is improved.

Table 14-34 Reported Depletion and Reconciliation with ROM Figures (Sept 2023)

	Tonnes	Au g/t	Cu %	Agtc g/t	Au ozs	Cu Tonnes	Agtc ozs
Mineral	1,206,132	13.13	0.78	13.2	509,357	9,360	511,978
Waste	160,152						
Total	**1,366,284**				**509,357**	**9,360**	
Other	180,049	4.06	0.35	5.5	23,495	625	31,901
Total	**1,546,333**	**10.72**	**0.65**	**10.9**	**532,851**	**9,984**	**543,879**
Mill Feed	1,585,358	10.6	0.60	9.6	538,984	9,010	487,686
Difference %	**-2.5**	**1.1**	**7.1**	**12.2**	**-1.2**	**9.8**	**10.3**

(the use of significant figures does not imply accuracy)(minor rounding errors)

The end result is that the centroid method for the new block model has slightly understated the mined tonnage by 2.5% with a slightly higher gold grade by 1.1%. This has resulted in a slight under-reporting of gold ounces from the new resource model by 1.2% as compared to the mill feed.

Whilst the reconciliation with the gold grade of the ROM material is good, a more disappointing result is achieved for copper and silver with an approximate 10% over-statement of grade for both metals. This might be due to the lack of a top cut for copper but in H&SC's experience any sensible top cut is not going to have a significant impact. A top cut of 15% Cu for K1 would impact 10 samples but only reduce the copper composite mean by 0.6% with a very minor change in CV. Applying the same top cut to K2 results in 25 samples being affected but results in the same 0.6% drop in composite mean. If the same top cut was applied to J1 only 1 composite would be affected.

K92ML has pointed out that as the 'other' tonnage includes material from Irumafimpa it is unlikely to have the 0.35% copper grade allocated to it. Therefore the tonnes of copper metal tonnes for the 'other' material would decrease reducing the copper grade discrepancy with the reconciliation to only 4% (if all copper tonnes were removed). The discrepancy with the silver is not explained but could be a function of the detection limits with the on-site laboratory.

It is worth reiterating that the partial percent volume adjustment method for reporting tonnes and grade from the new resource model is potentially more accurate but the supplied wireframes did not easily allow for this method to be used this time. The more accurate partial percent method has been shown in the past to be very slightly less than the centroid in/out method.

The results of the reconciliation would appear to justify that the combination of using both drillhole and face sampling data, limited use of top cuts, variography, composite length, geological interpretation and search parameters has removed/significantly reduced the smearing of the very high gold grades and any subsequent over-statement of the resource estimates.

The reconciliation outcomes are reasonably similar, if not improved, to the outcomes from October 2021, March 2020 and September 2018 and allow for a good level of confidence in the new resource estimates and the methodologies used to generate them.

14.8 Resource Classification

Classification of the Mineral Resources is derived from the search pass number associated with each block, which essentially is a function of the drillhole and face sample data point distribution. Additional considerations were included in the assessment of the classification, in particular the geological understanding and complexity of the deposit, variography, sample recovery, quality of the QAQC sampling and outcomes, density data, block model validation and reconciliation with the ROM material.

Table 14-35 contains details of the resource classification from the pass categories.

Table 14-35 Resource Classification Details

Pass Category	Resource Classification
1	Measured
2	Indicated
3	Inferred

Additional post-grade interpolation work for the block model included the use of defined shapes to clean up the distribution of Measured Resources and remove the 'spotted dog' effect. An additional defined shape for Indicated Resource was applied to the J1 mineral blocks.

Issues impacting on the resource classification are:

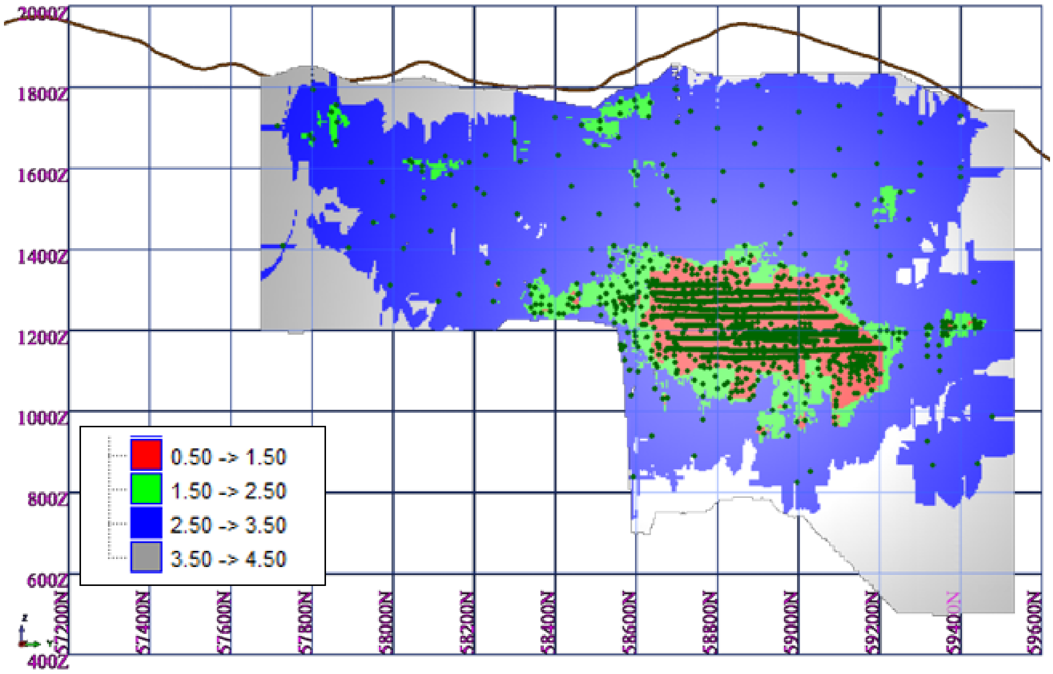
- The geology of the deposit and the style of mineralization: shear zone hosted gold mineralization is notorious for poor grade continuity. The ability to physically put a finger on any of the gold mineralization contacts for the K1 and K2 lodes is considered variable, which can lead to sub-optimal resource estimation if care is not exercised with the geological interpretation and grade interpolation. To counteract the complex grade distribution H&SC has fused a combination of composite length, geological interpretation, variography and search parameters (especially the minimum number of data points required) so as to minimise the possible over-statement of grade within the resource estimates. This appears to have been successfully completed based on the block model validation and reconciliation with mill feed numbers.
- The sampling methods: the bulk of the resource estimates have been generated from diamond drilling results which is generally considered the best sampling technique (assuming good core recoveries). However, a substantial amount of the high grade assays are from the face sampling which can be prone to variance associated with the actual sampling method e.g. not passing into background on both hangingwall and footwall and is considered a sub-optimal sampling method with respect to diamond drilling. Counter to these potential negatives is that K92ML have a good, documented face sampling procedure that attempts to minimise the risk with the actual sampling technique, and it is worth noting that the development and stoping associated with the face sampling appears to be reconcilable with the block model.

- The general drill hole spacing and hence data distribution is considered wide for a large part of the deposit. This impacts negatively on the variography, which in turn indicates that much closer spaced drilling is required for more confidence in the grade continuity, which in turn is reflected in the current resource classification, generally Inferred. The close spaced face sampling, diamond drilling and subsequent mining outcomes provides a high level of confidence in the measure of the gold grade continuity in the general mined areas, hence Measured Resource.
- Density data: there is a sufficient amount of data for density grade interpolation for the main lodes. OK was used to model the density sample data and is a more sophisticated technique that would be expected to result in greater accuracy for the local density block grades. However, it was revealed late on in the piece that a substantial proportion of density data was possibly suspect. The main suspect area was selected holes in the southern upper section of K1, K2 and J1, in an area where there was limited drilling and thus confidence in the Mineral Resource was low i.e. Inferred. Elsewhere, particularly in the general mine area there is a moderately high level of confidence in the density values which is reflected in the higher levels of resource classification. Sample selection for density measurement is at risk of a positive bias with samples of competent core preferentially selected. An assumption has been made that there are no significant cavities associated with the mineralization or that if they exist their impact is minimal.
- The QAQC procedures and outcomes: these are considered to be to industry standard and the QAQC outcomes impart a high level of confidence in the appropriateness of the sampling methods and the accuracy of the assays. For the 2018-2023 period QAQC has been completed on the face sampling with acceptable results. The pre-Sept 2018 face sampling had no QAQC data, but sample preparation and assaying had been completed on concurrent drilling samples, which indicated no significant issues with the CRM analysis. QAQC Data has been procured for the Highlands and Barrick drilling and indicates no systematic bias with the historic data. There are no discernible issues with the K92ML drilling and face sampling data.
- Core recoveries: the current recovery of >91 to 96% for the main lodes is reasonable but some of the initial drilling was a little low (around the 90% mark). However, there is no relationship between gold grade and recovery such that in conjunction with the QAQC data the confidence level in the gold grade of the samples is high.
- Grade continuity/variography: this was reasonably good for the detailed area of K1 mainly due to the face sampling backed up by relatively detailed drilling. Whilst grade continuity was not as extensive for K2 or J1 the amount of detailed data was good enough for reasonable variography outcomes with good confidence in the grade ranges which were supported by the visual review of grade continuity.
- Impact of top cuts: the main issues are with the K2 (and J1) lodes where extreme values have been demonstrated to show high grade smearing out with the general development area. In areas of K2 where this effect is abundant the classification of any Measured material has been downgraded to Indicated.
- Reconciliation with mill feed: this appears to be very good for gold (and reasonably good for copper and silver) with predicted block model ounces reported to within 1-2% of the mill feed numbers up to the end of September 2023. This has allowed for a reasonably high level of confidence in the gold content for material in the immediate vicinity of the development drives and mined stopes, hence Measured Resource. The copper and silver numbers are good enough to allow for the same resource classification as for the gold data.
- The resource classification works on the pass number allocated to the block grade interpolation value. Often this produces what is called a "spotted dog" effect whereby blocks of Pass 2 lie within a mass of Pass 1 blocks. This often happens on the margins of deposits where there is a natural lack of data points, due to the wireframe edges, available for grade interpolation. To clean up this scenario H&SC used defined shapes to better define Measured Resources for K1 and K2 and Measured and Indicated Resources for J1. The Measured Resources were squared off with the solid edges a function of the distance away from the development, whilst the redefinition of the Indicated Resources was based on a visually decided level of drillhole density.
- Validation of the block model indicated no issues with the grade interpolation or post-modelling processing.

- The general lack of data and certainty in the geological interpretation for the J3 and J1W lodes meant that they were classified as Inferred.

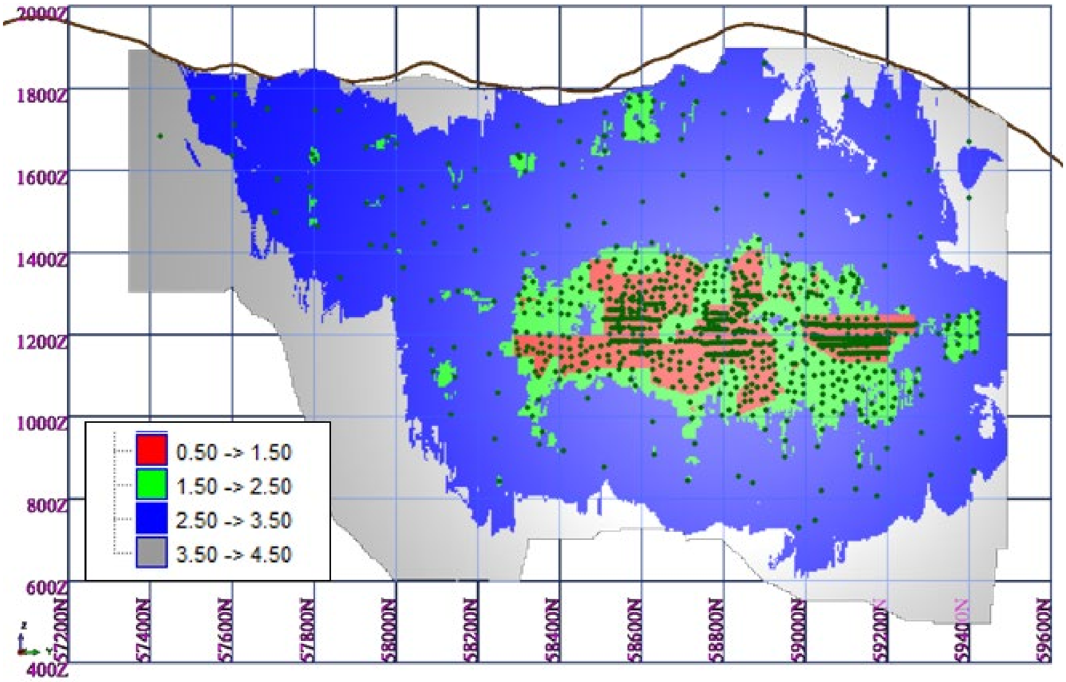
Figure 14-80 represents the distribution of Measured (red blocks), Indicated (green blocks) and Inferred Resources (blue blocks) for the K1 lode irrespective of cut-off grade. The gaps in the model and the grey-shaded areas are blocks with no interpolated grades due to a lack of data and conceivably represent areas for expanding the resource estimates via additional drilling.

The outcome of the 2023 modelling relative to the 2021 model has been the consolidation of the Measured and Indicated Resources i.e. replacement of mined material mainly on the upper side of the general mine area and the expansion of the Mineral Resource, mainly in the south, which was the main aim of the past one and half years of drilling.



(view : looking west)(green dots = drillhole and face sample pierce points)
(red =Measured, green = Indicated, blue = Inferred, grey = Exploration Potential)

Figure 14-80 Resource Classification for the K1 Lode (H&SC)



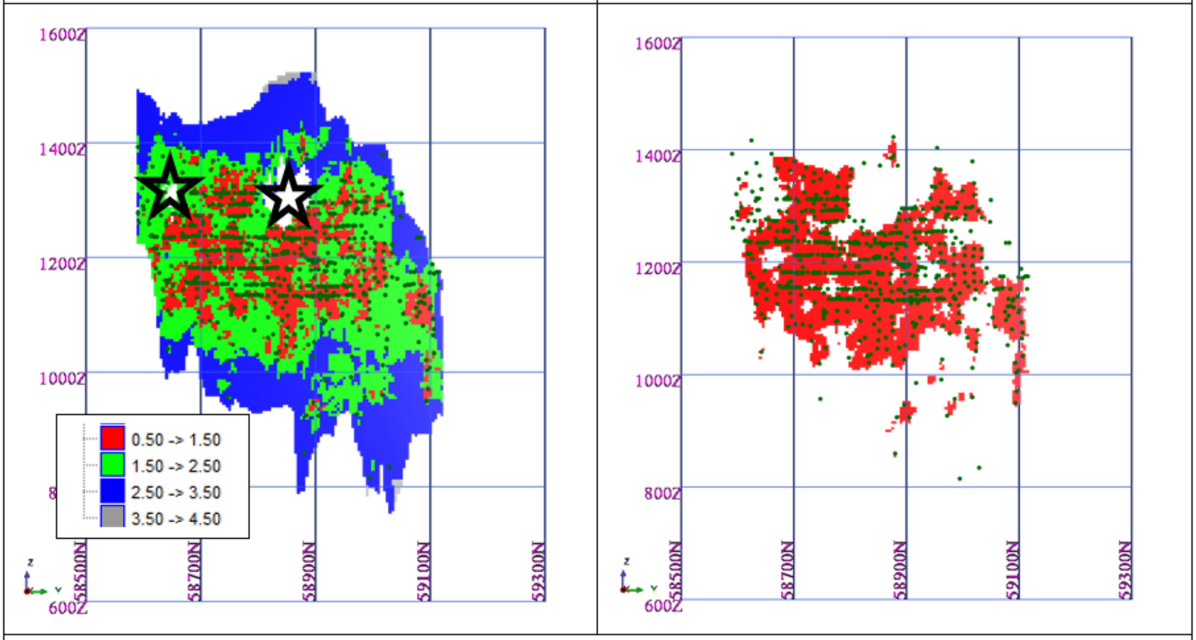
(view : looking west)(green dots = drillhole and face sample pierce points)

(red =Measured, green = Indicated, blue = Inferred, grey = Exploration Potential)

Figure 14-81 Resource Classification for the K2 Lode (H&SC)

Figure 14-82 represents the distribution of Measured, Indicated and Inferred Resources for the Kora link lode. There are two small gaps (black stars) in the model where it is believed that K1 and K2 butt against each other. The right-hand figure shows the Measured Resource and indicates a greater level of coherency that might be apparent in the left-hand figure.

Kora Link Resource Classification

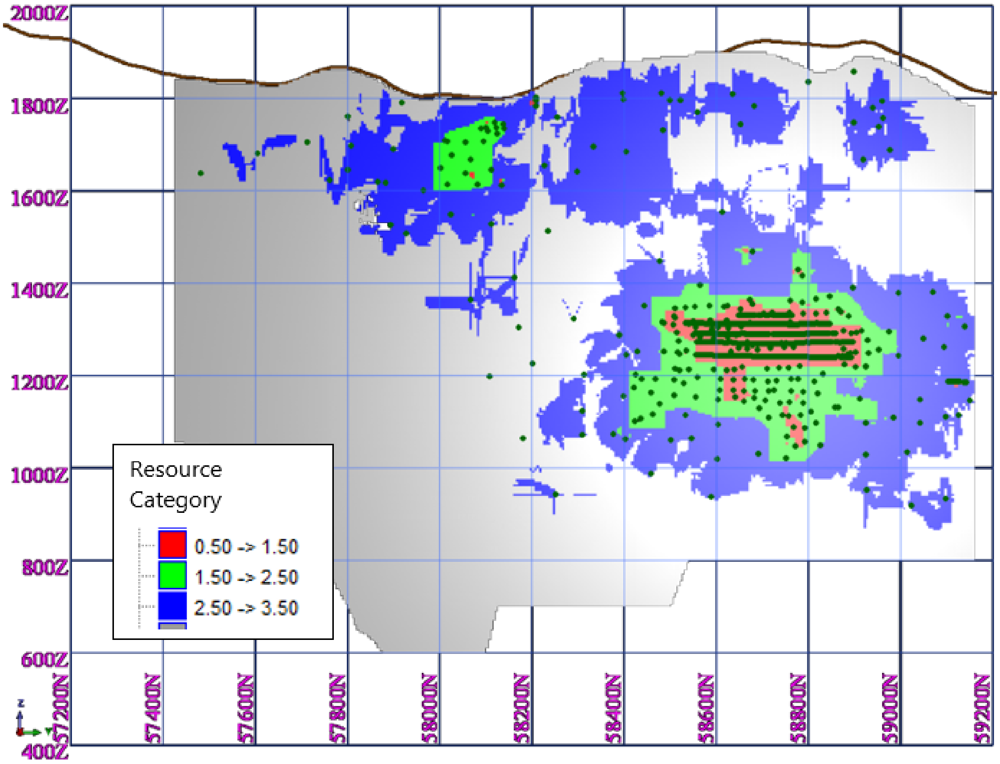


(view : looking west)(green dots = drillhole and face sample pierce points)

(red =Measured, green = Indicated, blue = Inferred, grey = Exploration Potential)

Figure 14-82 Resource Classification for the Kora Link Lode (H&SC)

Figure 14-83 represents the distribution of Measured, Indicated and Inferred Resources for the J1 lode. The gaps in the model and the grey shaded area are blocks with no interpolated grades due to a lack of data and conceivably represent areas for expanding the resource estimates via additional drilling. The relative narrow nature of the lode and the wide-spaced drilling has resulted in a lack of data at the hangingwall and footwall margins of the lode causing the classification to be Inferred. A defined shape based on proximity to the development drives was used to classify isolated blocks of Indicated Resource to Measured Resource. A defined shape was used to upgrade blocks from Inferred to Indicated Resources based on the visual observation of the drillhole density.



(view : looking west)(green dots = drillhole and face sample pierce points)

(red =Measured, green = Indicated, blue = Inferred, grey = Exploration Potential)

Figure 14-83 Resource Classification for the J1 Lode (H&SC)

14.9 Discussion Of Factors For The Mineral Resources

- Ordinary Kriging as a valid modelling method. Coefficients of variation for the gold composites are relatively low for this type of deposit, around the 2-2.5 mark, which indicates limited skewed data and/or extreme values. This in combination with visual reviews of the composite grade distribution showing well-structured data, means the method is acceptable for grade interpolation. This is in preference to a more sophisticated and time-consuming modelling method like Multiple Indicator Kriging ("MIK"), at this stage. The success of the method does rely on the variography, geological interpretation, block size and search ellipse parameters. With increased levels of mining and drilling there have been more complexities associated with the distribution of mineralization and it may be appropriate to consider running MIK models in the future.
- Geological interpretation completed by H&SC. The wireframing method used by H&SC is based on encompassing the gold mineralization as discrete gold lodes with a nominal cut-off grade of 0.1-0.2 g/t Au in conjunction with copper and silver grades and geological sense. This helps to avoid over-constraining the data i.e. the low threshold reduces the effect of any introduced Conditional Bias to the composite data. The sampling has been to geological control which also adds some control to the interpretation of the gold mineralization. The wireframes have also been based in part on the current mining method which can involve some dilution with generally lower gold grade material and possibly a small amount of barren waste rock. In more peripheral areas there is

sometimes less certainty in the interpretation due to multiple mineral drill intercepts and the wide drillhole spacing. The poddy nature to the gold mineralization for K2 and possibly J1 might be considered indicative of the multiple intercept/wide drill spacing issue.

- A further potential geological complication is the occurrence of faulting in and around the mineral lodes, particularly for Kora. A Kora Fault has been delineated, mainly for geotechnical purposes, but K92ML are uncertain as yet if any faulting has a major impact on the spatial location of the mineralization. There are circumstantial indications of both cross cutting and oblique angled faulting, mainly based on doglegs associated with the mineral wireframes and the gold and copper block grade distribution within the lodes.
- Wide drillhole spacing. Large parts of the mineral lodes have been interpreted from wide spaced drilling, generally in the order of 100 m. In H&SC's experience modelling of gold composite data with such wide spacings is relatively high risk, hence the Inferred classification for a substantial part of the deposit. As a result, any follow up drilling could lead to substantial changes, most likely reductions, in the size and spatial geometry of the resource estimates.
- Incorporation of the face sampling data into the grade interpolation. Summary statistics for the K1 lode have shown a much higher mean for the face sampling (with a relatively low CV) compared to the drillhole values, which is attributable to the high grade zone encountered in the current mining. Check modelling of just the drillhole data without the face sampling has failed to fully pick up the intensity of gold in the high grade K1 zone. Therefore, inclusion of the face sampling data in the grade interpolation is considered justified. Summary statistics for the K2 lode indicates reasonably similar data populations for drilling and face sampling with the latter having a lower coefficient of variation. Therefore, inclusion of the face sampling data in the grade interpolation is considered justified. In general, the summary statistics support the inclusion of the face sampling data with the drilling data with the vast majority of the face sampling included in the wireframes en masse. In addition, modelling just the drillhole data could significantly under-report the remaining resource estimates for the K1 lode.
- Density data. There is enough density sample data in and around the general mine area for interpolating density block grades, using OK, for all the main lodes (including Kora Link and J2). OK is a reasonably sophisticated technique and is used in preference to other less sophisticated industry standard methods of Inverse Distance Squared and Nearest Neighbour. Previous estimates for Kora have used average default values for the different lodes which is probably globally accurate but on a local scale having all the blocks for one lode with the exact same density is probably unrealistic. OK allows for interaction between samples to generate more accurate localised density block grades. In peripheral areas the limited drilling means the unfilled blocks from the grade interpolation within the lode wireframes require the insertion of a range of default density values. It has emerged that there are suspicious density data measurements i.e. very low values associated with some of the surface exploration holes which need checking. The impact of these suspect values is considered modest with the main affected area, the southern upper section of K1, K2 and J1, generally classified as Inferred. More appropriate default density values have replaced any modelled grades in this area that have a density value <2.5 t/m^3.
- The relatively small block size. Often a small block size can lead to over-smoothing of grades and thus an over-statement of grade for the resource, especially associated with a deposit of this type. However, in this case the block size is a function of the relatively close spaced drilling and face sampling associated with the K1 and K2 lodes. In 2018 a check model was completed using a more typical 1 m by 15 m by 15 m block size for the wider-spaced drilling of the K2 lode and there seemed to be no significant difference in the reported estimation results. In 2023 an unconstrained check model for Kora with a 2 m by 10 m by 10 m block size again confirmed no significant difference in tonnes, gold grade and gold ounces for the three Kora lodes relative to the smaller block size model.
- The use of top cuts for gold. This has provoked considerable discussion with K92ML. Limited top cutting was applied to extreme high grade gold composites for the K2 and Judd lodes. Whilst the use of top cutting is regarded as standard industry practice, H&SC considers that it is often used rather arbitrarily with no sound geological or statistical basis. H&SC is generally reluctant to apply top cuts preferring to control any high grade samples by a combination of geological interpretation, composite length, variography and search ellipse parameters. However, very extreme values with a

strong demonstrable negative impact on the veracity of likely block grades were top cutted i.e. for the K2 and Judd lodes. If a top cut is required the level of cutting is always hotly debated but factors that must be considered are the resultant composite mean and coefficient of variation, a cumulative frequency plot with breaks in continuity, and perhaps the most important the spatial location of the extreme grades. K92ML has been an advocate for very high top cuts which H&SC has accepted but wishes to point out that maintaining too high a cut off will often result in subsequent disappointment with follow up infill drilling especially in areas of wide spaced drilling and low sample numbers. At Kora it is noted that the reconciliation outcomes strongly suggest that gold top cuts are not needed for the general mined area. However, it is important to point out that while a substantial part of K1 has been mined, it has included extreme grades and the impact on mining has been observed, whereas for K2 no mining of extreme grades has been undertaken and so no measure of the impact of extreme grades has been assessed.

- Minimum number of data. H&SC has kept the minimum number of data for the Pass 3 grade interpolation search relatively high at 6. In H&SC's experience using a lower number of minimum data invites an increase in risk to the interpolated grades particular at the margin of the lodes or in areas of wide drillhole spacing.
- Variography: This provides a measure of the grade continuity of the composites and the weighting required when undertaking block grade interpolation. Good variography is often dependent on close spaced drilling and to this end sub-zones of close spaced drill data for K1, K2, Kora Link and J1 lodes were used in the variogram analysis. This yielded gold continuity ranges of 30-50 m for K1 and 10-20 m for the other lodes.
- Reconciliation. Considering the geological complexity and localised high grades, being within 1 to 2% of the ROM ounces is considered a good result. However, linking the confidence for the resource estimates to reconciliation is significantly dependant on the details of tonnes and grade of the 'other' material provided by K92ML and the geological interpretation of the mineral lode in conjunction with the current mining method. A significant variation in the current numbers used for reconciliation may undermine some of the confidence in the resource estimates, particularly on account of the generally small volumes being considered as a proportion of the overall MRE.

The key to the confidence of the resource estimates is the apparent good reconciliation of the block model with the mill feed in an area of very high gold grades. This would strongly support the methodologies used for the resource modelling, in particular the geological interpretation, the composite interval, the apparent lack of need for top cutting, the search parameters and the relatively small block size.

The Qualified Person is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the potential development of the MRE.

14.10 Mineral Resource Estimates

Reporting of the new MRE for the Kora and Judd deposits is included as Table 14-36 for a 3 g/t gold equivalent cut off respectively. The 3 g/t AuEq cut off was supplied by K92ML and is based on mine design studies incorporating previous mining. The constraints for the K1 and Kora Link lodes are uncut gold grade for block centroids inside the K1 and Kora Link mineral wireframes with mining depletion removed. The constraints for the K2 lode are cut gold grade (1,000 g/t) for block centroids inside the K2 mineral wireframe with mining depletion removed. The same wireframe constraints are used for reporting the MRE for the Judd lodes but for cut gold grade data of 400 g/t.

The Mineral Resources reported in this section have been classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves.

Table 14-36 Mineral Resources for the Kora & Judd Deposits (3g/t Au equivalent cut-off)

	Tonnes	Gold		Silver		Copper		AuEq	
	Mt	g/t	Moz	g/t	Moz	%	kt	g/t	Moz
Kora									
Measured	3.7	8.74	1.0	20.5	2.5	1.21	45.0	10.96	1.3
Indicated	3.1	6.99	0.7	21.9	2.2	1.31	41.3	9.40	1.0
Total M&I	6.9	7.94	1.8	21.1	4.7	1.25	86.2	10.24	2.3
Inferred	14.3	5.60	2.6	28.7	13.2	1.62	231.2	8.60	3.9
Judd									
Measured	0.4	9.05	0.12	19.0	0.25	0.80	3.2	10.58	0.14
Indicated	0.8	6.37	0.17	15.6	0.42	0.73	6.2	7.76	0.21
Total M&I	1.2	7.24	0.29	16.7	0.67	0.75	9.4	8.68	0.35
Inferred	2.3	6.27	0.45	15.8	1.15	0.76	17.2	7.72	0.56
Kora and Judd									
Measured	4.1	8.77	1.2	20.4	2.7	1.17	48.2	10.92	1.5
Indicated	4.0	6.86	0.9	20.6	2.6	1.19	47.4	9.05	1.2
Total M&I	8.1	7.83	2.0	20.5	5.3	1.18	95.6	10.00	2.6
Inferred	16.5	5.69	3.0	27.0	14.3	1.50	248.3	8.48	4.5

(minor rounding errors)

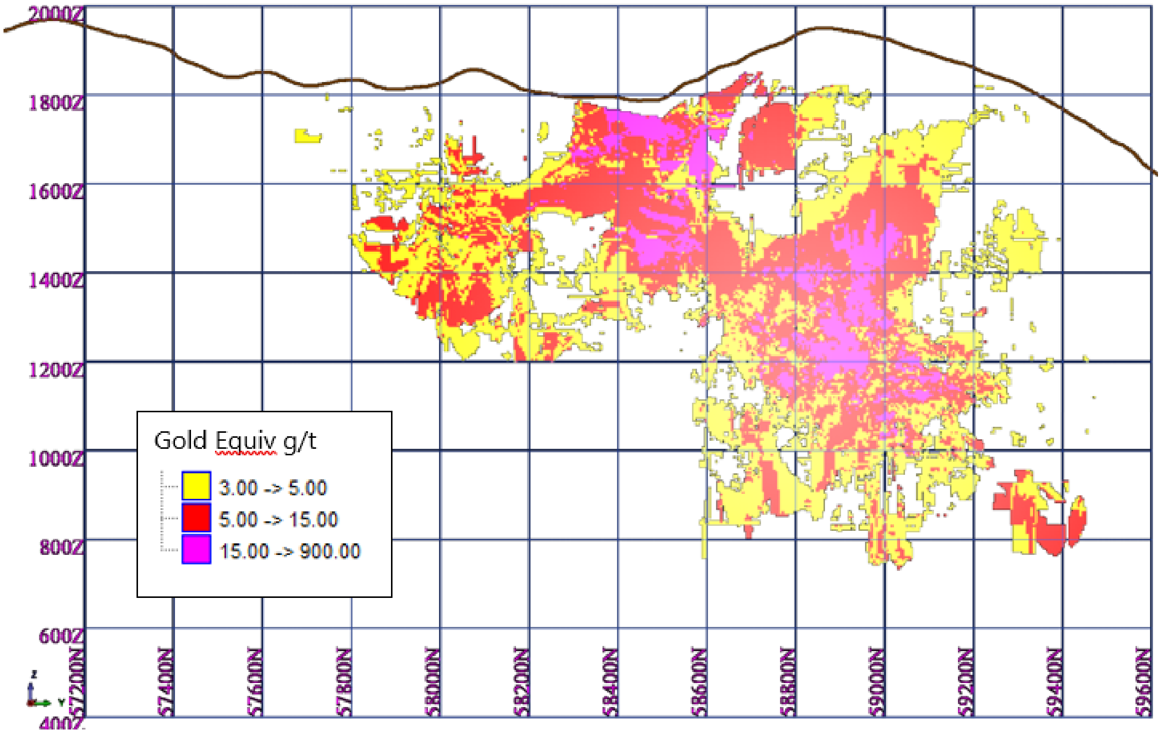
K92ML requested reporting of a gold equivalent g/t (AuEq) to include copper and silver credits using the formula:

AuEq = Au g/t + (Cu %*1.6481) + (Ag g/t*0.0114)

Assumptions include:

- Gold prices of USD $1,700/oz: Silver USD $22.5/oz; Copper USD $4.0/lb.
- Recoveries relative to gold of 93% for copper and 80% for silver.

Figure 14-84 shows the gold equivalent block grade distribution for the K1 Mineral Resources at a 3 g/t AuEq cut off (includes depletion). The brown line represents the ground surface.

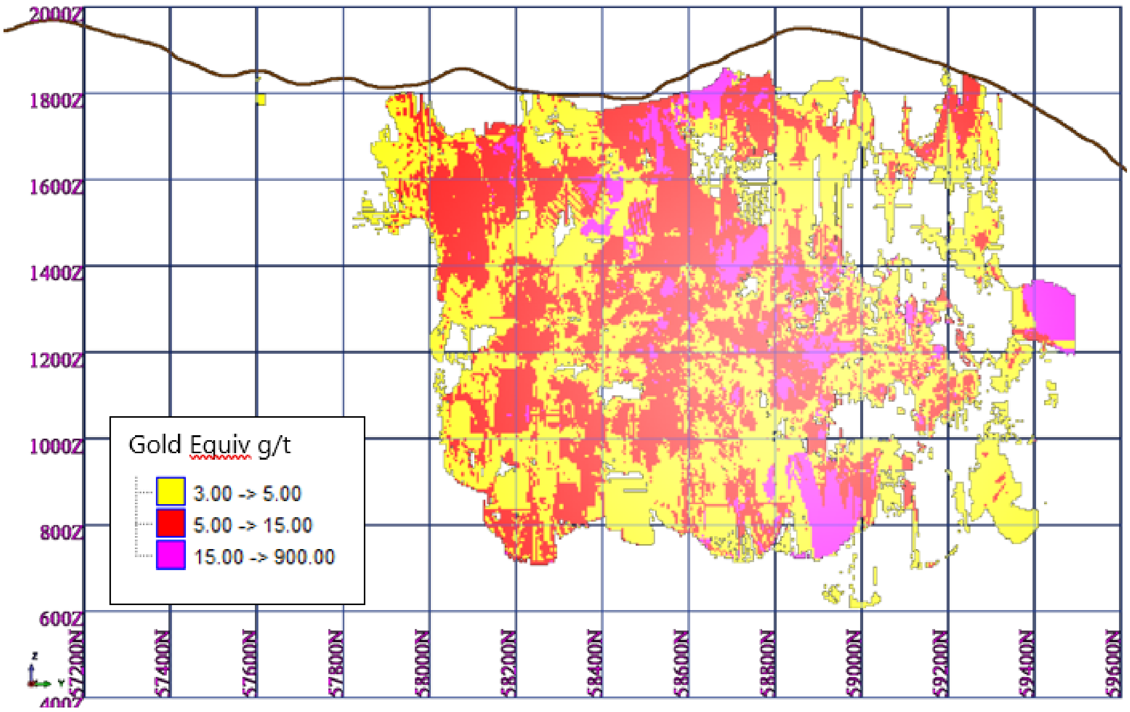


(view : looking west)

Figure 14-84 K1 Lode Mineral Resources Gold Equivalent Long Section 3g/t AuEq Cut Off (H&SC)

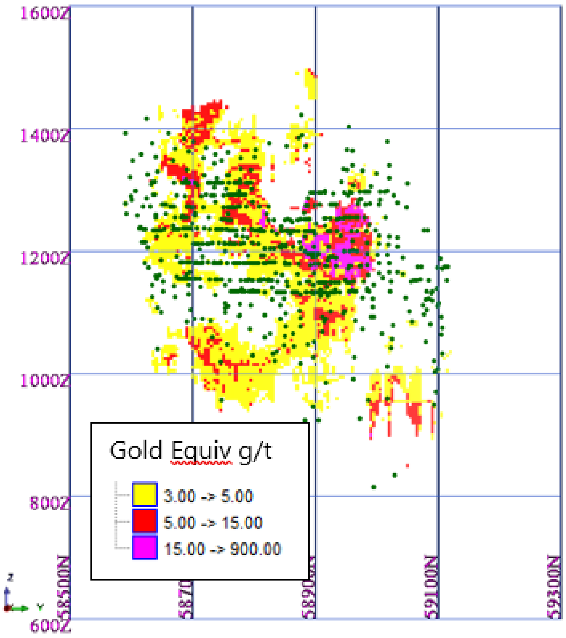
Figure 14-85 shows the gold equivalent block grade distribution for the K2 Mineral Resources at a 3 g/t AuEq cut off (includes depletion).

Figure 14-86 shows the gold equivalent block grade distribution for the Kora Link Mineral Resources at a 3 g/t gold-equivalent cut off (includes depletion).



(view : looking west)

Figure 14-85 K2 Lode Mineral Resources Gold Equivalent Long Section 3g/t AuEq Cut Off (H&SC)

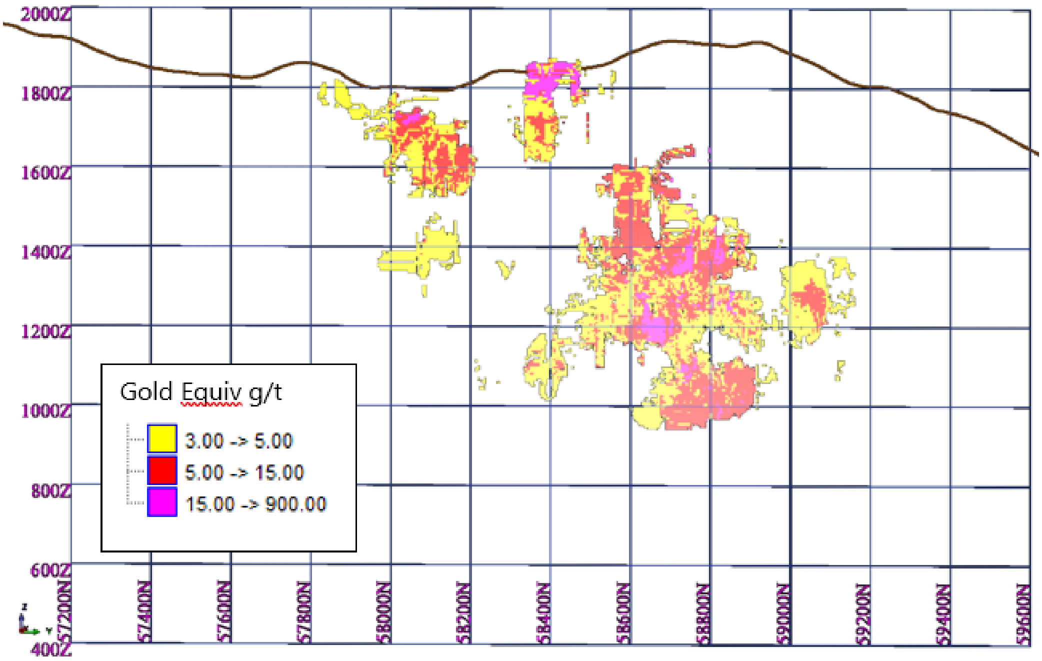


(view : looking west)

Figure 14-86 Kora Link Lode Mineral Resources Gold Equivalent Long Section 3g/t AuEq Cut Off (H&SC)

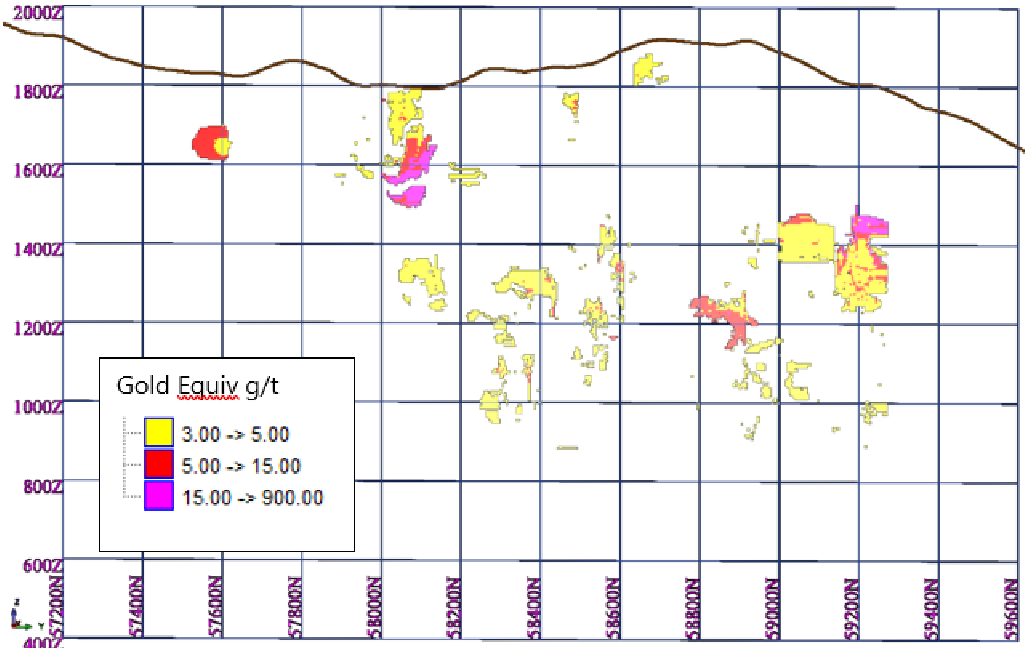
Figure 14-87 shows the gold equivalent block grade distribution for the J1 Mineral Resources at a 3 g/t gold equivalent cut off (includes depletion).

Figure 14-88 shows the gold equivalent block grade distribution for the J2 Mineral Resources at a 3 g/t gold equivalent cut off (includes depletion).



(view : looking west)

Figure 14-87 J1 Lode Mineral Resources Gold Equivalent Long Section 3g/t AuEq Cut Off (H&SC)



(view : looking west)

Figure 14-88 J2 Lode Mineral Resources Gold Equivalent Long Section 3 g/t AuEq Cut Off (H&SC)

To complete the resource reporting K92ML requested that the global MRE be reported for a range of gold equivalent cut off grades (Table 14-37). The grey band highlights the reported Mineral Resources.

Table 14-37 Global Mineral Resources for a Range of Gold Equivalent Cut Off Grades

Total Measured & Indicated									
AuEq cut off	Mt	Au g/t	Cu %	Ag g/t	AuEq g/t	Au Mozs	Cu kt	Ag Mozs	AuEq Mozs
0.25	15.8	4.56	0.75	13.5	5.96	2.31	118.9	6.9	3.02
0.50	15.0	4.78	0.79	14.1	6.24	2.31	118.2	6.8	3.01
1.00	13.4	5.32	0.86	15.4	6.92	2.28	115.5	6.6	2.97
1.25	12.5	5.65	0.91	16.1	7.33	2.27	113.5	6.5	2.94
1.50	11.7	5.98	0.95	16.8	7.74	2.24	111.3	6.3	2.90
1.75	10.9	6.31	1.00	17.5	8.15	2.22	109.0	6.1	2.87
2.00	10.3	6.63	1.03	18.1	8.54	2.20	106.6	6.0	2.83
2.25	9.7	6.94	1.07	18.7	8.92	2.17	104.2	5.8	2.79
2.50	9.2	7.25	1.11	19.3	9.29	2.15	101.8	5.7	2.75
2.75	8.7	7.55	1.14	19.8	9.66	2.12	99.5	5.6	2.71
3.00	8.3	7.85	1.17	20.4	10.02	2.09	97.1	5.4	2.67
3.50	7.4	8.50	1.23	21.4	10.78	2.04	91.8	5.1	2.58
4.00	6.7	9.19	1.29	22.5	11.57	1.98	86.3	4.8	2.49
5.00	5.4	10.68	1.39	24.4	13.26	1.86	75.4	4.2	2.31
7.50	3.3	14.81	1.58	28.4	17.74	1.58	52.5	3.0	1.90
10.00	2.2	19.47	1.64	31.0	22.53	1.36	35.8	2.2	1.58

Inferred AuEq cut off	Mt	Au g/t	Cu %	Ag g/t	AuEq g/t	Au Mozs	Cu kt	Ag Mozs	AuEq Mozs
0.25	35.9	3.02	0.88	17.1	4.67	3.49	315.6	19.7	5.39
0.50	33.8	3.20	0.93	17.9	4.93	3.47	314.0	19.5	5.36
1.00	29.6	3.59	1.04	19.7	5.53	3.42	307.3	18.7	5.26
1.25	27.5	3.82	1.10	20.6	5.86	3.37	302.3	18.2	5.19
1.50	25.6	4.05	1.16	21.5	6.20	3.33	296.4	17.7	5.10
1.75	23.8	4.28	1.22	22.4	6.55	3.28	289.6	17.1	5.01
2.00	22.2	4.52	1.27	23.3	6.89	3.22	282.4	16.6	4.91
2.25	20.5	4.80	1.34	24.3	7.28	3.16	273.6	16.0	4.79
2.50	19.0	5.08	1.40	25.3	7.67	3.10	264.7	15.5	4.68
2.75	17.5	5.38	1.46	26.3	8.08	3.03	255.5	14.8	4.56
3.00	16.4	5.66	1.51	27.1	8.46	2.98	246.9	14.2	4.45
3.50	14.1	6.29	1.61	28.6	9.28	2.86	228.3	13.0	4.22
4.00	12.4	6.92	1.70	30.0	10.07	2.76	210.3	11.9	4.01
5.00	9.4	8.43	1.86	32.4	11.87	2.54	174.4	9.7	3.57
7.50	5.0	13.01	2.11	34.4	16.88	2.10	105.8	5.5	2.72
10.00	3.0	18.55	2.11	34.3	22.42	1.79	63.5	3.3	2.17

Table 14-38 provides a comparison of the Mineral Resources for the current 2023 model with the published Mineral Resources in 2022. It should be noted that a gold equivalent cut-off grade (3 g/t) was used for the 2023 reporting compared to a gold cut-off grade in 2022 (1.75 g/t). The comparisons are acceptable indicating a modest increase in the size of the Measured and Indicated (~6.5%) with a minor increase in gold grade (~1.5%) and an ~8% increase in the gold equivalent grade. There is a more significant increase in the copper grade (~30%) for Measured and Indicated which is believed to be a reflection of additional infill drilling for the more copper-rich K2 lode and the expansion of the Mineral Resources into the southern more copper-rich zone.

Table 14-38 Comparison of Mineral Resources 2021/2 & 2023

Measured & Indicated										
Cut off	Model	Mt	Au g/t	Cu %	Ag g/t	AuEq g/t	Au Mozs	Cu kt	Ag Mozs	AuEq Mozs
3 g/t AuEq	2023	8.1	7.83	1.18	20.5	10.00	2.0	95.6	5.3	2.60
1.75 g/t Au	2021/2	7.6	7.72	0.91	18.3	9.29	1.88	69.2	4.5	2.26
	Diff %	6.6	1.4	29.7	12.0	7.6	6.4	38.2	17.8	15.0
Inferred										
3 g/t AuEq	2023	16.5	5.69	1.50	27.0	8.48	3.0	248.3	14.3	4.5
1.75 g/t Au	2021/2	9.1	6.8	1.32	25.5	9.05	1.99	119.9	7.45	2.65
	Diff %	81.3	-16.3	13.6	5.9	-6.3	50.8	107.1	91.9	69.8

(minor rounding errors)

Figure 14-89 contains the above data as a set of grade tonnage curves for the total Measured and Indicated Resources.

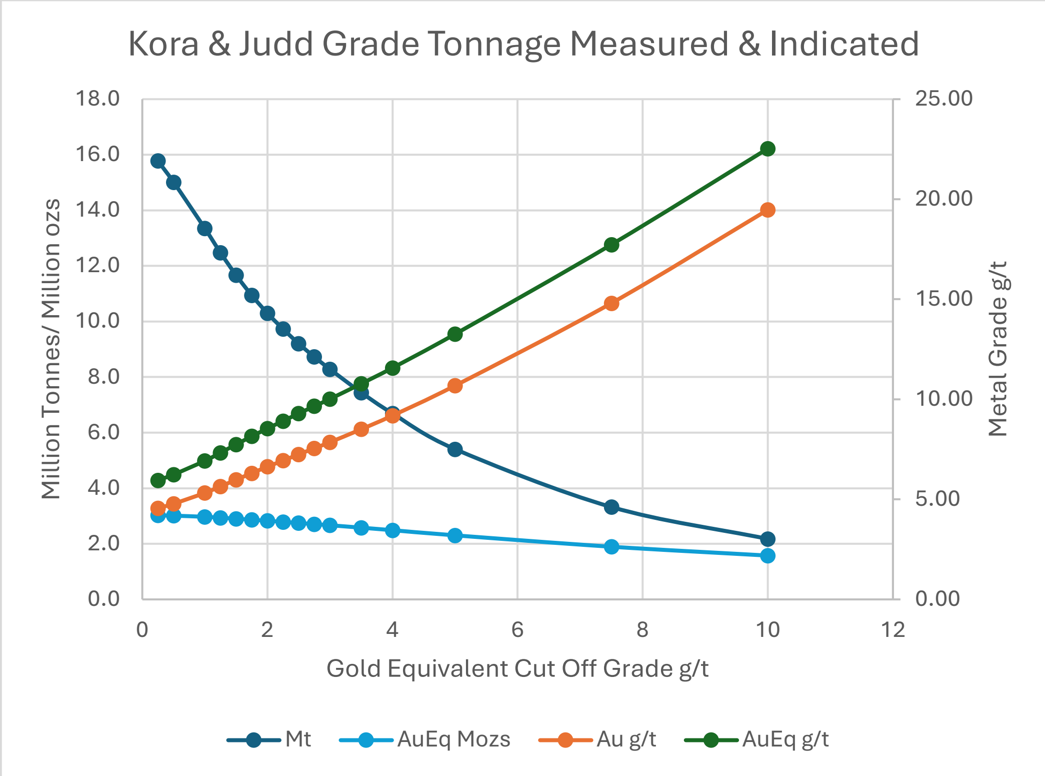


Figure 14-89 Kora & Judd Grade Tonnage Curves for Total Measured and Indicated Resources (H&SC)

Figure 14-90 contains the above data as a set of grade tonnage curves for the total Inferred Resources. The main item of note is the smoother curves for the metal grades compared to the 2021 curves which reflect an improved grade interpolation technique helped by increased amounts of drilling data.

Figure 14-90 Kora & Judd Grade Tonnage Curves for Total Inferred Resources (H&SC)

For the purposes of announcing the new global MRE (Table 14-39) a 3 g/t gold equivalent cut off was used with an effective date of September 12th, 2023.

Table 14-39 Kora and Judd Global Mineral Resources for a 3g/t AuEq Cut Off

	Tonnes	Gold		Silver		Copper		AuEq	
	Mt	g/t	Mozs	g/t	Mozs	%	kt	g/t	Mozs
Kora and Judd									
Measured	4.1	8.77	1.2	20.4	2.7	1.17	48.2	10.92	1.5
Indicated	4.0	6.86	0.9	20.6	2.6	1.19	47.4	9.05	1.2
Total M&I	**8.1**	**7.83**	**2.0**	**20.5**	**5.3**	**1.18**	**95.6**	**10.00**	**2.6**
Inferred	16.5	5.69	3.0	27.0	14.3	1.50	248.3	8.48	4.5

(minor rounding errors)

15. MINERAL RESERVE ESTIMATES

The Mineral Reserve estimate outlined in the DFS was prepared by Daniel Donald FAusIMM MSME of Entech, in accordance with the classification criteria set out in the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves prepared by the CIM Standing Committee on Reserve Definitions. Daniel Donald is an independent consultant of the Company and is a Qualified Persons as defined by NI 43-101.

The total Mineral Reserve for the Kainantu Project is shown in Table 1-2. The Mineral Reserve estimate is based on the Global Kora and Judd Mineral Resource estimate (September 12, 2023 effective date - refer to Table 1-3), net of post-resource mining depletion from September 12, 2023 to December 31, 2023, of 183,768 tonnes at 8.1 g/t Au, 0.9 % Cu and 15 g/t Ag.

Table 15-1 Kainantu Mineral Reserve Statement (Effective Date 1 January 2024)

	Tonnes	Gold		Silver		Copper		AuEq	
	Mt	g/t	moz	g/t	moz	%	kt	g/t	moz
Kora									
Proven	2.95	7.41	0.70	19.48	1.85	1.06	31.31	9.39	0.89
Probable	2.52	5.65	0.46	19.23	1.56	1.03	25.89	7.58	0.61
Total P&P	5.47	6.60	1.16	19.37	3.41	1.05	57.20	8.55	1.50
Judd									
Proven	0.24	8.26	0.06	16.53	0.13	0.56	1.33	9.40	0.07
Probable	0.47	6.48	0.10	13.15	0.20	0.52	2.45	7.50	0.11
Total P&P	0.71	7.08	0.16	14.28	0.32	0.54	3.79	8.14	0.18
Kora and Judd									
Proven	3.19	7.47	0.77	19.27	1.98	1.02	32.64	9.39	0.96
Probable	2.99	5.78	0.56	18.28	1.75	0.95	28.34	7.57	0.73
Total P&P	6.18	6.66	1.32	18.79	3.73	0.99	60.98	8.51	1.69

- This estimate is based on underground mine design work undertaken by Entech Pty Ltd. The estimate includes modifications to account for un-mineable material, dilution, and inferred metal within the mining shapes (any contained inferred material was set to waste grade).
- The long-term metal prices used for calculating the financial analysis are USD $1,900/oz gold, USD $4.50/lb Copper, USD $25/oz Silver.
- Gold Equivalents are calculated as AuEq = Au g/t + Cu % *1.62404 + Ag g/t*0.01316, based on commodity pricing. Metal payabilities and recoveries are not incorporated into this formula.
- A minimum mining width of 3.0 m has been applied for stoping, inclusive of a 1.0 m dilution skin at contained Mineral Resource grade.
- In addition to the 1.0 m dilution skin, dilution of 5% has been added for Avoca mined stopes and 2.5% for long hole stoping with paste fill. Where a stope is within 5.0m proximity of the HW or FW of the fault gouge, an additional 1.0m of dilution was added at a grade averaging 1.42g/t AuEq. This results in a total average dilution of 27.8%.
- Mining recoveries of 90% have been applied to Avoca mined stopes, and 95% for long hole stoping with paste fill.
- A cut-off grade of 3.5 g/t AuEq was used to define stoping blocks. Stope shapes with uneconomic development were excluded. The cut-off grade takes into account site operating costs, G&A costs, sustaining capital costs and relevant processing and revenue inputs.
- Measured Mineral Resources were used to report Proven Mineral Reserves.
- Indicated Mineral Resources were used to report Probable Mineral Reserves. No Measured Mineral Resources were used to report Probable Mineral Reserves.
- Tonnage and grade estimates include dilution and recovery allowance.
- The Mineral Reserves reported are not added to Mineral Resources.

- The mine plan assumes mining of Mineral Reserve material only and was shown to be economically viable with a reasonable degree of margin to buffer against unfavourable input movements. It should, however, be noted that sufficient negative movements in the assumed gold prices and/or exchange rates, metallurgical recoveries and/or costs could potentially render portions or the whole of the Mineral Reserves mine plan uneconomic.

16. MINING METHODS

16.1 Introduction

Entech was engaged by K92ML to complete a Definitive Feasibility Study (DFS) on underground mining of the Kainantu Gold Mine. The Kainantu Gold Mine is a high-grade gold, copper, silver mine with gold and copper being the primary revenue generating elements. Ore grades and metal are generally reported as a gold equivalent (AuEq).

The underground mine has been operated by K92ML since October 2016 and has a current processing throughput of 600,000 tpa as per the Stage 2A Expansion. The 2024 DFS targets an expansion to a peak processing rate of 1,200,000 tpa with a 7-year mine life. The DFS assumes a start date of the life of mine schedule of 1 January 2024.

16.2 Geotechnical

No additional core was geotechnically logged for the Kora Underground in addition to the 4,134 m of drill core that was geotechnically logged from core photographs for the Kora and Judd underground mining assessment during Q1 and Q3 2021.

Due to covid restrictions during 2021 and resourcing availability in-country, structural orientation measurement data was unable to be collected by Entech for this study. However, defect orientation measurements had been taken from scanline mapping from underground development by K92ML staff and provided to Entech. No new useable data has been collected and made available since this time.

Rock property testing specific to the orebodies was unable to be sampled and sent for testing due covid restrictions and resourcing availability in country, and available drill holes for sampling (2021). No rock property testwork has been completed since this time. Entech tested sensitivity to rock strength in lieu of available data for elements of this study to ensure sufficiency of design for this study. Entech considers that intact rock property test results will remain a significant information gap for the project and should be addressed as soon as practical.

The drill core logging undertaken and analysed by Entech in 2021 forms the basis of this study and allows for the characterisation of the rock mass, assessment of stable stoping span predictions and estimates of dilution factors.

There has been no geotechnical drilling, logging or sampling since the completion of the 2021 Feasibility Study to improve confidence in assumptions made in this study and to confirm the rock mass characteristics of all mining areas for the life of mine design and infrastructure.

Twelve stope designs (K1, K2 and Judd) with corresponding CMS shapes were provided to Entech in May 2024, these were reviewed spatially to examine any trends in overbreak from the hanging walls or footwalls.

Based on manual measurements of stope wall overbreak and a review of the provided Stope reconciliation Master Sheet, the dilution on Fault Affected Stopes (within 5 m of the Fault Gouge Zone) has been increased by 0.5 m, there is no change to the dilution factor outside of the 5.0 m boundary (NFA).

Follow up geotechnical drilling, logging, and sampling will be required to improve confidence in assumptions made in this study and to confirm the rock mass characteristics of all mining areas for the life of mine design and infrastructure.

Due to the geographic location of the mine (within a hill above the valley floor), no significant issues are expected to be caused by mining-induced stresses at the current designed mining depths and proposed mining methods. However, a suite of in-situ stress measurement testing utilising either the WASM AE (Acoustic Emission) method or Hollow Inclusion Cell (HI-Cell) method is advised to be commissioned from the Kora mining area to confirm this assumption. On-going visual inspections and analysis of stope and development performance is also recommended during mining to determine if stress-related issues are becoming prevalent within the mine.

A 3D geotechnical model was developed with the logging data, geological models and mine designs utilised for visualisation of geotechnical data, and to determine spatial trends within the data sets. At this stage, boundaries between geotechnical domains have been based primarily on proximity to stoping and mining areas, i.e. hangingwall, footwall and ore. There is currently insufficient data density to further define geotechnical domains.

Analyses were undertaken in the 2021 Kora Geotechnical Feasibility Assessment to define stoping parameters at Kora and Judd. These included stope stability analyses using the Mathews Potvin Stability Graph Method, overbreak/expected dilution, and 3D numerical modelling to validate sequences and stand-off distances of infrastructure. There has been no additional data collected since this time to justify reviewing these parameters.

Indications are that a Modified Avoca (with rock fill) mining method, with 20 m level spacings and bottom-up mining will be suitable in the short term while waiting for the establishment of the paste fill plant. Once the plant is established, open stoping with cemented pastefill with both bottom-up & top-down mining sequence will be suitable for the narrow, near vertical orebodies. Stopes may be extracted in a transverse mining sequence throughout the wider sections of the orebody, extracted continuously and in a longitudinal manner in the narrow sections of the orebody.

Overall stability of the Kora stoping panels is largely controlled by the proximity of the K1 hangingwall and K2 footwall to the Fault Gouge Zone (FGZ). Rock mass conditions in terms of Q rating ranged significantly from 'Extremely Poor' up to 'Extremely Good'. These worst and best cases are considered rarer occurrences being spatially limited and linked to lithology contacts and/or fault / shear zones, with the ground deteriorating with increased proximity to the FGZ. On average the both the Kora and Judd orebodies can be classed as good ground. Table 16-1 outlines the recommended stope spans and dilution estimates for the Kora and Judd orebody.

Table 16-1 Summary of Assessed Stoping Parameters for Kora and Judd

Orebody	Parameters	Hangingwall	Footwall
K1 (NFA)	Allowable Strike Length	16 m	20 m
	Dilution	0 - 0.5 m	0 - 0.5 m
K1 (FA)	Allowable Strike Length	6 m	6 m
	Dilution	0.5 - 1.0 m	0.5 - 1.0 m
K2 (NFA)	Allowable Strike Length	31 m	19 m
	Dilution	0 - 0.5 m	0 - 0.5 m
K2 (FA)	Allowable Strike Length	6 m	6 m
	Dilution	0.5 - 1.0 m	0.5 - 1.0 m
Judd	Allowable Strike Length	35 m	35 m
	Dilution	0 - 0.5 m	0 - 0.5 m

The FGZ is often situated between the K1 and K2 veins. Where it is proximal or intersecting, there is the possibility of slabbing, sliding and unravelling failure types accruing along structure planes and the FGZ. To control this, restricting stope spans, avoidance of undercutting of the FGZ or within a critical standoff distance of the FGZ, along with drill and blast practices is required to minimize stoping performance issues. This is the current practice and is expected to be improved upon with Pastefill.

16.3 Current Site Mining Methods

Currently Avoca and Modified Avoca stoping methods (as described below) are used on site.

Note: this selection process was conducted as part of the 2022 DFS with the findings still current for the 2024 DFS.

16.4 Mining Method Selection Process

The 2020 PEA, in combination with the 2022 site mining practices, were the basis for the following inputs with regards to mining method:

- Only underground mining methods considered for extraction.
- The sub-level spacing will be 20 m floor to floor for LHS methods.
- Only mechanised mining methods to be considered for stoping.
- Lateral development will be completed by jumbo drills.
- Vertical development will be completed by mechanical methods.

The following process was completed for the mining method trade-off analysis for the initial few years of mining, and then the remaining LOM:

- Produced a long list of potential mining methods for consideration.
- Eliminated mining methods and defined a short list of potential mining methods for consideration.
- Generated a set of stope optimisation parameters for each of the potential mining methods.
- Completed stope optimisations over a range of cut-off grades (COG).
- Estimated mining physicals and mining costs.
- Produced excel based schedules with stope optimisation results and a range of production rates.
- Assessed the resulting inventory metrics by COG.
- Generated a cashflow and net present value (NPV) model.
- Assessed the NPV's in combination with other inventory metrics.
- Selected a cut-off grade and mining method(s) as a basis for the feasibility study.

16.4.1 Mining Method Long List

A long list of mining methods was produced to take a systematic approach in identifying potential mining methods for Kora. Table 16-2 shows the four key criteria applied against the considered mining methods, these criteria being technical feasibility, mining cost, resource recovery, and production rate.

Table 16-2 Evaluation Criteria

Evaluation Criteria	Evaluation Options		
Technical Feasibility	Yes	No	
Mining Cost	Low ($ 0-25 / t)	Medium ($ 25-50 / t)	High ($ >50 / t)
Resource Recovery	Low < 60 %	Medium 60 - 80 %	High > 80 %
Production Rate	Low < 1.0Mt per annum	Medium 1.0 - 1.5 Mt per annum	High > 1.5 Mt per annum

Using the 2020 PEA as a guide, and in consultation with K92, the following longlist was generated. Handheld mining methods and mechanised mining methods with low production rates such as cut and fill were discounted. The mining method long list is shown Table 16-3.

Table 16-3 Mining Method Long List

2 Year Assessment

Method	Backfill / No Backfill	Comment	To Consider / Eliminated
LHOS Pillars	No Backfill	Technically feasible, low cost, low resource recovery, medium production rate	To Consider
Avoca	Backfill	Technically feasible, medium cost, medium resource recovery, low production rate	To Consider
LHOS CRF	Backfill	Technically feasible, medium cost, medium resource recovery, low production rate	To Consider

LOM Assessment

Method	Backfill / No Backfill	Comment	To Consider / Eliminated
LHOS Pillars	No Backfill	Not feasible due to no reduction in TSF capacity requirement, low cost, low resource recovery, medium production rate	Eliminated
Avoca	Backfill	Not feasible due to no reduction in TSF capacity requirement, medium cost, medium resource recovery, medium production rate	Eliminated
LHOS CRF	Backfill	Not feasible due to no reduction in TSF capacity requirement, medium cost, medium resource recovery, medium production rate	Eliminated
SLC	No Backfill	Not feasible due to no reduction in TSF capacity requirement, low cost, medium resource recovery, high production rate	Eliminated
LHOS Pastefill	Backfill	Technically feasible, medium cost, high resource recovery, medium production rate	To Consider

For the initial few years of mining all three LHS methods remained for consideration in the short list. For the remaining LOM, some of the key considerations were that the mining method must result in a lowest mining unit cost, maximise value, and reduce the surface tailings storage facility requirements. This eliminated sub level caving (SLC), in addition to longhole open stoping (LHOS) with pillars, Avoca/modified Avoca, and LHS with cemented rockfill methods. After these eliminations, the result was selection of a mining method incorporating paste fill, which would reduce the tailings storage facility capacity requirements.

A cut and fill methodology utilising paste fill was discounted due to the low production rate and higher cost in comparison to LHS methods. The LOM mining method analysis therefore led to the selection of LHS backfilled with paste fill.

16.4.2 Short Listed Mining Methods

The following section provides a description of the short-listed mining methods for the initial two years of the LOM plan and the remainder of the LOM plan.

Longhole Open Stoping With Pillars

LHOS with pillars is a mechanised mining method commonly used throughout underground mines globally. The proposed LHOS stoping design assumptions included 20 m vertical level spacing, a top-down extraction sequence, and 76 or 89 mm diameter up-hole production drilling. A schematic depicting an example LHOS extraction sequence is shown in Figure 16-1.

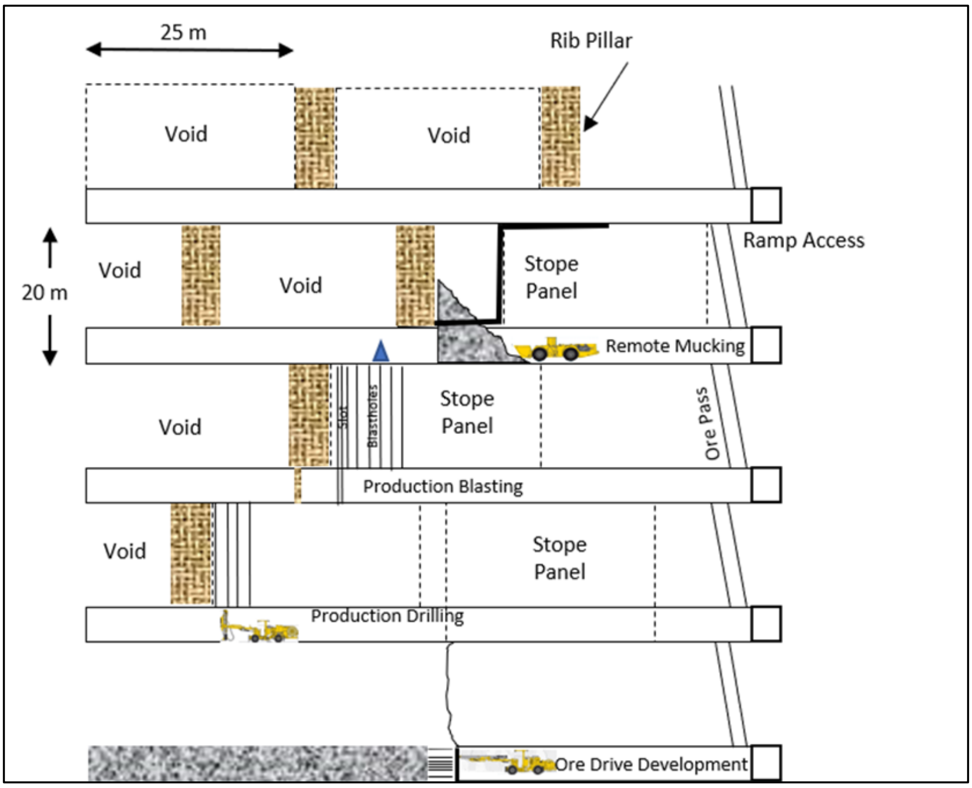


Figure 16-1 Example Long Section of longitudinal LHOS Extraction

Avoca

Avoca is a mechanised mining method which is widely used in global underground mining. The proposed Avoca stoping design assumptions included a 20 vertical level spacing, a bottom-up extraction sequence, and 76 mm or 89 mm diameter down-hole production drilling. The stoping panels are subdivided into shorter strike openings, which are then backfilled with unconsolidated rockfill.

The next set of rings are then fired against this waste rock material, the ore bogged, and the process repeated. In Avoca there is end on (dual access) to the stoping horizon at both ends at the top and the bottom. This allows for ore extraction to occur from the bottom at one end of the stoping panel and backfilling to then be conducted from the opposite end of the stoping panel from the level above.

Avoca assumes a continuous filling mining method with the following steps:

1. Ore drives are developed along strike at the top and bottom of the stoping block or panel using jumbo drills as per Figure 16-2.

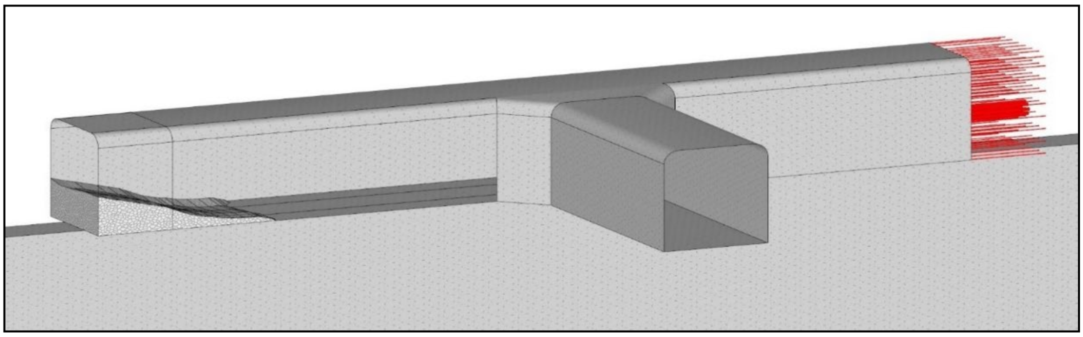


Figure 16-2 Step 1 - Ore Development

2. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block / panel. A strike length of approximately 5 m for each firing is typical.
3. Blasted ore is bogged to level stockpiles or ore passes for truck loading using conventional diesel-powered underground loaders, utilising both conventional (i.e. manual) and tele-remote techniques.
4. Waste material is dumped into the stope void from the level above.
5. Once filled, the next stope panel along is fired against the unconsolidated waste fill which allows sufficient void for firing into without requiring another slot to be established as per Figure 16-3.

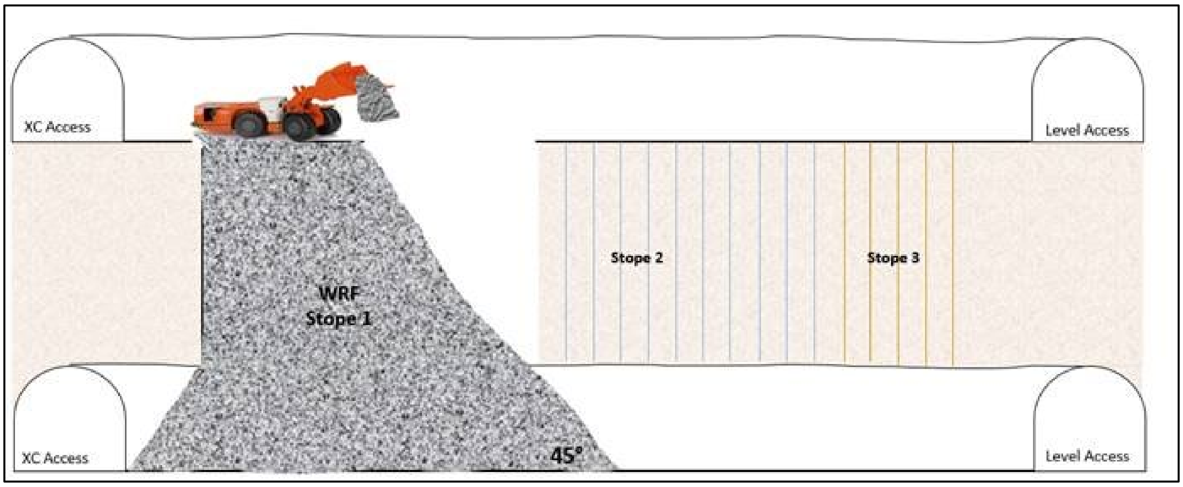


Figure 16-3 Avoca Mining Method Schematic

16.4.3 Modified Avoca

Modified Avoca is a mechanised mining method which is well understood in underground mining environments. The proposed modified Avoca stoping design assumptions included a 20 m level spacing, a bottom-up extraction sequence, and 76 mm or 89 mm diameter down-hole production drilling. The stoping panels are subdivided into shorter strike openings, which are then backfilled with unconsolidated rockfill. The next set of rings are then fired against this waste rock material, the ore bogged, and the process repeated. It differs from traditional Avoca as end-on access from both ends is not required, so loading and backfilling can occur from the same end of a stoping panel.

Modified Avoca assumes continuous fill mining method involves the following steps:

Ore drives are developed along strike at the top and bottom of the stoping block or panel using jumbo drills as per Figure 16-4.

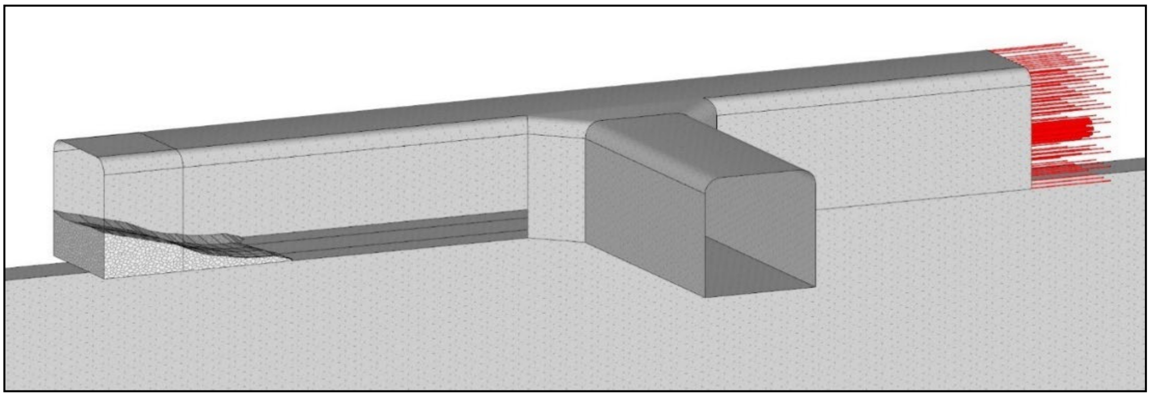


Figure 16-4 Step 1 - Ore Development

1. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block. A strike length of 5 m for each firing is shown in this example as per Figure 16-5.

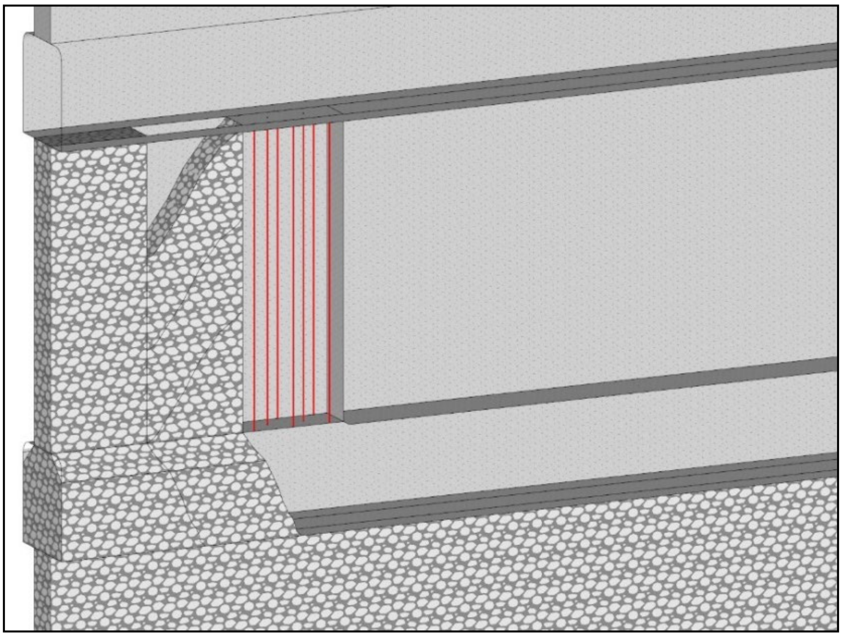


Figure 16-5 Step 2 - Stope Drill & Blast

2. Blasted ore is bogged to level stockpiles for truck loading using conventional diesel-powered underground loaders, utilising both conventional (i.e. manual) and tele-remote techniques as in Figure 16-6.

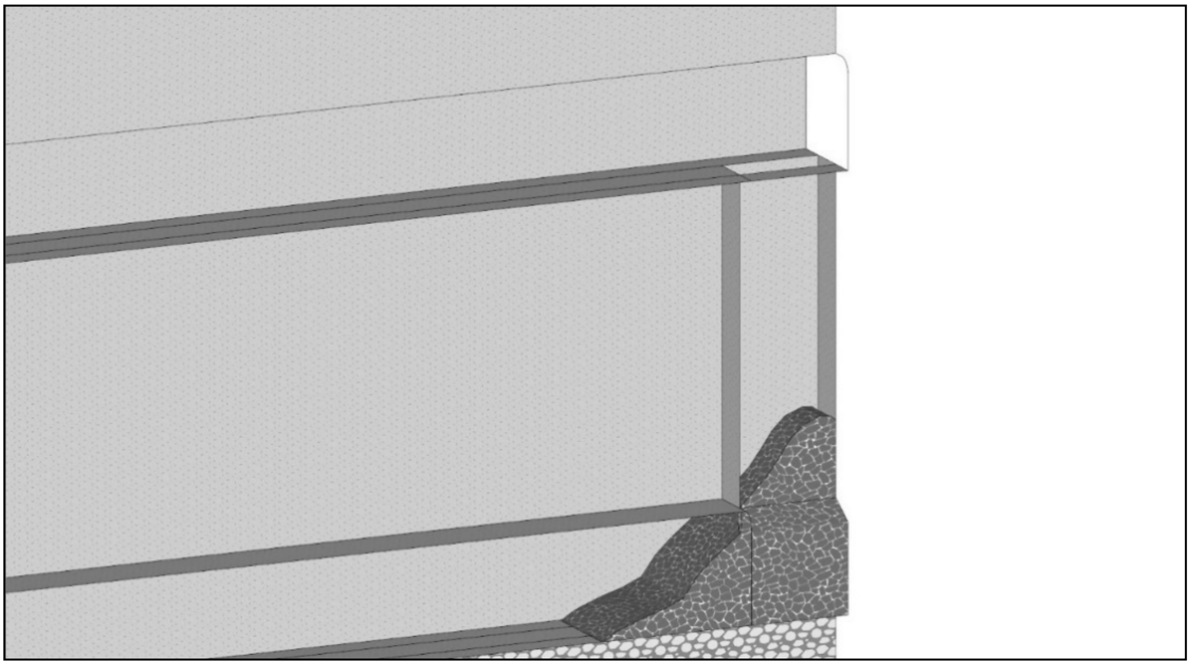


Figure 16-6 Step 3 - Stope Loading

3. Waste material is dumped into the stope void from the level above as per Figure 16-7.

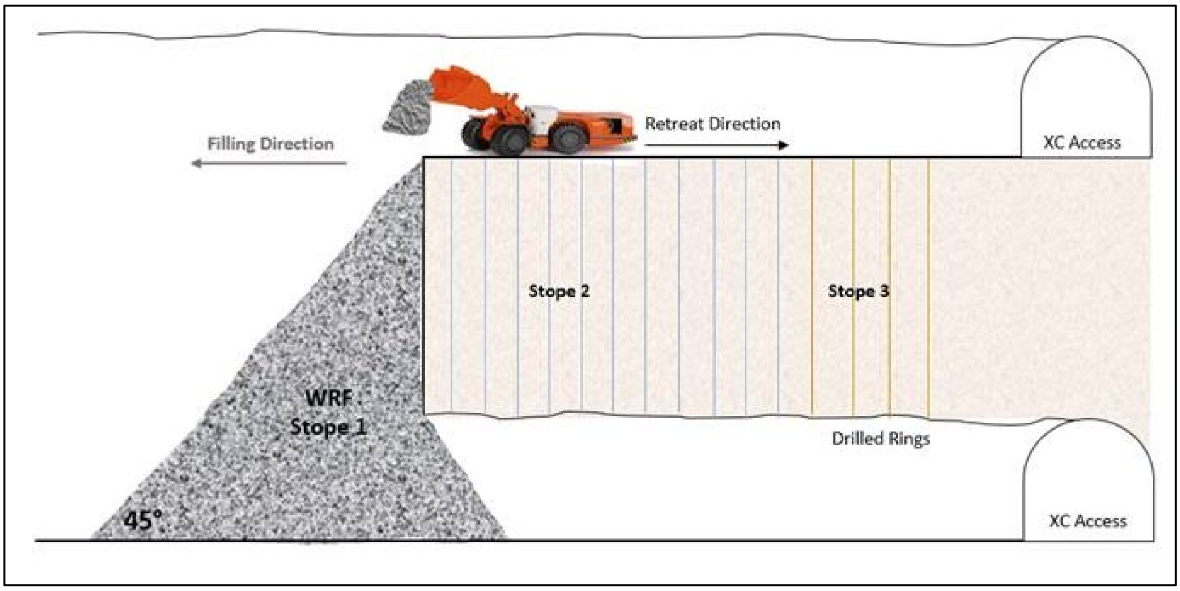


Figure 16-7 Modified Avoca Waste Tipping and Cycle

4. Once filled, the next stope panel along is fired against the unconsolidated waste fill which allows sufficient void for firing into without requiring another slot to be established as per Figure 16-8.

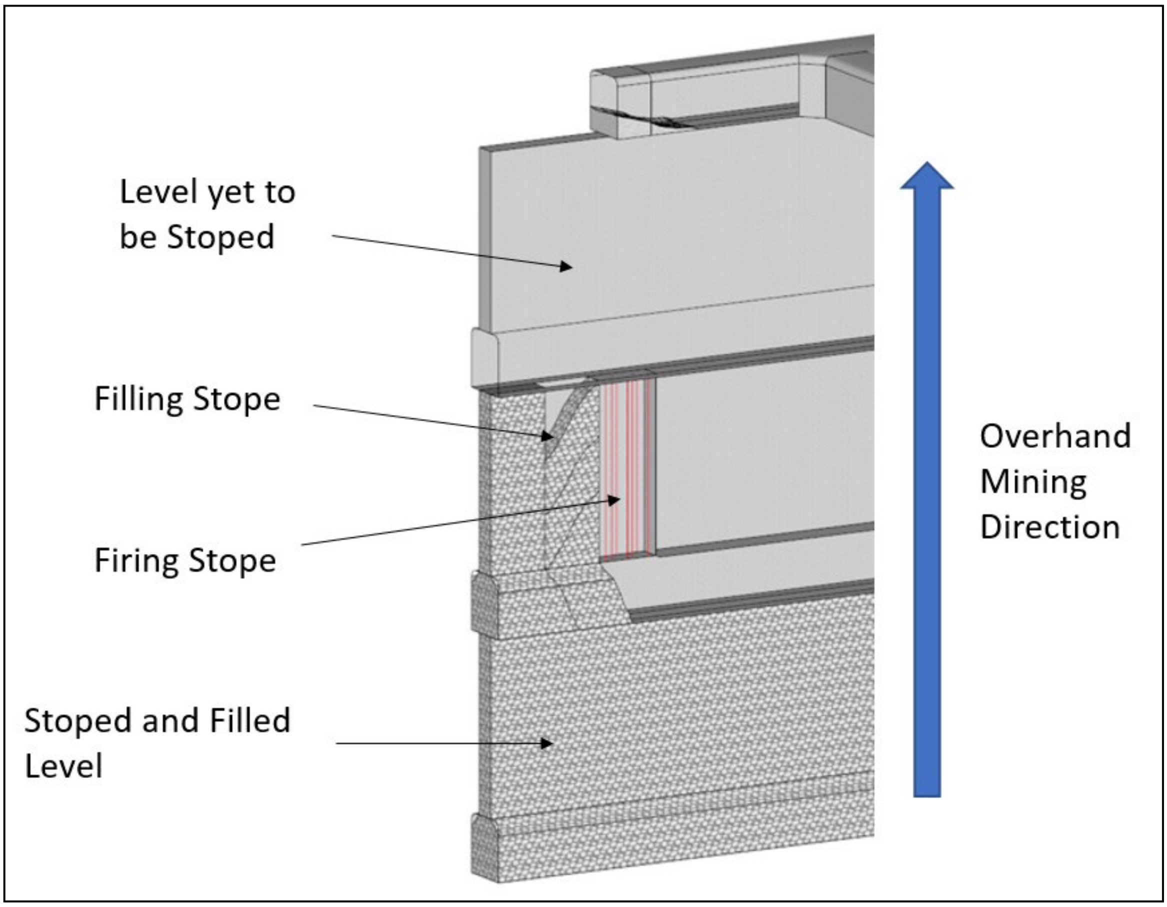


Figure 16-8 Modified Avoca Overall Mining Method Schematic

16.4.4 Longhole Stoping with Cemented Rock Fill

The LHS with CRF assumes a continuous fill mining method involving the following steps:

1. Ore drives are developed along strike at the top and bottom of the stoping block or panel using jumbo drills as per Figure 16-9.

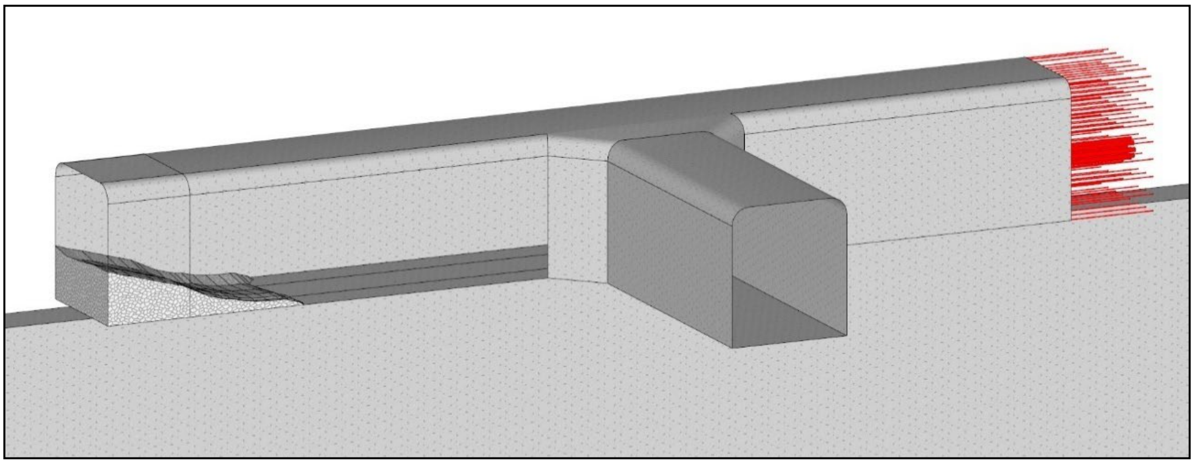


Figure 16-9 Step 1 - Ore Development

2. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block. A strike length of 5 m for each firing has been assumed in this example as per Figure 16-10.

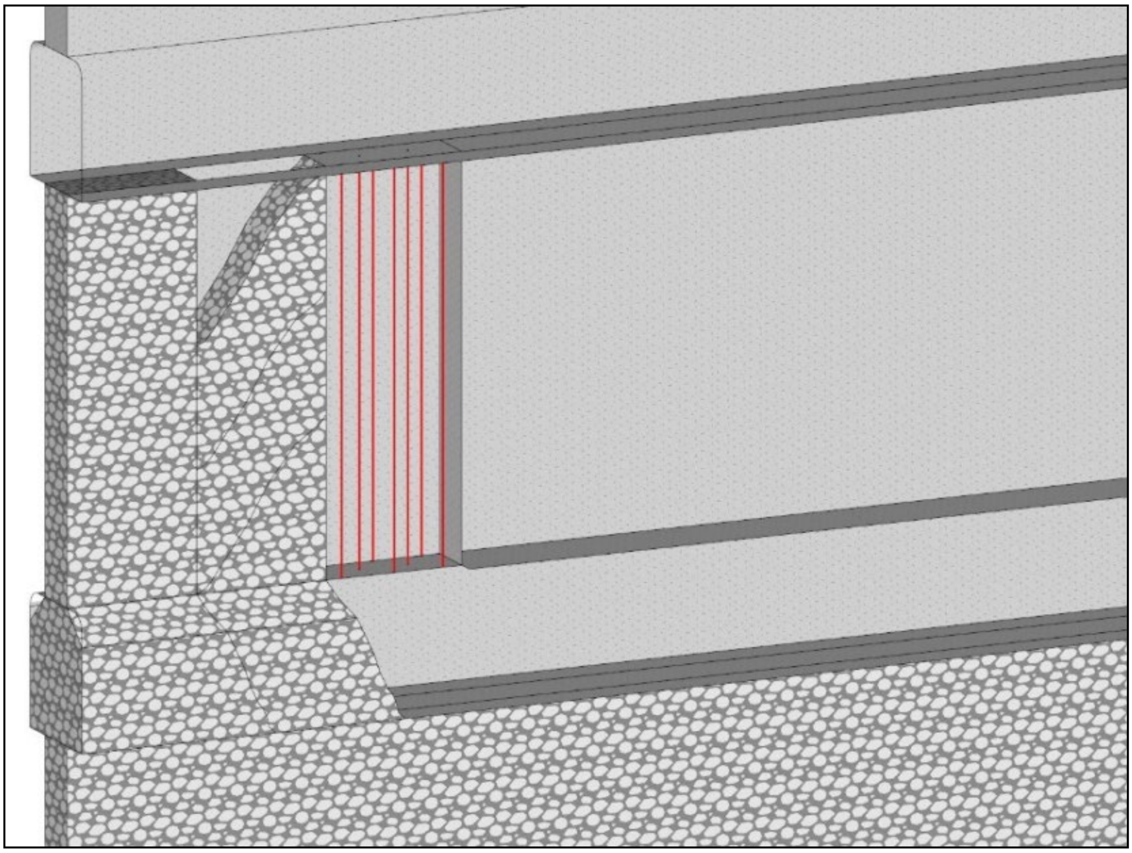


Figure 16-10 Step 2 - Stope Drill & Blast

3. Blasted ore is bogged to level stockpiles for truck loading using conventional diesel-powered underground loaders, utilising both conventional (i.e. manual) and tele-remote techniques as per Figure 16-11.

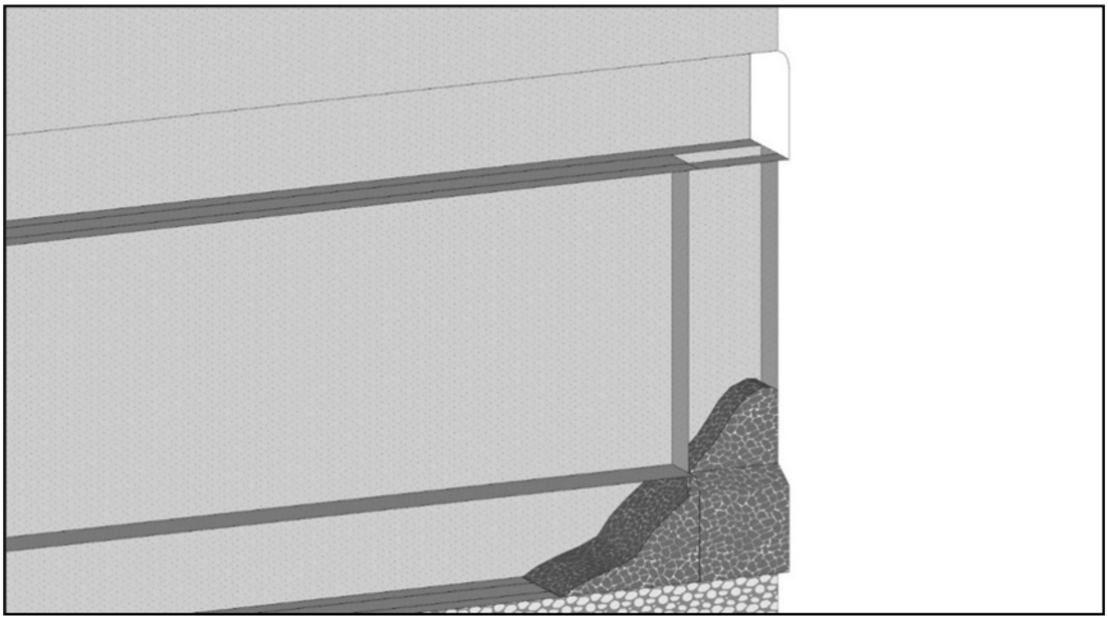


Figure 16-11 Step 3 - Stope Loading

4. Waste material mixed with a cement slurry is dumped into the stope void from the level above.
5. Once filled, the next stope along is fired against the fill typically within 12-36 hours which allows for firing against the fill without requiring another slot to be established. Alternatively larger stopes can be excavated, filled with the cement slurry, and a new slot opening will be required to mine the next adjacent panel in the sequence.

16.4.5 Longhole Stoping with Paste fill

LHS with paste fill is proposed to be utilised primarily in longitudinal extraction but broken into multiple panels along strike. Footwall drives are developed on levels to allow multiple access points to the orebody to generate additional mining fronts. The proposed mining sequence will be a combination of bottom up and top-down extraction. There will also be opportunity to utilise transverse extraction where required in wider areas of the ore zone or as dictated by geotechnical requirements.

The use of paste fill as a backfill method will allow more selectivity in extraction, and a higher extraction ratio which can be regulated with the paste fill cement content, as well as increased flexibility in extraction sequence allowing multiple mining fronts to achieve higher production rates. Geotechnical conditions support a flexible mining sequence, however stoping in proximity to the Fault Gouge zone requires different stope dimension and dilution considerations, and potentially a stricter extraction sequence. A depiction of the stoping cycle (drill and blast, bog, paste fill) utilised in a top down / underhand mining sequence is shown in Figure 16-12.

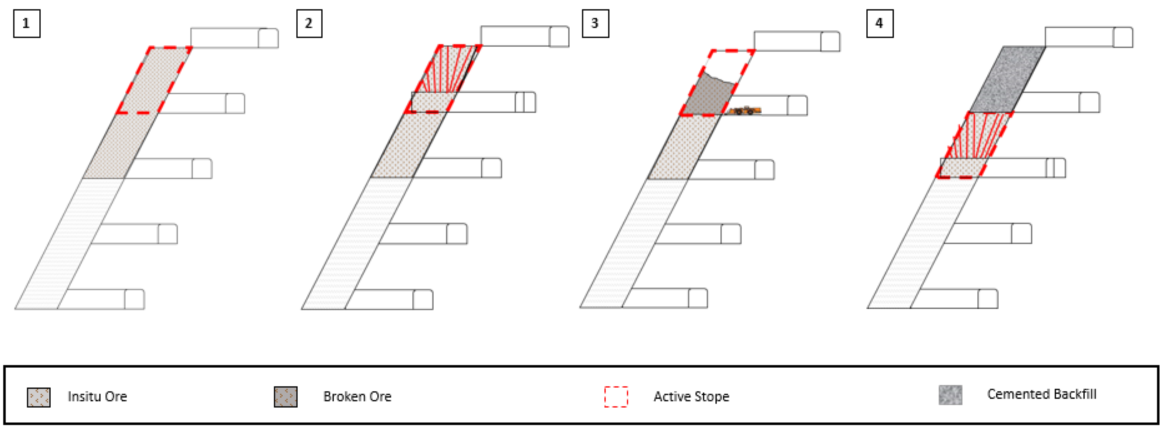


Figure 16-12 Example Cross-section of Stoping Extraction with Paste Fill Mining Cycle

16.4.6 Final Method Selection

The mining method selection process resulted in Avoca and modified Avoca being selected for mining prior to commissioning of the paste fill plant in Q3 2025, and longhole stoping with pastefill for the remainder of the mine life. An economic comparison was done in 2022 on extending the Avoca and modified Avoca mining methods for the full life of mine schedule instead of transitioning to longhole stoping with paste fill and confirmed that longhole stoping with pastefill provided superior project economics.

16.5 Stope Design

On completion of the trade-off studies in 2022, Entech were provided with the 2022 DFS geological model. Stope optimisations were completed to compare to the trade-off studies and validate the selected Avoca, and LHS with paste fill mining methods. The stope optimisation results aligned with the findings of the trade-off studies and K92ML confirmed selection of the Avoca/Modified Avoca and LHS with paste fill mining methods for design and scheduling of the FS.

A new DFS geological model completed in September 2023 was provided for this (2024) update of the DFS mine plan - it includes Measured and Indicated Mineral Resources.

16.5.1 Initial Cut-Off Grade Estimation

A provisional cut-off grade was estimated based on costs derived from both the 2022 DFS and PEA studies along with updated inputs provided by K92. The cut-off grade is typically generated as a base input for stope optimisation to generate an operating only, and incremental stoping cut-off grade. The initial cut-off grade ranges are shown in Table 16-4. Initial Cut-off Grade Estimation below.

Table 16-4 Initial Cut-off Grade Estimation

Mining Cut-off Calculation	Units	Operating Cut-off	Incremental Stope Cut-off
Lateral Operating Development	USD / t ore	16.42	0.00
Stoping Cost	USD / t ore	60.86	60.86
Processing Cost	USD / t ore	50.70	50.70
Total Cost	USD / t ore	127.98	111.56
Gold Price	USD / ounce	1750	1750
Metallurgical Recovery	%	93.00%	93.00%
Royalties	%	2.50%	2.50%
Revenue	$/g AuEq	51.0	51.0
Operating Cut-off Grade	AuEq g/t	2.51	
Incremental Stope Cut-off Grade	AuEq g/t		2.19

This information was then used in the trade-off study to generate the lower bound of 2.5 g/t AuEq. for the DFS stope optimisations. To assess the appropriate final cut-off grade to use in design and scheduling, it was agreed to produce stope optimisations on a range of cut-off grades to conduct further analysis and compare results.

16.5.2 Stope Optimisation

The stope optimisation parameters for the DFS are shown in Table 16-5 and Table 16-6 below, based on the proposed mining method.

Table 16-5 LHOS with Pillars / LHS with Paste Fill

Stoping Parameter	Value
Stoping COG (g/t AuEq.)	2.5,3.0,3.5,4.0.4.5,5.0
Min. Mining Width (m)	2.0
Max. Mining Width (m)	Not Limited
Vertical Level Interval (m)	20
Section Length (m)	5
HW Dilution (m)	0.5
FW Dilution (m)	0.5
Min. Parallel Waste Pillar Width (m)	7.5

Table 16-6 Avoca

Stoping Parameter	Value
Stoping COG (g/t AuEq.)	2.5,3.0,3.5,4.0,4.5,5.0
Min. Mining Width (m)	2.0
Max. Mining Width (m)	10
Vertical Level Interval (m)	20
Section Length (m)	5
HW Dilution (m)	0.5
FW Dilution (m)	0.5
Min. Parallel Waste Pillar Width (m)	10

Stopes mined in the initial period of the LOM plan prior to paste plant construction and commissioning (until July 2025 of the mine plan as advised by K92), and therefore planned to be mined with the Avoca method, are shown in Figure 16-13 below. All areas outside of this rectangle are planned to be mined using LHS with paste fill.

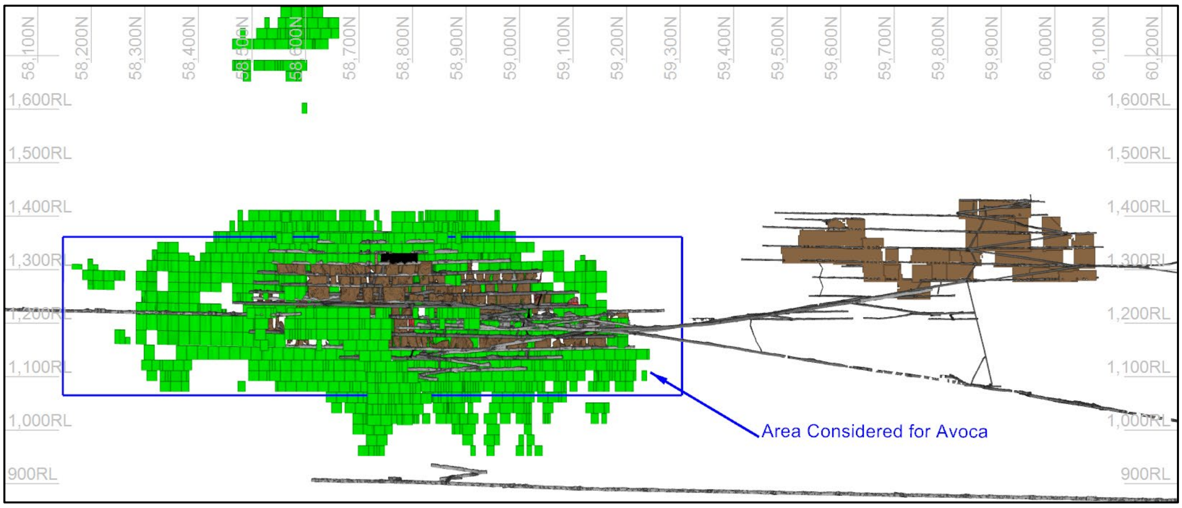


Figure 16-13 Kainantu As-builts with Preliminary MSO Shapes (Green)

An MSO metal inventory to Mineral Resource metal conversion was completed to ensure that the stope optimisations correctly captured economic material. The analysis considered the MSO AuEq metal ounces versus the AuEq metal ounces in the Mineral Resource at the same cut-off grade for only Measured and Indicated Mineral Resources.

This resource conversion analysis was completed for the Kora and Judd orebody's separately and then combined, which can be found in Figure 16-14, Figure 16-15, and Figure 16-16 below.

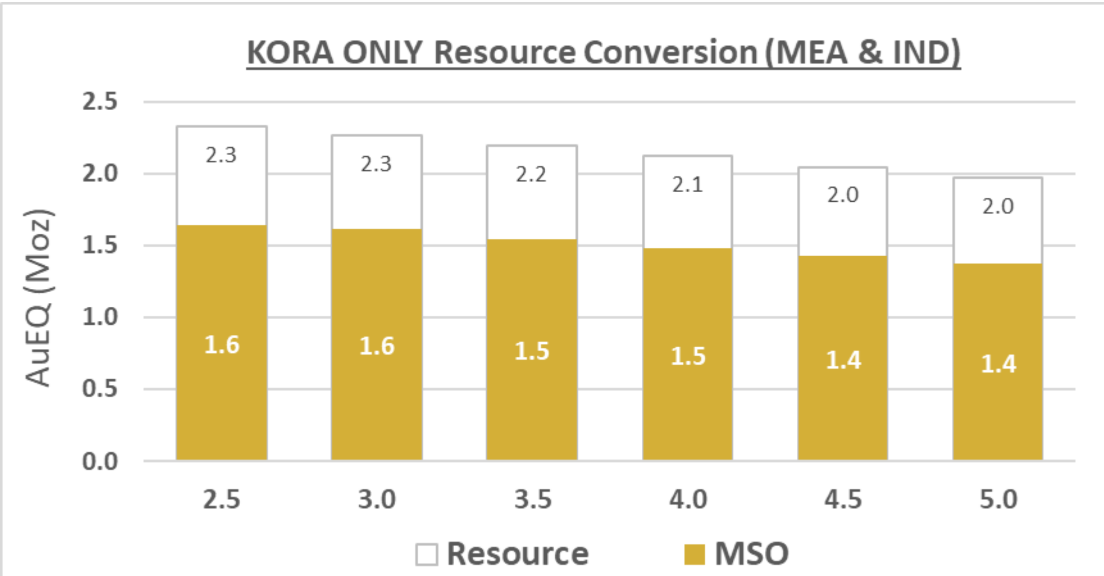


Figure 16-14 Kora Only MSO vs Resource - Moz AuEq (y-axis) vs Cut-off Grade (x-axis)

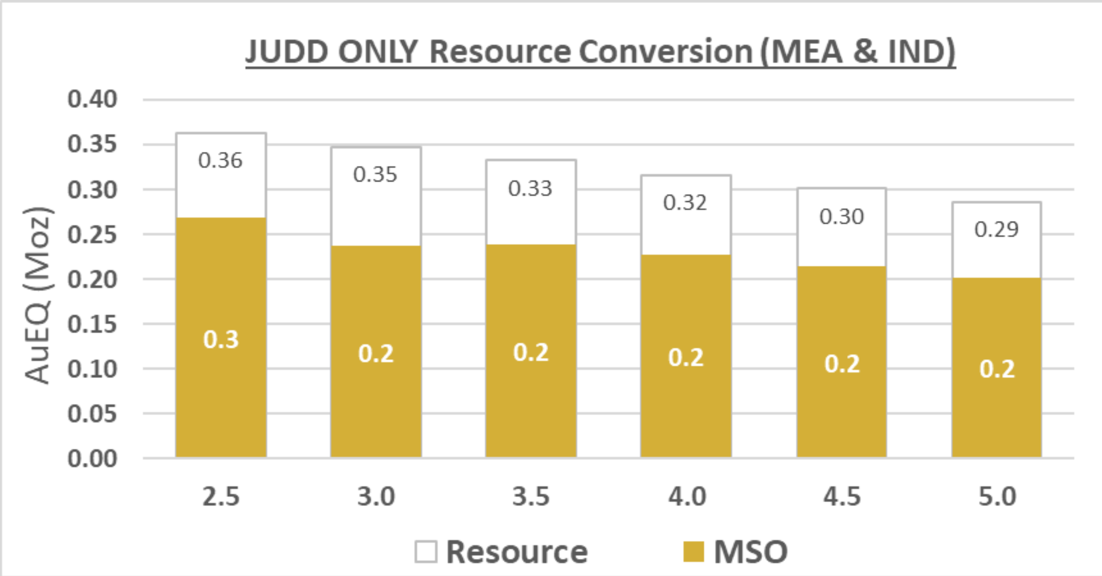


Figure 16-15 Judd Only MSO vs Resource - Moz AuEq (y-axis) vs Cut-off Grade (x-axis)

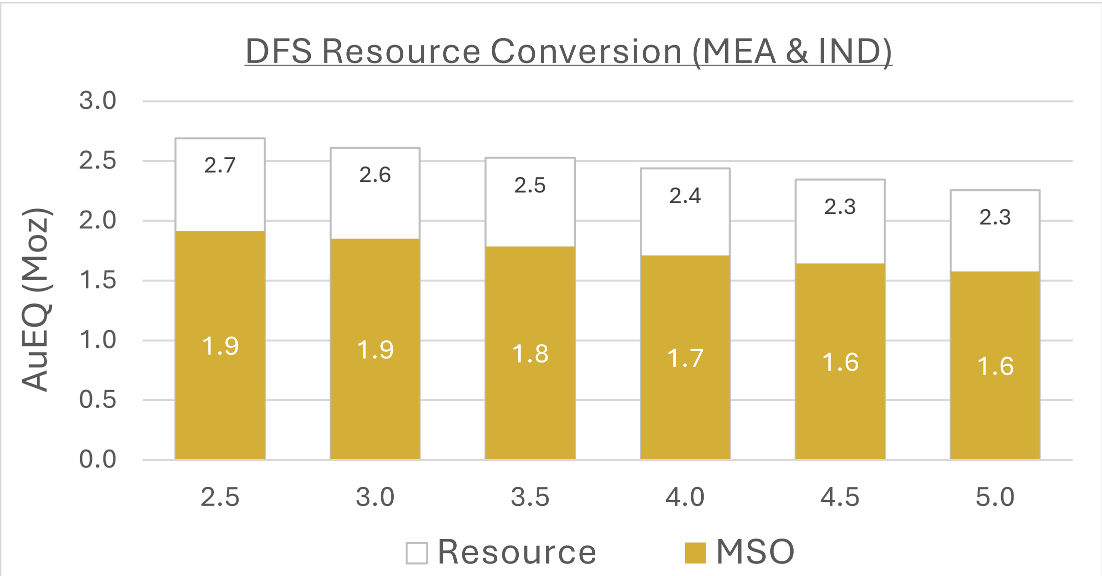


Figure 16-16 Total MSO vs Resource - Moz AuEq (y-axis) vs Cut-off Grade (x-axis)

A comparative net present value (NPV) analysis was carried out to assist in selection of the appropriate cut-off grade for the DFS. Note this NPV was estimated for a relative comparison only between cut-off grades and is not representative in actual value terms. The chart of the relative NPVs is shown in Figure 16-17 below.

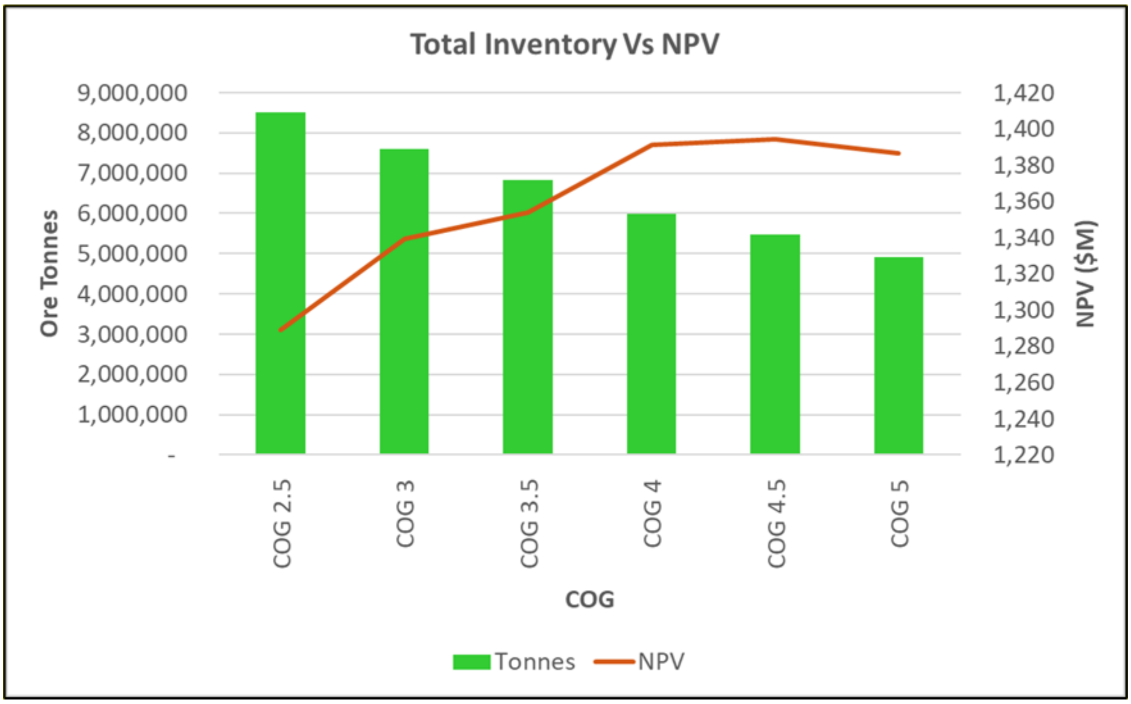


Figure 16-17 Comparative NPVs at COG Range 2.5 - 5.0 g/t AuEq Versus Inventory

16.5.3 Cut-Off Grade Selection

K92ML selected the 3.5 g/t AuEq stope optimisations to carry forward for the final mine design and DFS LOM scheduling. This selection was supported by the NPV analysis, cut-off grade estimates, and the trade-off study results as well as aligning with K92's objectives for inventory size and grade.

16.5.4 Stoping Methodology

The extraction methods that are proposed for the feasibility study are:

- Longitudinal LHS with paste fill.
- Avoca and modified Avoca mining.

The spatial application of these mining methods is shown in Figure 16-18.

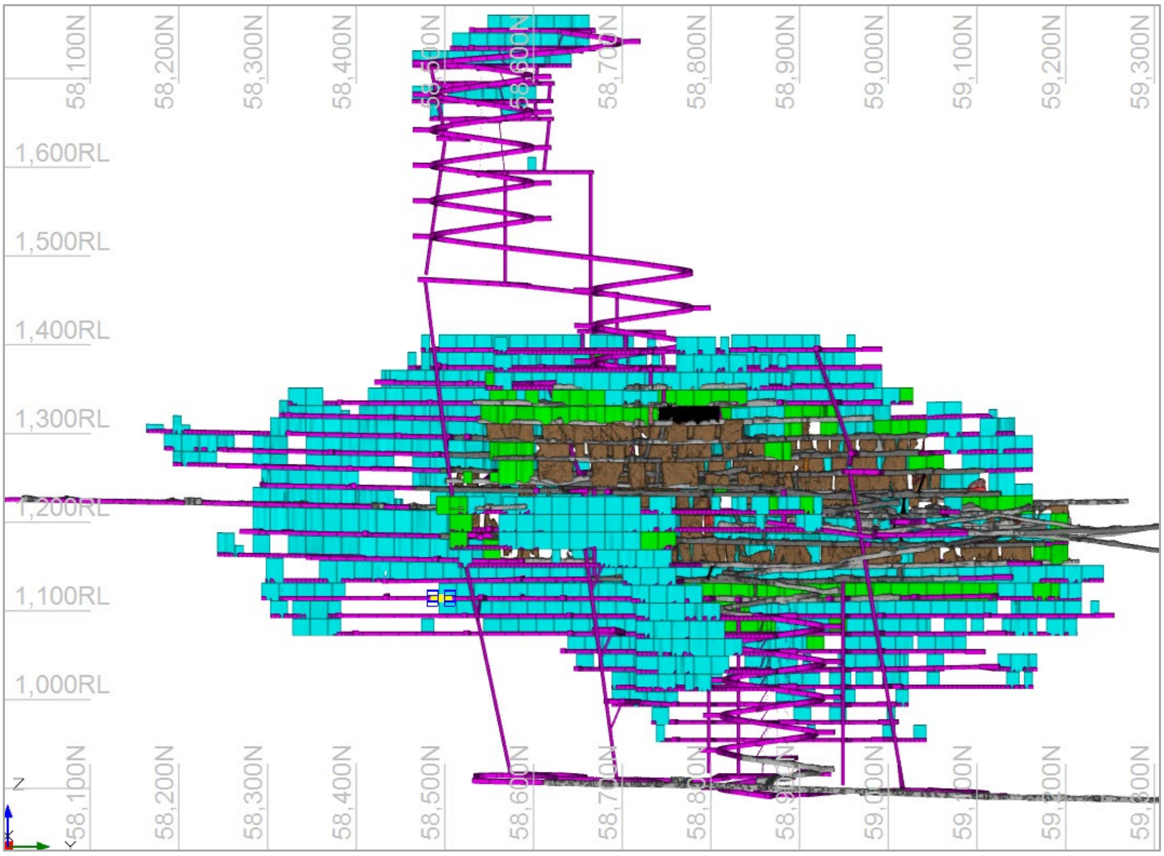


Figure 16-18 Stope Extraction Methods -Longitudinal LHS with Fill (blue), Avoca (green), As-builts in Brown

16.5.5 Stope Design Parameters

The mining method selection outcomes dictated the stope design parameters as per the MSO parameters described. Following the stope optimisation process, review and editing of the remaining stope shapes for depletion and mining practicality was conducted. The resultant stope shapes were reviewed by Entech's geotechnical consultant, and the design was endorsed. On completion of the final mine design and scheduling any sub-economic areas, when considering access costs, were removed from the schedule.

16.5.6 Ore Dilution

Dilution of 0.5 m on both the footwall and hanging wall of the stope shapes (1.0 m total), was applied during the optimisation phase. In addition to the 1.0 m dilution skin, additional dilution of 5% has been added for Avoca mined stopes and 2.5% for long hole stoping with pastefill. Where a stope is within 5.0 m proximity of the HW or FW of the fault gouge, an additional 1.0m of dilution was added at a grade averaging 1.42 g/t AuEq. This results in a total average dilution of 27.8%. These dilution parameters were based on Entech's geotechnical study.

Paste fill and Avoca dilution are based on benchmark and site dilution data. The LOM average stope dilution is 27.8%.

16.5.7 Ore Recovery

The recovery of all ore contained within a planned stope is generally not achieved due to:

- Ore loss on stope walls resulting from under-break (typically due to poor drilling and / or blasting results).
- Planned insitu pillars.
- Ore loss due to mixing with dilution when firing in Avoca.
- Inability to bog all ore from the stope due to remote loading difficulties.

To account for the above losses a mining recovery factor of 90% has been applied to Avoca stopes and 95% has been applied to LHOS stopes with paste fill. The 5% lower recovery factor for Avoca stoping accounts for ore loss, where ore and waste material mix and become sub-economic to mine.

16.5.8 Drilling, Blasting and Slotting Methodology

The current drill design philosophy for Kainantu is based around 89 mm blast holes, although a 76 mm blasthole could also be suitable for some stoping areas. Underground blasting of a 2 - 2.5 m burden for 89 mm blastholes generally produces a good fragmentation size distribution and safe margin from resultant brow line to each subsequent production ring. Table 16-7 outlines typical burden and spacing parameters that would be used for this hole size.

Table 16-7 Burden and Spacing Parameters

Drill Direction	Hole Diameter	Burden
Upholes	89 mm	2.0 - 2.5 m
Downholes	89 mm	2.0 - 2.5 m

When completing ring design, the toe spacing is determined by the perpendicular distance between the toe of the shorter hole and the adjacent longer hole as per the example Figure 16-19. Benchmark data shows spacing is typically linked to the burden as per the following formula:

- S = (1.0 to 1.4) x B; where
- S is the Spacing; and
- B is the Burden.

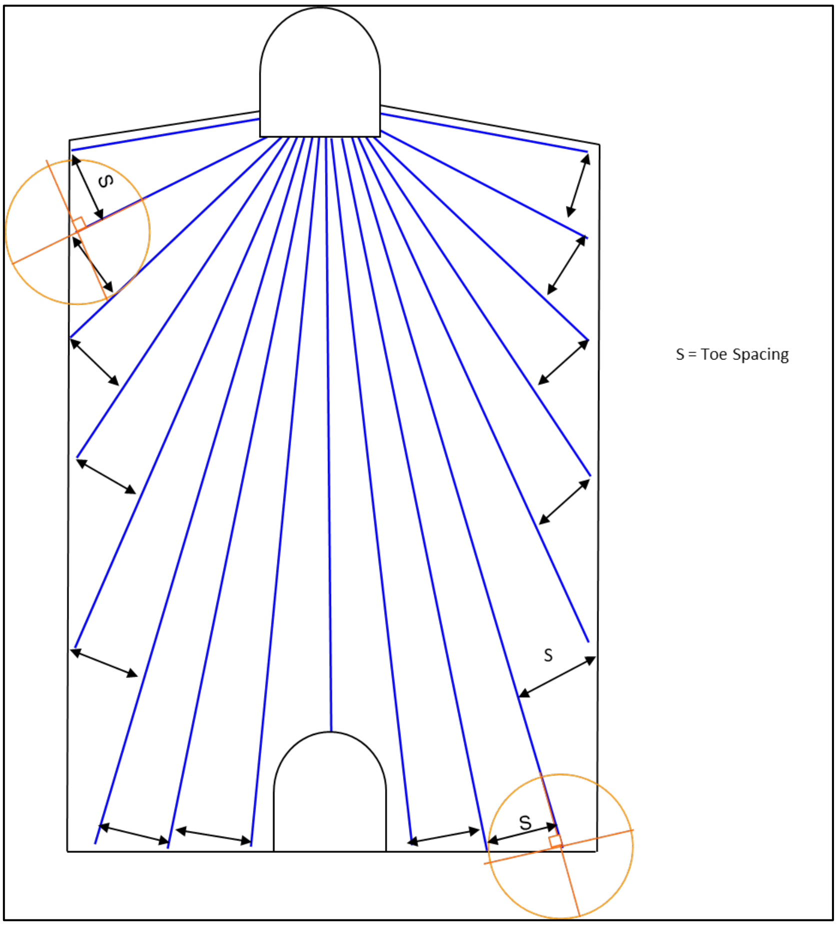


Figure 16-19 Production Drill Spacing Example

The drill yield used for the DFS is 6.0 t/dm which aligns with current site parameters of 5.5-6 t/dm and is appropriate for the average stope width.

Table 16-8 outlines the percentage of drilled metres that is charged in each stope size. These values have been used in calculating the charge metres and hence the total explosives used.

Table 16-8 Charged Amount of Drill Metres by Stope Type (%)

Stoping Area	% of Drill Metres Charged
LHS paste fill/Avoca	80%

For the LHS stopes, 0.75 m diameter slots are planned to be developed using an Epiroc Easer L machine for blasting voids as illustrated in Figure 16-20. Blastholes will be drilled at a diameter of 76/89 mm around box holes to create initial void. This is a proven pattern commonly used in underground LHS operations. Figure 16-21 shows the 'Epiroc Easer L' rig.

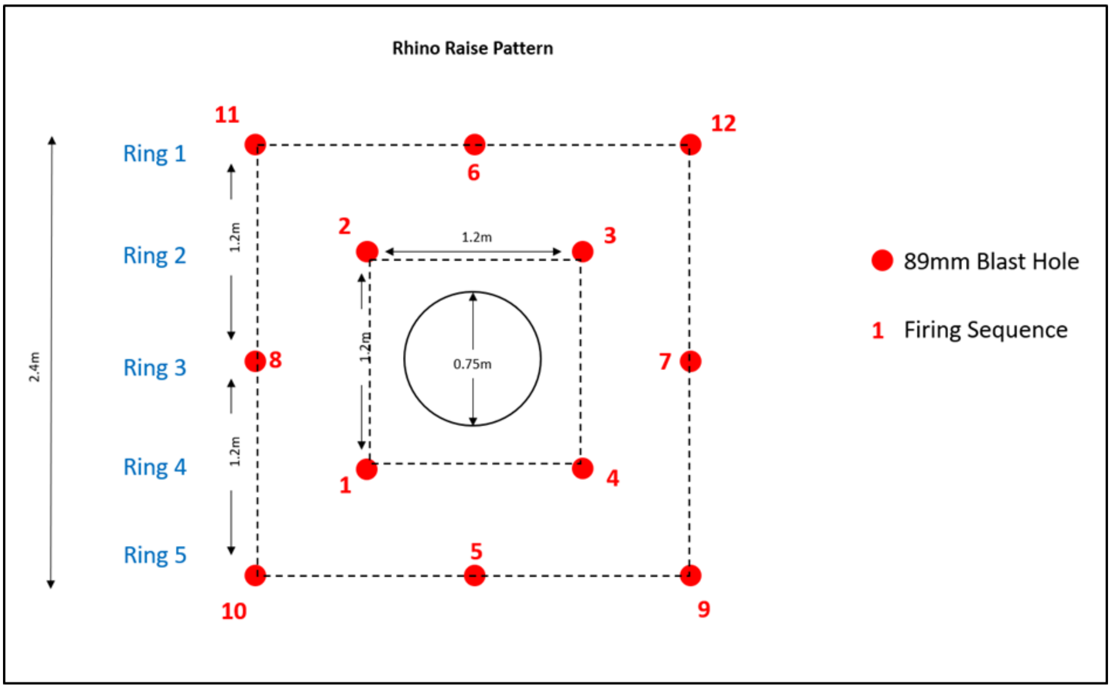


Figure 16-20 Example Slot Around 0.75 m Boxhole

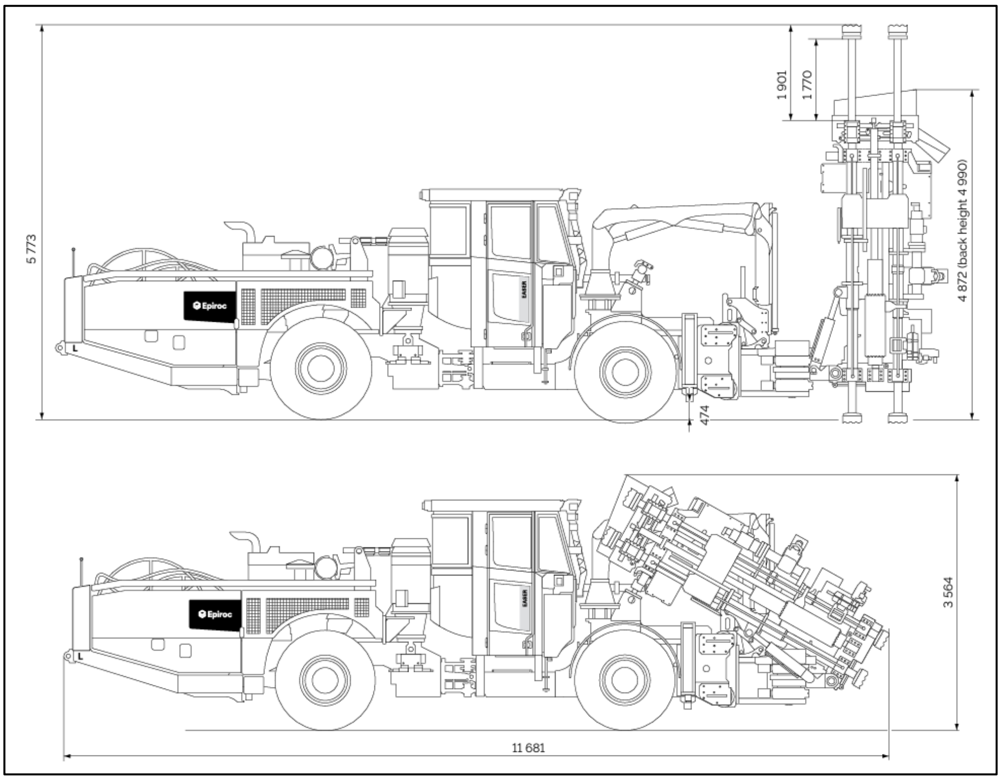


Figure 16-21 Easer L Raisebore Rig

16.5.9 Avoca and Modified Avoca Stoping

16.5.10 Stope Cycle

The stoping cycle is depicted in Figure 16-22 and a stope cycle time in Table 16-9.

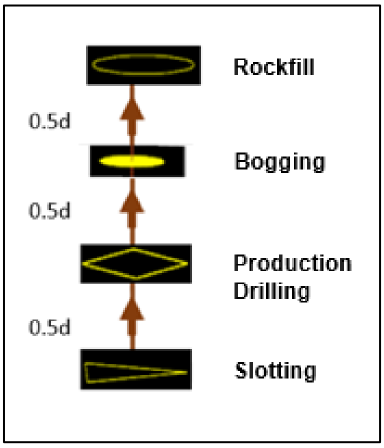


Figure 16-22 Stoping Cycle by Activity as Linked in Deswik (Activities Occur from Bottom to Top of Image), Slotting Only Occurs on Initial Stopes Mined in a Series of Panels

Table 16-9 Stope Cycle Activities and Their Quantity, Task Rate and Duration for an Average Size Avoca/Modified Avoca Stope

Activity	Quantity	Task Rate	Duration
Production drilling	1,200 m	250 m /d	4.8 d
Delay (Survey, engineering, blasting)			0.5 d
Loading	7,250 t	1000 t / d	7.25 d
Delay (Survey, fill walls, fill reticulation)			3.0 d
Backfill stope	3,340 m^3	500 m^3 / d	6.7 d
Total Stope Cycle Time			22 d

16.5.11 Longitudinal Sublevel Stoping with Paste Fill

16.5.12 Stope Cycle

The stope cycle activities, and stope cycle time for an average size LHS are shown in Figure 16-23 and Table 16-10. For a vertical paste exposure, the backfill cure activity is 14 days and 28 days for horizontal paste exposure.

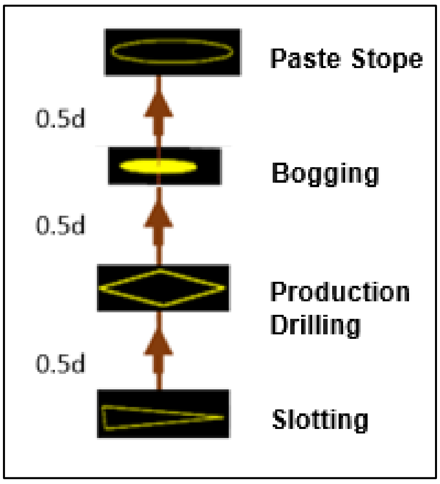


Figure 16-23 Stoping Cycle By Activity as Linked in Deswik (Activities Occur from Bottom to Top of Image)

Table 16-10 Stope Cycle Activities for an Average Size LHS with Paste Fill Stope

Activity	Quantity	Task Rate	Duration
Boxhole drilling (Easer L)	15 m	10 m/d	1.5 d
Delay (Survey, engineering)			1.0 d
Production drilling	1,350 m	250 m /d	5.4 d
Delay (Survey, engineering, blasting)			0.5 d
Loading	6,800 t	1,000 t / d	6.8 d
Delay (Survey, fill walls, fill reticulation)			3.0d
Backfill stope	3,000 m^3	1,800 / d	1.7 d
Backfill Cure (Adjacent/Underhand)			14/28 d
Total Stope Cycle Time			34/48 d

16.5.13 Stoping Sequence

The typical stoping and linking sequence for underhand longitudinal LHS with paste fill is shown in Figure 16-24, with overhand sequence in Figure 16-25.

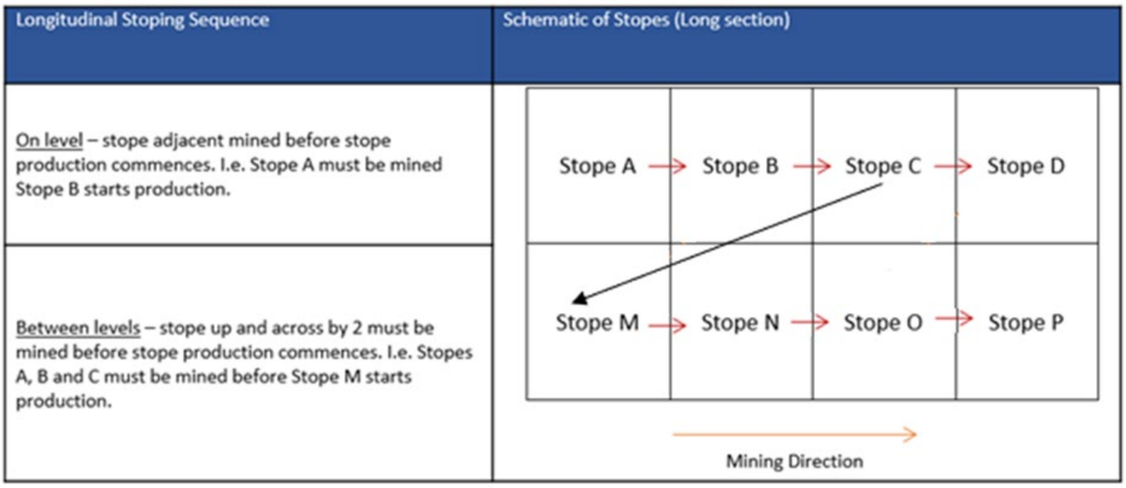


Figure 16-24 Example of a Longitudinal LHS with Paste Fill Underhand Mining Sequence (Sequence will Vary in Areas)

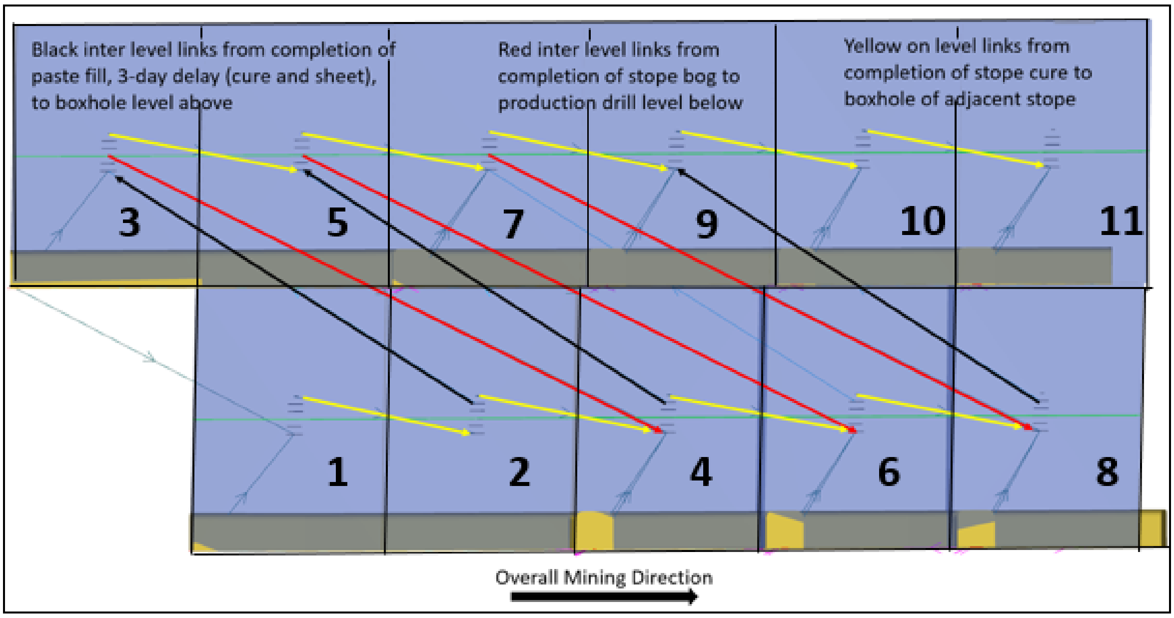


Figure 16-25 Example of a Longitudinal LHS with Paste Fill Overhand Mining Sequence (Sequence will Vary in Areas)

16.5.14 Backfill

Paste fill has been scheduled at approximately 105 cubic metres of paste per hour for a total of 17 hours per day. This equates to a filling rate of 1,800 cubic metres of paste per day.

16.6 Development Design

The mine access and development has been designed to suit the selected mining strategy and focuses on operational efficiency and bulk material movement. Existing underground workings have been incorporated into the design. There is an existing portal into the mine, the 800 Portal, in addition to two haulage inclines currently being excavated. The underground layout can be seen in Figure 16-26.

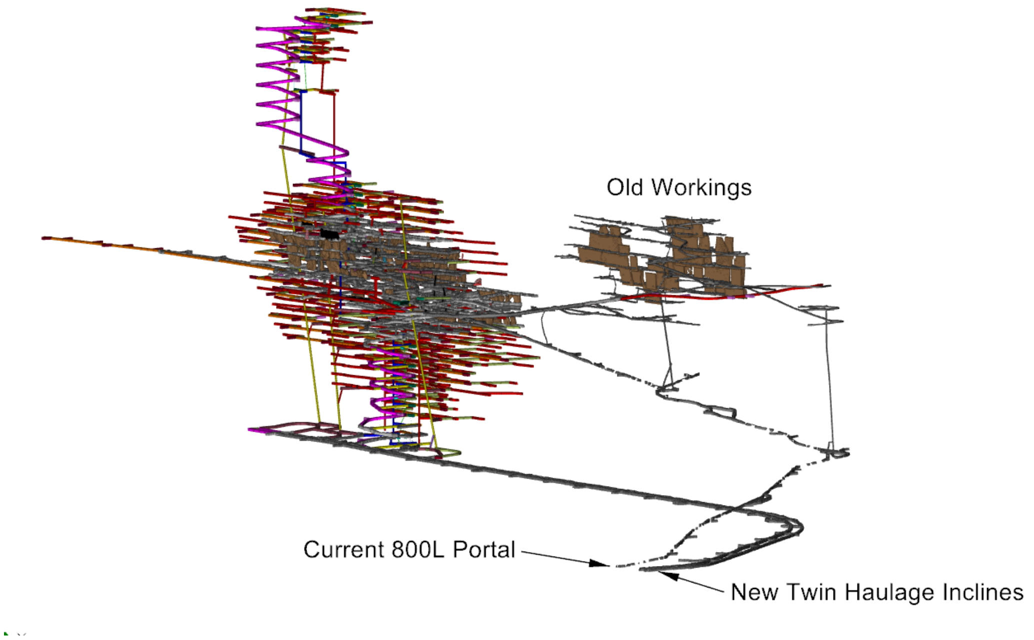


Figure 16-26 An Isometric View Looking Southwest of the Kainantu Underground Development Layout

Entech has completed development design from the current as built positions and following the current K92ML site design standards. Decline gradients (-1:7), decline standoffs to the orebody (~40 m) and other main design parameters have been maintained. Similar drive sizes have been adopted as those currently used on site which align with the current underground mining fleet. The current mining fleet should be suitable for the proposed mining methods and targeted production rate.

Lateral development is proposed to be mined by twin boom jumbos, and vertical development excavated by conventional raisebore machines in addition to mechanical slot rising with a 'Epiroc Easer L' rig. The development design includes the introduction of an ore pass network, allowing efficient material movement between the stoping areas and the twin haulage inclines which are the primary trucking access.

16.6.1 Mine Access

The mine is currently accessed via the 800 Portal which is a 4.5 mW x 4.5 mH access accommodating 45 t payload underground mining trucks. New twin haulage incline drives are being excavated, one at 6.0 mW x 6.0 mH drive size, and the other at 5.0 mW x 5.0 mH. These new drive sizes could allow future fleet size increase in the 6.0 mW x 6.0 mH (access points to the orepasses), and the 5.0 mW x 5.0 mH drive can be used as a secondary access drive, return airway, or potentially accommodate a conveyor.

16.6.2 Decline Design

The primary haulage drive will be the 6.0 mW x 6.0 mH Twin Incline drive which accesses the ore pass network, to allow for future trucking size increase, as per Figure 16-27. The other areas of the mine utilise a 5.0 mW x 5.5 mH decline drive size, with the current 800 portal and legacy incline at 4.5 mW x 4.5 mH suitable for the current 45t truck size.

The decline-orebody stand-off distance to stopes is a minimum of 70 m to the Kora lodes, and 15-20 m to the Judd lode.

For Kora lodes: This distance will minimise any damage to the decline due to ground stress changes and blasting resulting from stoping extraction. This stand-off distance will also allow sufficient space between the decline and the orebody for the excavation of the level accesses, stockpiles and sumps.

For Judd lode: The stand-off distance is too close resulting the panel at the intersection with the access to be mined at the end of mine life on retreat to minimise the impact of blasting to the decline

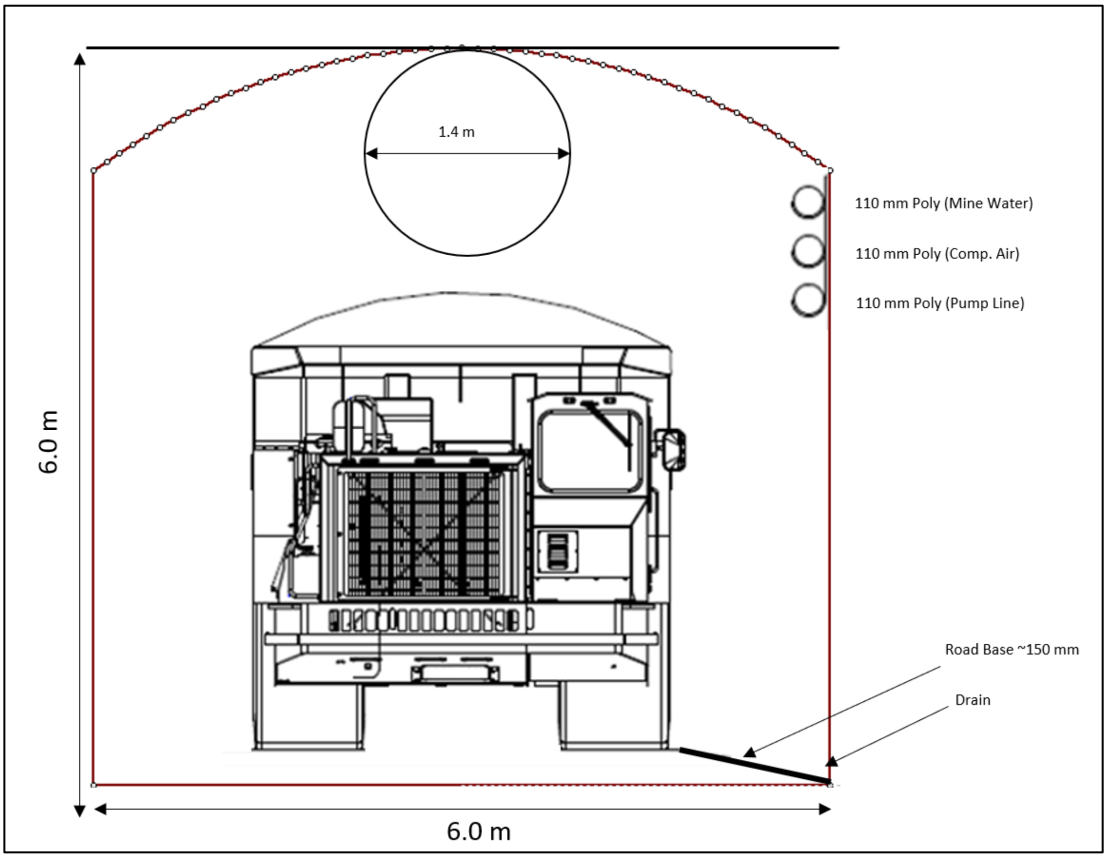


Figure 16-27 An Example of the Largest Decline Profile

The decline is designed at a gradient of 1 in 7, which is a commonly used gradient in modern underground mines and is within the operating limits of the truck. The decline will also act as the major fresh air intake. Air velocities in the decline will be between 5 and 6 m/s, which is not anticipated to be problematic for dust generation.

A 'racecourse' style decline configuration with a minimum radius of 25 m has been designed to provide optimal access to levels every 20 vertical metres whilst also allowing trucks to operate at a productive speed whilst travelling up and down the decline. Stockpiles are typically designed every 140 m along the decline, which is sufficient for high-speed development and allows for numerous locations to pass equipment and locate infrastructure items such as electrical substations and refuge chambers once the stockpile is no longer used for waste rock storage (some levels utilising Avoca).

16.6.3 Level Design

The access to each level has the same profile as the decline to enable truck travel to the level stockpiles for loading purposes as required.

A square profile stockpile has been designed on each level access at 5.5 mW x 5.5 mH and 20 m in length. The level stockpile will be used for the stockpiling of development and stope ore that is produced from the level until it can be loaded into a truck or ore pass. A typical level layout is shown in Figure 16-28.

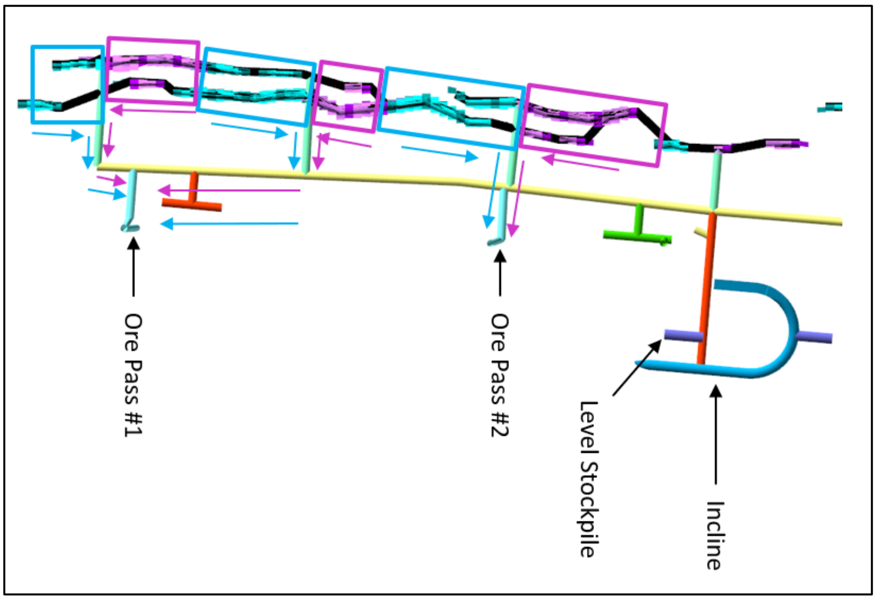


Figure 16-28 Typical Level Layout

Level footwall drives have a profile of 5.0 mW x 5.5 mH to enable haul trucks to tram as far onto a level as possible. All ore drives have a planned profile of 5.0 mW x 5.0 mH, an illustration of this profile is presented in Figure 16-29.

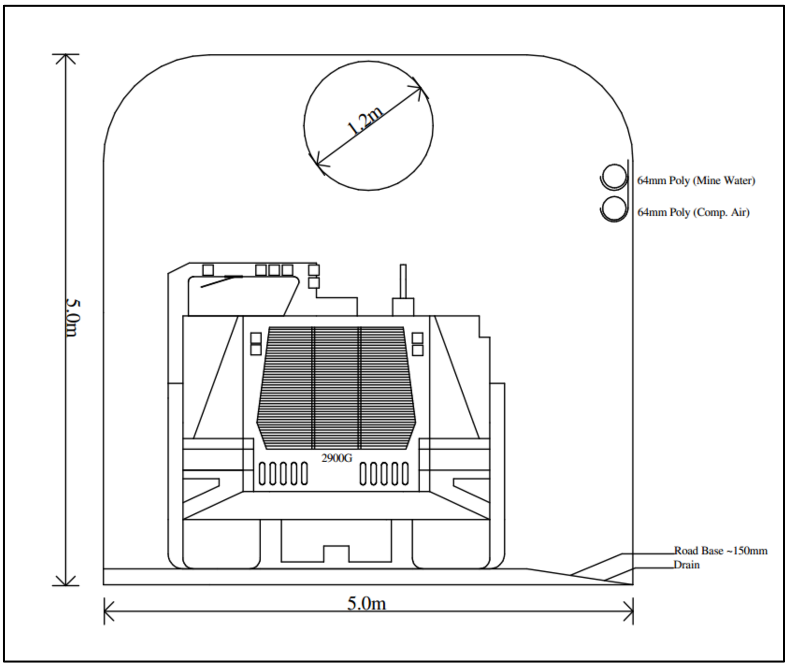


Figure 16-29 An Example of the Ore Drive Profile

16.6.4 Vertical Development

Vertical development is separated into capital and operating development. Capital development consists of ore passes, escapeway rises, ventilation return rises, and fresh air intakes. Escapeway rises are planned at 1.3 m diameter to be excavated with the 'Epiroc Easer L' rig once it is onsite in 2025. Ventilation return rises are planned as 4.0 m and 5.0 m diameter conventional raisebore excavations. Operating development consists of stope slots which use a 0.75 m diameter boxhole rise excavated with the 'Epiroc Easer L'.

16.6.5 Materials Handling

In the 2022 DFS two material handling methods were considered. The first option was all material to be hauled via truck from the extraction point where the material was blasted. This material would be loaded onto trucks via bogger. The second option considered the use of ore pass infrastructure at different locations within the mine so that blasted material would be tipped by a bogger into an ore pass and hauled from a dedicated trucking level.

A plan view of one side of a typical level layout depicting how the material movement functions for stope ore being distributed to the respective ore passes is shown in Figure 16-30.

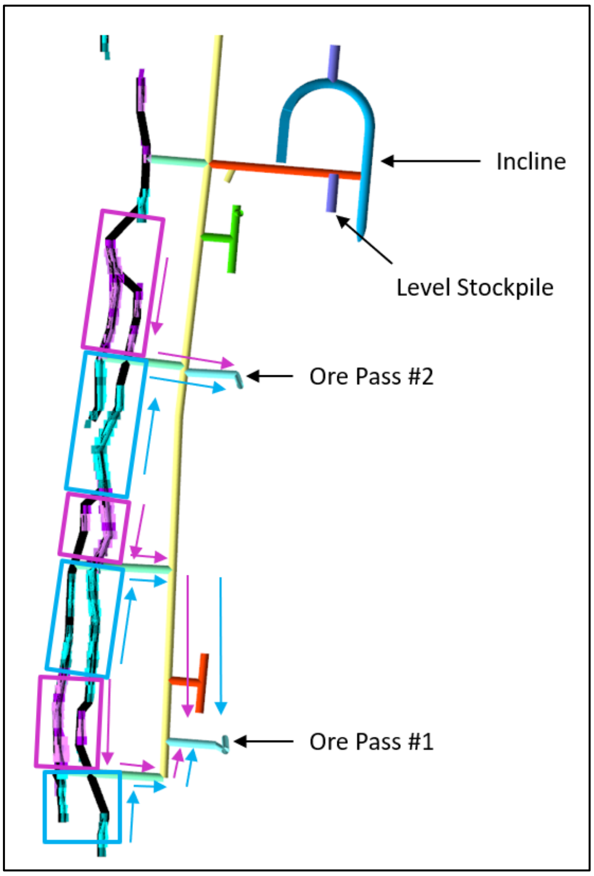


Figure 16-30 Plan View Typical Level Layout

The following points resulted in the selection of utilising an ore pass network versus trucking only. This selection has been carried over to the 2024 DFS.

- Reduction in loading and haulage TKM's.
- Increased loader productivity with shorter trams.
- Increased trucking productivity and lowered operating costs with travelling primarily through the flatter gradient haulage drives as opposed to steeper gradient decline and incline travel.
- Less vehicle interaction, increasing safety and productivity.
- Similar cost profile, where any future increase in production rate make the ore pass option more cost effective.
- Trucking congestion could become problematic if a trucking only option was used at a higher production rate.

16.7 Mining Inventory

The DFS inventory can be found detailed in the table below, with an annual material movement breakdown in Table 16-11 below.

Table 16-11 Material Movement Breakdown

		Total	2024	2025	2026	2027	2028	2029	2030
Waste	t	2,849,870	574,379	648,144	593,385	606,250	167,041	219,593	41,078
Dev Ore	t	859,371	148,626	234,127	256,026	169,059	15,353	18,142	18,038
Stope Ore	t	5,316,348	334,799	555,175	744,698	1,039,542	1,204,182	1,191,386	246,565
Ore	t	6,175,720	483,425	789,302	1,000,725	1,208,601	1,219,535	1,209,528	264,604
Ore Tonnes									
Measured	t	3,188,975	360,217	422,784	361,619	768,363	591,989	652,822	31,180
Indicated	t	2,986,745	123,208	366,518	639,106	440,239	627,546	556,705	233,424
Total	t	6,175,720	483,425	789,302	1,000,725	1,208,601	1,219,535	1,209,528	264,604
Au Grade	g/t	6.66	7.56	6.54	6.43	6.84	5.51	6.22	12.61
Cu Grade	%	0.99%	0.60%	0.55%	0.62%	1.11%	1.47%	1.10%	1.07%
Ag Grade	g/t	18.79	11.43	12.08	14.37	22.52	24.83	18.54	25.16
Metal									
Au	Moz	1.32	0.12	0.17	0.21	0.27	0.22	0.24	0.11
Cu	kt	60.98	2.90	4.35	6.21	13.40	17.97	13.32	2.84
Ag	Moz	3.73	0.18	0.31	0.46	0.88	0.97	0.72	0.21

16.8 Mine Schedule

An integrated life of mine design was prepared using Deswik.Sched® mine planning software. The software incorporates functionality to export all design and block model interrogation data to the scheduler, including volumes, tonnes, grades, and segment lengths. Graphical sequencing is exported for the critical links between all development and production activities.

The mine is planned to produce at a rate of 1,200,000 t per annum (1.2M tpa). The mine life is 7 years with a 3-year production ramp up (inclusive of 2024), sustained production of 1.2M tpa for 3 years, and final year production of 0.3M tpa.

To ensure overall mine production rates are achievable, activities associated with the mining works were scheduled in a logical sequence using the rates discussed in Section 16.8.

The major constraints on the underground scheduling were as follows:

- Ensure a smooth ramp-up to steady ore production.
- Minimise variations in development rates and production to avoid additional project costs due to under-utilisation of equipment.
- Establish capital development at an appropriate interval ahead of production.
- Stope production can only commence once the main return airway and second egress is established.

Attributed and reported development physicals include:

- Drive length (metres).
- Profile.
- Level (mRL).
- Activity type.
- Financial category (Capital / Operating).
- Mining direction (Horizontal / Vertical).
- Description (e.g. Stockpile).
- Ground support (units per metre).
- Dilution factor.
- Recovery factor.
- Tonnage.
- Grade.

- Sequence.

These properties were then used to calculate other mine physicals such as recovered tonnage and grades, equipment hours and tonne-kilometre (tkm) values required for production scheduling within Deswik and cost scheduling within MS Excel. The underground mining work was summarised into individual activities that provided the appropriate detail for DFS level mine scheduling and reporting.

The decline was divided into individual 20 m sections to become separate activities. Crosscuts were linked to the corresponding decline activity using a 'finish-to-start' constraint. In this manner, development, stoping, and backfilling activities were linked to subsequent activities.

A summary of the key mining physicals and schedule is detailed in Table 16-12 below.

Table 16-12 Key Physicals Per Annum

		Total/Av	2024	2025	2026	2027	2028	2029	2030
Lateral Development									
Decline	m	4,215	622	433	851	873	875	561	0
Other Capital	m	22,128	5,410	5,090	3,793	5,002	824	1,605	403
Ore Drive	m	18,472	2,598	5,443	5,636	3,613	301	561	320
Other Operating	m	7,199	1,461	1,228	1,748	1,513	365	681	202
Total	m	52,013	10,091	12,195	12,028	11,001	2,365	3,408	926
Vertical Development									
Return Air Rise 5.0 m	m	0	0	0	0	0	0	0	0
Return Air Rise 4.0m	m	861	180	360	47	0	174	100	0
Fresh Air Rise 5.0m	m	549	0	363	55	11	120	0	0
Ore Pass 3.0m	m	1651	169	642	361	253	152	73	0
Slot Rise 0.75m	m	14070	1065	1875	2640	2838	2281	2741	630
Production Drilling									
89mm Drill Metres	m	867,012	65,922	94,812	132,184	185,500	169,498	178,707	40,389
Haulage									
Underground	tkm	14,954,948	4,197,514	3,437,485	1,579,572	1,884,153	1,591,379	1,858,914	405,931
Ore Profile									
Ore Tonnes	t	6,175,720	483,425	789,302	1,000,725	1,208,601	1,219,535	1,209,528	264,604
Au Grade	g/t	6.66	7.56	6.54	6.43	6.84	5.51	6.22	12.61
Au Ounces	oz	1,321,494	117,547	165,927	206,912	265,893	215,967	241,963	107,286
Cu Grade	%	0.99%	0.60%	0.55%	0.62%	1.11%	1.47%	1.10%	1.07%
Cu Tonnes	t	60,985	2,904	4,348	6,205	13,396	17,974	13,320	2,838
Ag Grade	g/t	18.79	11.43	12.08	14.37	22.52	24.83	18.54	25.16
Ag Ounces	oz	3,730,264	177,624	306,436	462,490	875,085	973,741	720,849	214,039

16.8.1 Jumbo Development

The life of mine horizontal development schedule is shown in Figure 16-31.

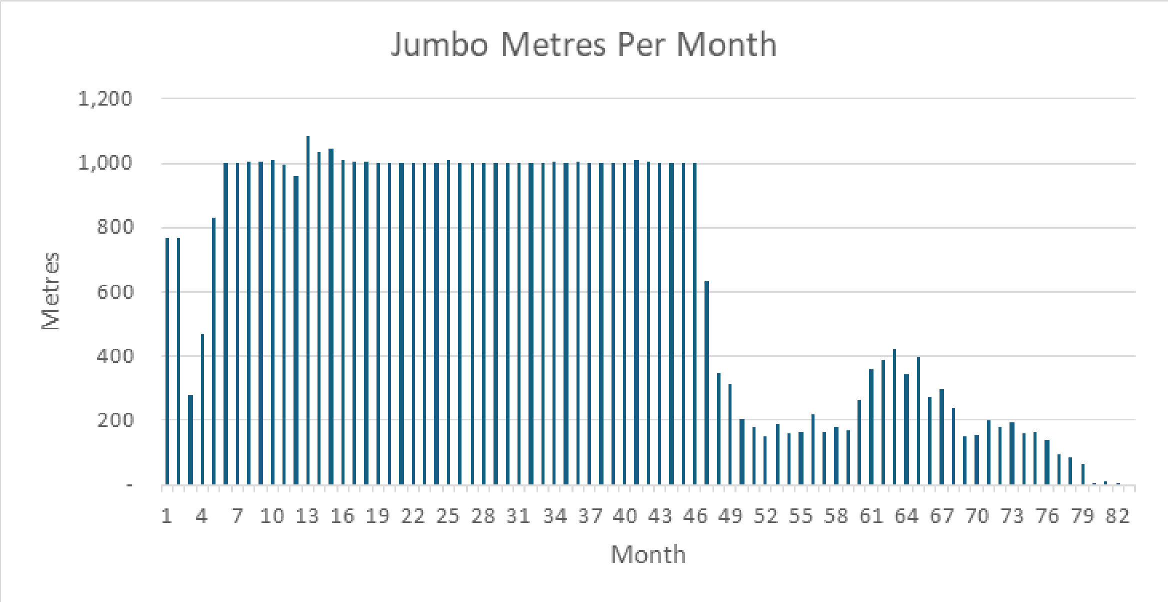


Figure 16-31 Lateral Jumbo Development Metres by Month

Advance rates have been assigned to development drives according to priority, with the remainder of a jumbo's available metres being distributed according to mining sequence.

Task priorities followed were:

- Establish primary ventilation drive / escapeway.
- Establish primary ventilation.
- Push decline to establish multiple production horizons.
- Develop ore drives as required for stoping.

16.8.2 Vertical Development

The life of mine vertical development schedule is shown in Figure 16-32.

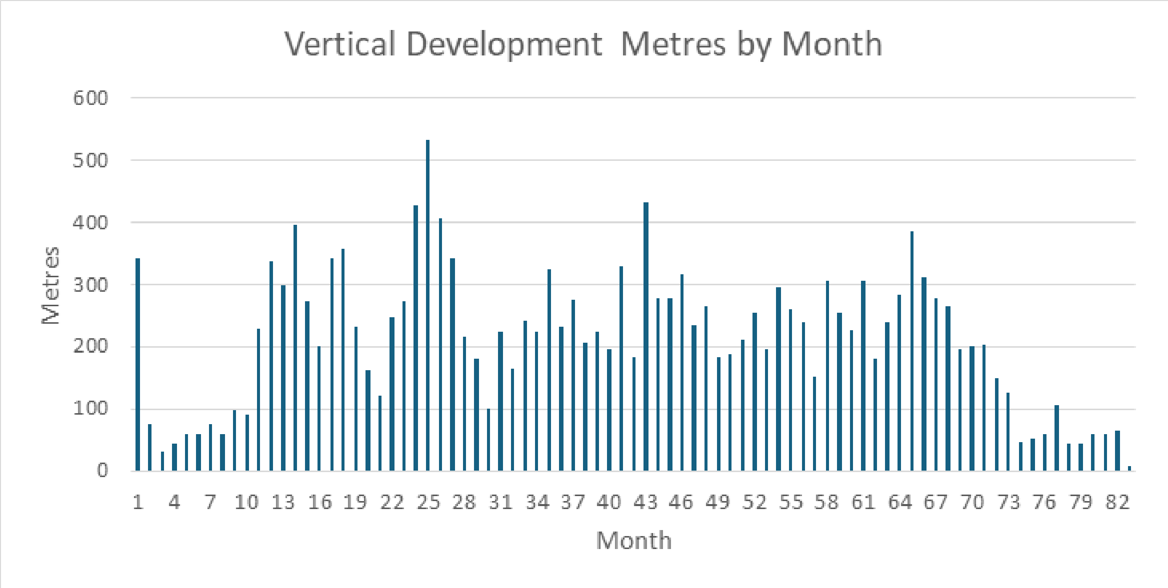


Figure 16-32 Vertical Development Metres by Month

16.8.3 Production Charging

Production charging has not been scheduled as a separate activity; however, the time required to charge each stope has been included in the stope loading time and delay between production drilling and loading activities.

16.8.4 Loading

Stoping commences in Month 1 and production ramps up to full production over a 36-month period. The life of mine loading by month is shown in Figure 16-33, and ore production profile in Figure 16-34.

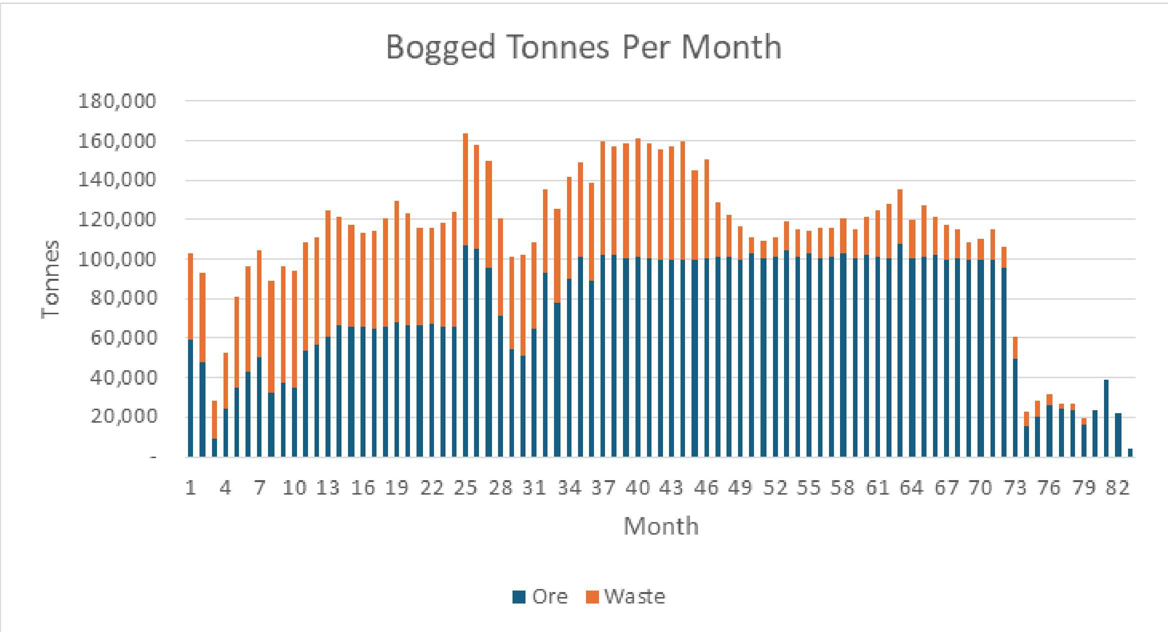


Figure 16-33 Bogged Tonnes by Month

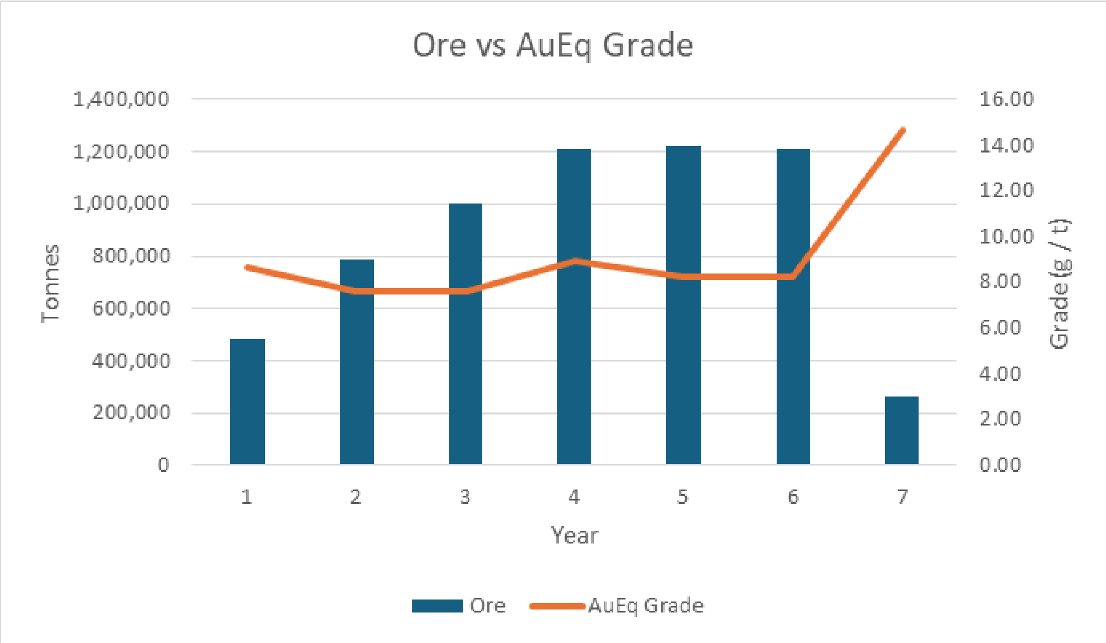


Figure 16-34 Ore Tonnes and AuEq Grade per Annum

16.8.5 Backfilling

Backfilling of stopes is carried out with paste fill. Paste filling of these stopes has been scheduled at a rate of 1,800 m^3 per day and commences upon the completion of stope loading. Figure 16-35 below shows the life of mine paste filling schedule. Note that the final year (i.e. 11 months) of paste filling was removed on request from K92ML as it is expected at the end of the mine life that stopes will be rock filled or left open.

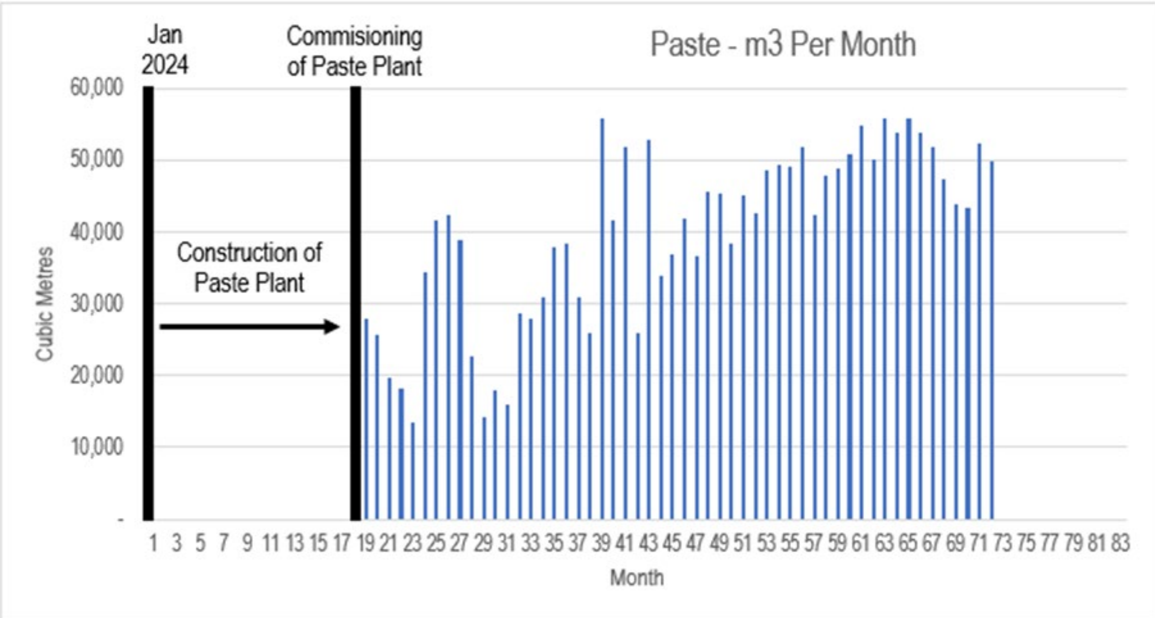


Figure 16-35 Paste Fill by Month

16.8.6 Haulage

The haulage requirement from the mine, inclusive of all ore and waste material, is illustrated in Figure 16-36 and is stated on a tkm basis. Surface haulage (including ore from the bottom of the 890L one the ore passes are completed in July 2025) is captured by a separate fleet and has been costed independently of the underground haulage cost and physicals estimates.

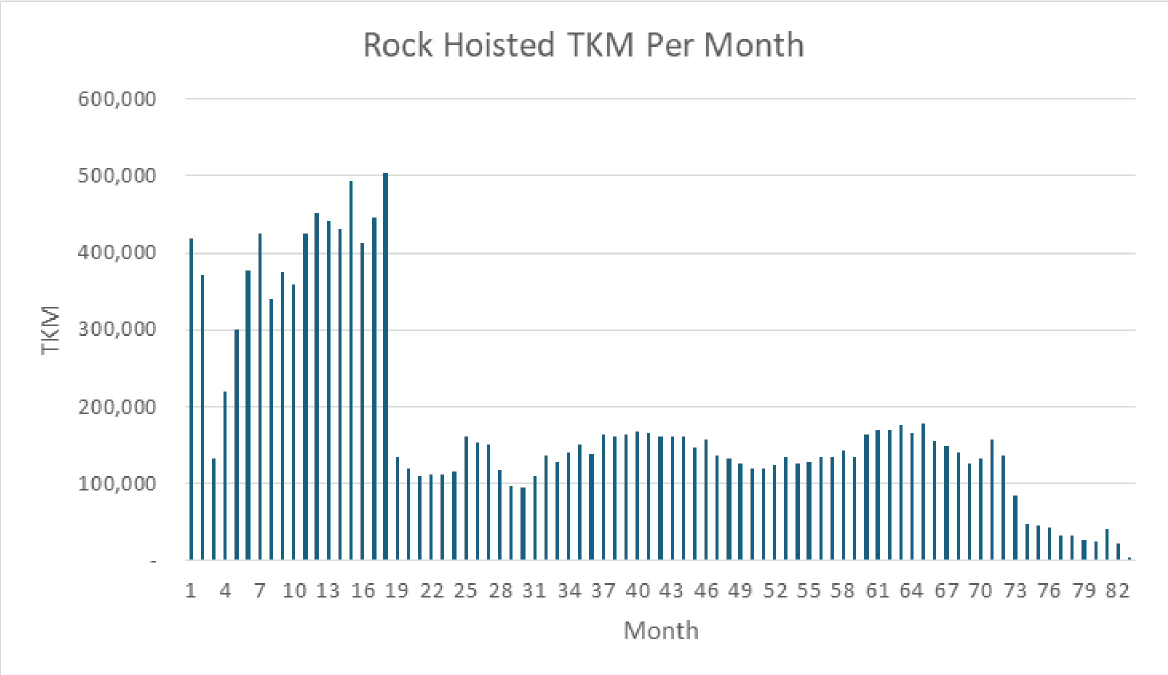


Figure 16-36 Rock Hoisted TKM by Month

16.8.7 Mine Productivity

The mine operates on 12-hour shifts, with Papua New Guinean nationals typically working on a 10-day shift, 10-night shift, 10-off roster and expatriate workers typically working a fly in fly out arrangement with a 4-weeks on and 4-weeks off roster. Productivity of scheduling tasks have been derived by Entech, with fleet requirements built up by K92ML based on site productivities. Entech have reviewed the specified fleet requirements to ensure they correlate with appropriate productivity assumptions.

16.8.8 Jumbo Development

The development productivities are based on using modern electric-over-hydraulic twin boom jumbo drills (e.g. Sandvik DD420-60 or equivalent). These drill 45 mm blastholes for development rounds and will be used for the installation of ground support.

Maximum development advance rates in all headings were set to 80 m/mo. The advance rate includes all activities and delays related to the development cycle, including drill rig up, drill rig down, face drilling, charging and firing, re-entry, loading, ground support installation, services installation, shift change and meetings, meal breaks, breakdowns, maintenance, face markup and geology/survey control delays.

Jumbo fleet requirements were previously based on K92's site productivities (150-175 m/mo per jumbo drill) but have been increased to 200-210 m/mo after assurances from K92ML that the outcomes of planned productivity improvements would start to be seen coming into 2025. The schedule has a maximum total of 1,000 m/mo jumbo development at any point in the schedule.

16.8.9 Vertical Development

A development rate of 3 m / day has been applied to all conventional raisebore vertical development. Boxhole activities completed by the Easer L machine are assumed to occur at a rate of **10 m/day.** These rates include rig up and rig down, pilot hole drilling, hole reaming and ladderway installation (where relevant).

16.8.10 Production Drilling

The production drilling requirements have been estimated by applying a calculated drill yield (ore tonnes per drill metre) to each stope's evaluated tonnage. The applied 6.0 t per drill metre yield was based on Entech's estimates from the average stope widths and is similar to those currently used on site.

A drilling rate of **250 m/day** has been applied to all production drilling activities in the schedule. These drilling rates are assumed to include all activities and delays related to production drilling, including drill rig up, drill rig down, slot drilling, production drilling, shift change and meetings, meal breaks, breakdowns, maintenance, services installation, and geology/survey control delays.

16.8.11 Stoping Cycle

Based on Sandvik LH517 loader capabilities a loading rate of **1,000 t/d** was applied in the schedule. This incorporates a time allowance for stope charging, as well as any delays, shift change and meetings, meal breaks, breakdowns, maintenance, services installation, and geology/survey control delays.

16.8.12 Backfill

Paste fill has been scheduled at approximately 106 cubic metres of paste per hour for a total of 17 hours per day. This equates to a filling rate of 1,800 cubic metres of paste per day. Once pastefill begins in Q3 2025 the tailings from the mill required to feed the paste plant will be picked up from the transfer point at the Kokomo bus stop to be taken underground by up to five CAT 730ADT trucks.

16.8.13 Materials Handling

All the ore and waste are currently planned to be hauled using conventional 45-tonne underground haulage trucks to either the ore passes or the portal transfer point respectively. Through the passes the ore will transfer to the 890L where it will be loaded into up to eight (at peak production), 60-tonne contractor Volvo A60H trucks and taken out through the new twin haulage drive to the surface ROM near the processing plant. Prior to the ore pass system being completed in Q3 2025 a total of seven trucks will be required to move the rock (both ore and waste) to the portal transfer point. This will reduce to three trucks post Q3 2025 once the ore pass system is in operation. Productivity for a Sandvik TH545 underground truck in this scenario would average 75,000 tkm/mo (tonne-kilometres per month) per unit.

16.9 Ventilation

16.9.1 Introduction

Entech Pty Ltd (Entech) was commissioned to construct a working model of the ventilation plan for Kainantu underground mine as part of the PEA (Preliminary Economic Assessment) and DFS. When a mining jurisdiction has no specific regulations or guidelines for underground mining ventilation, Entech chooses to base ventilation design parameters on those specified by Department of Mines, Industry Regulation and Safety (DMIRS) Western Australia. The scope of work covered by Entech in this report includes:

- Construct a PEA working model for Life of mine (LOM) based on ventilation design by Entech.
- Supply ventilation analysis in support of the ventilation plan's compliance with Western Australian legislation, including recommendations for primary fan design.
- Design analysis was conducted using Ventsim™ simulation software to set the mine up for the PEA mining option and that it is also able to be paired back for the DFS to show that both scenarios can be achieved within the current design parameters.

16.9.2 Model Design Parameters and Assumptions

Design parameters and assumptions used in the modelling analysis were based on guidelines provided by DMIRS Western Australia for underground mining, and industry best practice as per Table 16-13.

Table 16-13 Ventilation Design Parameters

Parameter Description	Design Parameter	Remarks
Recommended air velocity in declines to prevent dust generation.	< 6.0 m/s.	Industry best practice for preventing dust downwind of moving equipment.
Minimum air velocity provided to all working areas.	0.3 m/s.	Prescribed by Work Health & Safety (Mines) Regulations 2022 for the removal of airborne contaminants, such as heat, dust particles, diesel particulates and welding fumes.
Intake shaft velocity.	< 20 m/s (unequipped).	Guide for maintaining safe air velocities in underground workings.
Avoiding suspended water particles in wet exhaust shafts.	Prevent air velocities between 7 & 12 m/s.	Water suspension (water curtain) forms in shafts when air velocity sits within this range. This is dependent on shaft length and whether it intersects water bearing strata.
Economic upper limit for exhaust shaft velocity.	< 25 m/s.	Guide for optimising power due to excessive friction in air shafts from high air velocities.
Heat modelling parameters set to the 90th percentile of mean annual surface wet bulb (WB) temperatures, and their corresponding coincidental dry bulb (DB).	WB set to 22.6°C and DB set to 25.0°C.	Derived from long term climate data of the Kainantu district, obtained from: worldweatheronline.com/Kainantu-averages/eastern-highlands/pg.aspx
Use of the K92ML mine's elevation above sea level for setting barometric pressure.	Elevation set to 800mRL for portal.	Elevation estimated for the main haulage portals. Air density is set to this elevation.
WB design limit before applying safety measures.	28°C.	Industry best practice for the prevention of heat stress in workers.
WB stop-work upper limit.	32°C.	Industry best practice for the prevention of heat stress in workers.
Diesel exhaust dilution factor regarding flowrate in underground workings.	0.05 $m^3/s/kW$ of rated diesel engine power.	Prescribed by Work Health & Safety (Mines) Regulations 2022 as the minimum flowrate for diluting diesel emissions.
K92ML airway sizes for modelling purposes.	Main incline = 26 m^2 (arched airway). Dual haulage drives = 34 m2 & 23 m2 (arched airways). Return adits = 45 m^2 (arched airway). Return LHR = Ø4.0 m or 12.6 m^2 (RB airway) Internal fresh air intake shaft = Ø3.0 m or 7.1 m2 (RB airway). Footwall drives for primary vent = 23 m^2 (arched airway).	
Exposure standard for diesel particulates.	< 0.1 mg/m^3 TWA (measured as sub-micron elemental carbon).	Set by Work Safe Australia.

Parameter Description	Design Parameter	Remarks
Exposure standard for respirable silica dust.	< 0.05 mg/m³ TWA.	Set by Work Safe Australia.

- Design assumptions affecting modelling outputs.
- No more than five production levels operating simultaneously, requiring primary ventilation to support a loader along each footwall drive to each ore pass connection.
- No more than one operating level per ore pass at any time, requiring primary ventilation to load a truck at the haulage level.
- System leakage is minimised by replacing redundant louvres and ore pass flaps with bulkheads once production has ceased on a level.
- All remnant connections to surface are blocked except for the Puma portal.

16.9.3 Primary Ventilation

The PEA and DFS plan to ventilate K92ML underground sees fresh air entering the mine through four portals in the side of the mountain hosting the Kora deposit, with air exhausting out a large single exhaust air drive that is currently being developed. The Puma portal will be repurposed as an intake, which is currently the main exhaust for the existing workings. Figure 16-37 illustrates the PEA ventilation strategy and Figure 16-38 the DFS version without the three sets of booster fans.

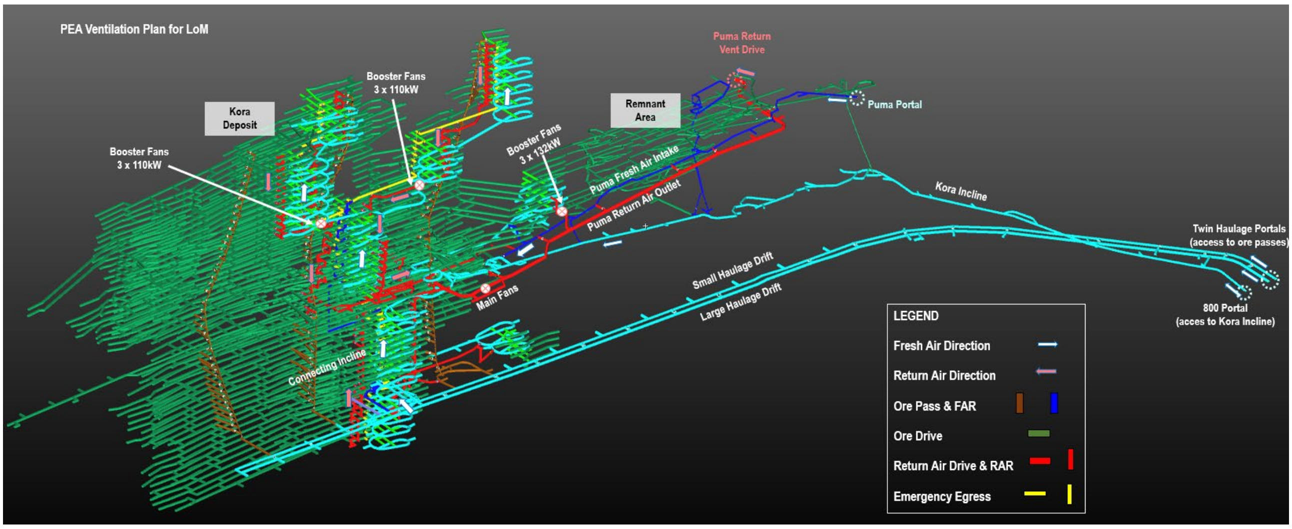


Figure 16-37 LOM Ventilation Plan Profile Facing Northwest

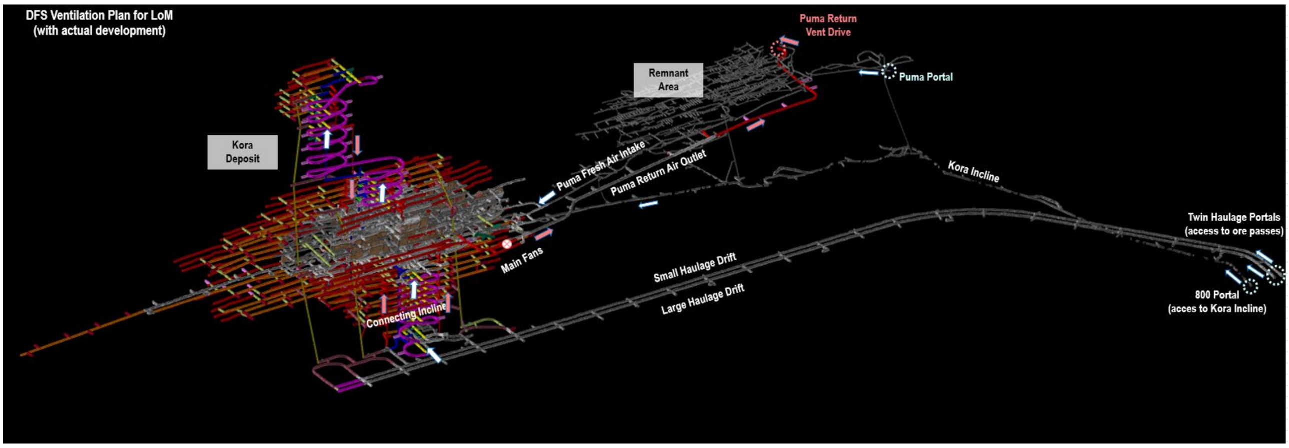


Figure 16-38: DFS LOM Ventilation Plan Profile Facing Northwest

Production haulage is confined to two horizontal side-mountain drifts, with development haulage using the Kora and internal inclines. Fresh air enters all portals and converges at the mid-point, where production is currently occurring along the Kora Incline. From here a spiral incline continues to the top of the deposit. Exhausting air links all production zones to the newly developed 6.0 mW x 6.0 mH Puma return air drive that daylights near the old workings in the remnant area.

An internal FAR will assist meeting the design parameter for decline velocities, avoiding unnecessary dust generation from mobile equipment. Where this fresh air connects to the incline, a semi open return will coincide downwind forming an air interchange that will regulate the flowrate. Figure 16-39 below illustrates the concept.

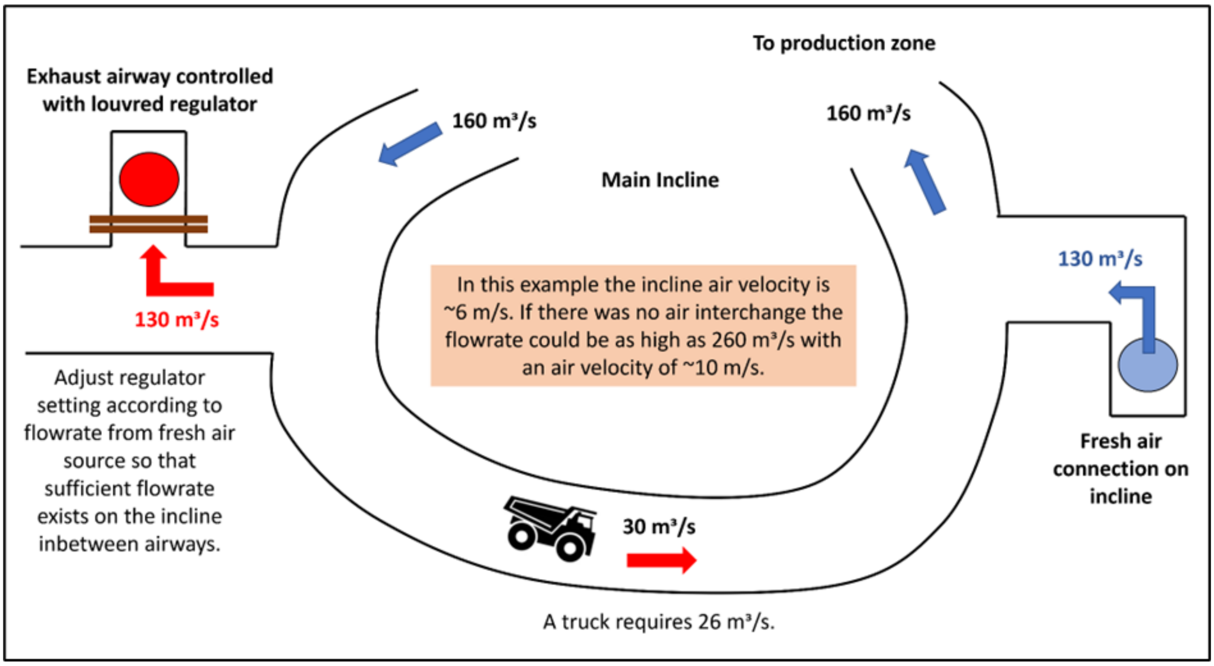


Figure 16-39 How an Air Interchange Works

The development, which connects the horizontal haulage drives with the main Kora incline, will be under the influence of an open return at the top of this incline.

Footwall drives and ore pass drives will be ventilated with primary air using automatic louvred regulators that will adjust with changes in air pressure and maintain the required flowrate in haulage routes.

16.9.4 Secondary Ventilation

Production level design will combine secondary air with primary air depending on mining activity.

Development headings will be forced ventilated with single 55 kW fans up to twin 110 kW fans (depending on the length of the drives) from the nearest flowthrough ventilation for the loader. Trucks will be loaded in flowthrough ventilation at the access to levels or on the incline.

Once production occurs in the level, the loader will only require forced ventilation beyond the footwall drive to access the stope draw point. Primary air will be drawn into the footwall drive by the return airway located up to 100 m inside the level on each side of the access, beyond this will be ore passes where the ore will be tipped into. Any dust generated from the ore pass will directly report to the return without polluting the incline. See Figure 16-40 for a typical ventilation plan.

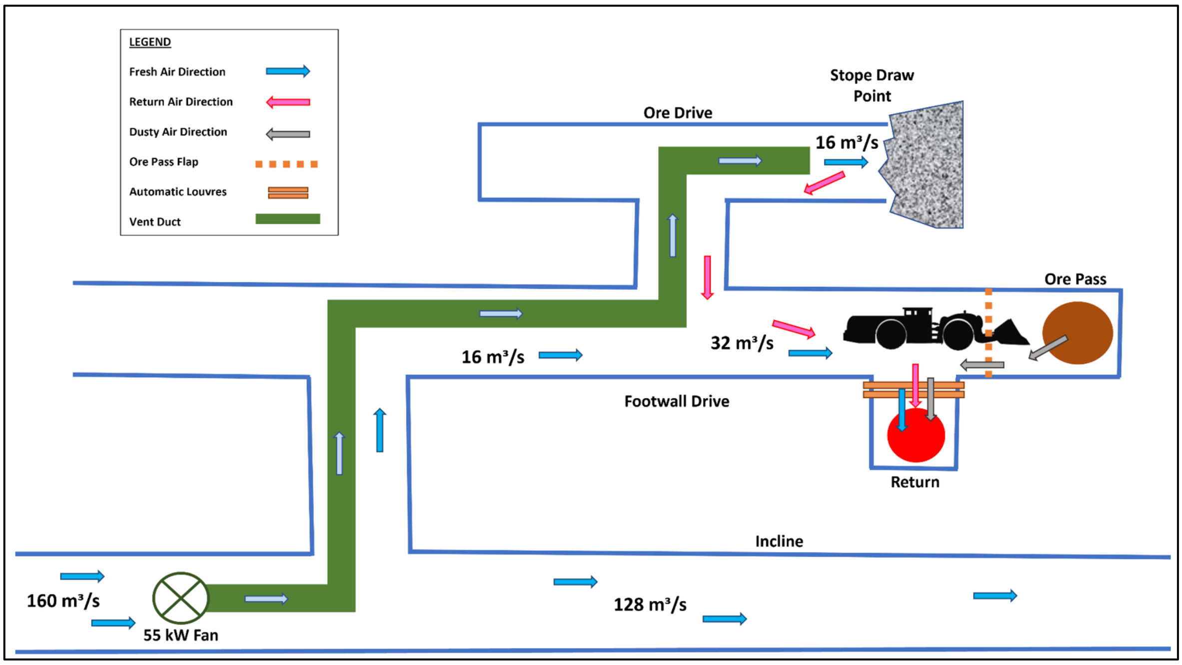


Figure 16-40 Example of a Production Level Ventilation Plan

16.9.5 Ventilation Analysis

Flowrate Definition by Diesel Exhaust Dilution

Table 16-14 and Table 16-15 contain a list of the mobile equipment intended for use by K92ML for both the DFS and PEA studies and shows the peak number of vehicles anticipated with the subsequent flowrate total. According to WA regulations, utilisation is not a factor in calculating flowrate, however an individual vehicle will need to be considered regarding the available flowrate at the time it trams between work sites. A single operator will alternate between the grader and the water cart; therefore, the grader was omitted from the total as it has the lowest rated power of the two vehicles.

Table 16-14 DFS Peak Flowrate Calculation According to the Entire Underground Mobile Fleet

Diesel Unit	Assumed Model	Engine Power Rating (kW)	Flowrate Requirement (m^3/s) *	Count	Total Flowrate (m^3/s)
Truck	Sandvik TH545	515	26	3	78
Loader	Sandvik LH517	310	16	7	112
Charge-up	Normet / Jaycon	120	6	2	12
Development Drill **	Sandvik DD421-60C	110	6	6	0
Slot Machine	Epiroc Easer L	170	6	1	0
Production Drill **	Sandvik DL431-7C	119	6	3	0
Cable Bolter**	Sandvik DS421C	119	6	1	0
Grader	12K Grader	134	7	1	0
Water Cart	Volvo A30D	242	12	1	12
Fibrecrete Sprayer	Jacon and Normet	90	5	1	5
Agitator	Jacon Transmixer	170	9	1	9
IT	Volvo L120H	203	10	3	30
LV	Toyota Landcruiser	151	8	22	176
Total Flowrate for Diesel (m^3/s)					434
Leakage @ 15%					65
Total Flowrate Including Leakage (m^3/s)					499
Activities ***					
Lowest Level Development	Auxiliary Fan	2 x 55 kW	35	1	35
Decline Development	Auxiliary Fan	2 x 110 kW	50	1	50
Total Flowrate for Activities (m^3/s)					85

Table 16-15 PEA Peak Flowrate Calculation According to the Entire Underground Mobile Fleet

Diesel Unit	Assumed Model	Engine Power Rating (kW)	Flowrate Requirement (m^3/s) *	Count	Total Flowrate (m^3/s)
Truck	Sandvik TH545	515	26	4	104
Loader	Sandvik LH517	310	16	10	160
Charge-up	Normet / Jaycon	120	6	2	12
Development Drill **	Sandvik DD421-60C	110	6	6	0
Slot Machine	Epiroc Easer L	170	6	1	0
Production Drill **	Sandvik DL431-7C	119	6	3	0
Cable Bolter**	Sandvik DS421C	119	6	2	0
Grader	12K Grader	134	7	1	0
Water Cart	Volvo A30D	242	12	1	12
Fibrecrete Sprayer	Jacon and Normet	90	5	2	10
Agitator	Jacon Transmixer	170	9	2	18
IT	Volvo L120H	203	10	4	40
LV	Toyota Landcruiser	151	8	27	216
Total Flowrate for Diesel (m^3/s)					572
Leakage @ 15%					86
Total Flowrate Including Leakage (m^3/s)					658
Activities * **					
Lowest Level Development	Auxiliary Fan	2 x 55 kW	35	1	35
Decline Development	Auxiliary Fan	2 x 110 kW	50	1	50
Total Flowrate for Activities (m^3/s)					85

**Flowrate calculated for diesel equipment at 0.05 m^3/s per kW of rated engine power.*

***Vehicle primarily operating under electric power. Flowrate allocation is given when tramming under diesel power, however, unit is omitted in the total count.*

****Minimum flowrate for activities relying on secondary air. The combination of two fans refers to activities in parallel.*

The DFS peak flowrate of 434 m^3/s for diesel exhaust dilution can be delivered to the working parts of the mine if the fans' intake draws no less than 499 m^3/s. This total incorporates the leakage value, which estimates airflow prematurely leaking through closed ventilation controls, such as louvres and bulkheads, before reaching work areas.

With an incline profile of 26 m^2, the flowrate, is limited by design parameter 1 (see Table 16-13) for velocity, will allow a maximum of 160 m^3/s in a travel way before velocities exceed 6 m/s. K92ML will need to manage the peak number of diesel units between multiple production zones in meeting the total flowrate requirement. Table 16-16 offers an example of a flowrate calculation for a production zone according to the incline velocity limit.

Table 16-16 Peak Flowrate Calculation According to Expected Mobile Fleet for a Production Area with Single Incline.

Diesel Unit	Assumed Model	Engine Power Rating (kW)	Flowrate Requirement (m³/s) *	Count	Total Flowrate (m³/s)
Development Truck	Sandvik TH545	515	26	2	52
Loader	Sandvik LH517	310	16	2	32
Charge-up	Normet / Jaycon	120	6	1	6
Development Drill **	Sandvik DD421-60C	110	6	0	0
Slot Machine	Epiroc Easer L	170	6	0	0
Production Drill **	Sandvik DL431-7C	119	6	0	0
Cable Bolter**	Sandvik DS421C	119	6	0	0
Grader	12K Grader	134	7	1	7
Water Cart	Volvo A30D	242	12	0	0
Fibrecrete Sprayer	Jacon and Normet	90	5	0	0
Agitator	Jacon Transmixer	170	9	0	0
IT	Volvo L120H	203	10	1	10
LV	Toyota Landcruiser	151	8	4	32
Total Flowrate for Diesel (m³/s)					139
Leakage @ 15%					21
Total Flowrate Including Leakage (m³/s)					160
Activities ***					
Lowest Level Development	Auxiliary Fan	2 x 55 kW	35	1	35
Decline Development	Auxiliary Fan	2 x 110 kW	50	1	50
Total Flowrate for Activities (m³/s)					85

**Flowrate calculated for diesel equipment at 0.05 m³/s per kW of rated engine power.*

***Vehicle primarily operating under electric power. Flowrate allocation is given when tramming under diesel power, however, unit is omitted in the total count.*

****Minimum flowrate for activities relying on secondary air. The combination of two fans refers to activities in parallel.*

The truck count in Table 16-16 relates to mainly development haulage as most production trucking is isolated to the haulage declines at the base of the deposit.

Benchmarking indicates that K92ML, with an annual production rate of 1.2 Mt, is representational of other hard rock mines with equivalent flowrate (see Figure 16-41).

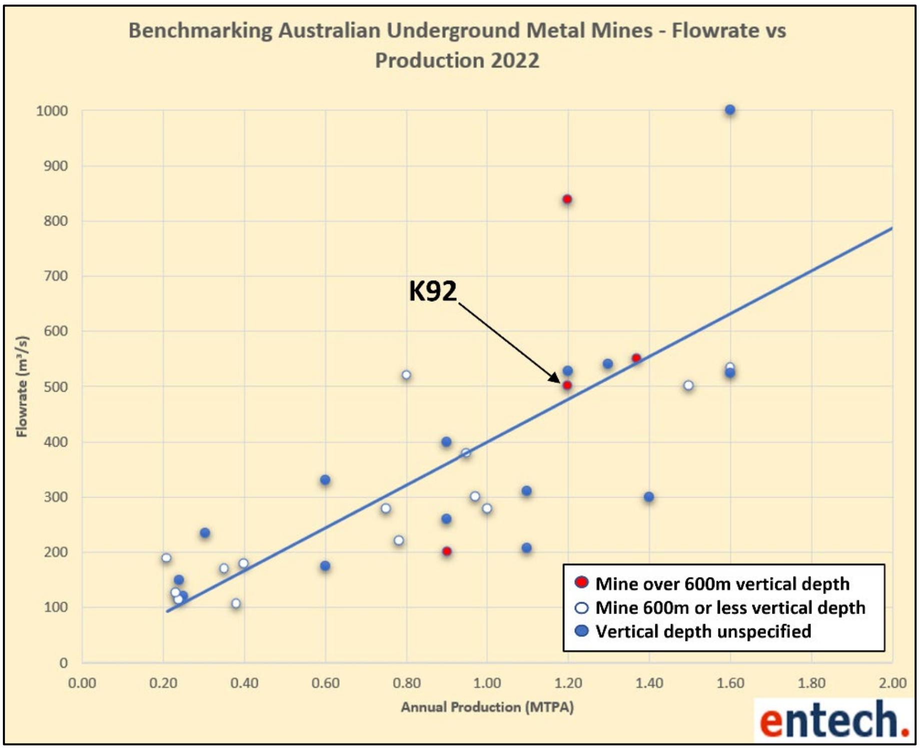


Figure 16-41 Benchmarking Kainantu with Australian Metal Mines

16.9.6 Heat Modelling

A high-level heat analysis was carried out in the 2022 DFS using the following heat settings as per Table 16-17.

Table 16-17 Heat Settings for Modelling

Setting	Input
Surface Datum Elevation	1,445 m
Surface Rock temperature	20.0°C
95th Percentile Surface Wet Bulb Temp.	22.8°C
Corresponding Surface Dry Bulb Temp.	25.0°C
Surface Relative Humidity	83.80%
Air Density	0.98 kg/m³
Geothermal Gradient	2.5°C/100 m
Rock Specific Heat	850.0 J/kgC
Rock Thermal Conductivity	1.23 W/mC
Rock Wetness Fraction	0.25

Modelling suggests that the lower half of the mine will have a wet bulb temperature 27°C to 28°C in the haulage routes with a reject wet bulb temperature of ~26°C at the topmost return airway. Although temperatures pose no elevated risk to underground workers, high surface humidity will make conditions uncomfortable.

16.9.7 Ventilation Infrastructure

Airways

The Puma exhaust drive where the main fans are located have a drive size of 6.0 mW x 8.0 mH and this was selected as it is sufficient to accommodate the peak flowrate required for the PEA vent model should the mine expand to this level. A smaller dimension of 6.0 mW x 6.0 mH would only work for the DFS vent model and so the exhaust drive was future proofed with the larger dimension whilst development was occurring. See Figure 16-42 and Figure 16-43 for a comparison.

Fixed Fow - RAD = 6.0 mW x 6.0 mH			A (m²)	Velocity (m/s)	Capacity (m³/s)	FTP (Pa)	Electric In (kW)	Diff Power (kW)	
	1	RAR	36.0	10.0	360	1,654	783	N/A	0%
	2	RAR	36.0	11.0	396	2,050	1068	285	36%
	3	RAR	36.0	12.0	432	2,491	1,416	348	33%
	4	RAR	36.0	13.0	468	2,980	1835	419	30%
	5	RAR	36.0	14.0	504	3,513	2330	495	27%
Threshold	6	RAR	36.0	15.0	**540**	4,073	2,894	564	24%
Current Fans 100% speed	7	RAR	36.0	16.0	**576**	4,691	3,555	661	23%
	8	RAR	36.0	17.0	612	5,347	4,306	751	21%
	9	RAR	36.0	18.0	648	6,043	5,152	846	20%
	10	RAR	36.0	19.0	684	6,775	6,098	946	18%

Figure 16-42 6 m x 6 m Return Air Drive Capacity

Fixed Fow - RAD = 6.0 mW x 8.0 mH			A (m²)	Velocity (m/s)	Capacity (m³/s)	FTP (Pa)	Electric In (kW)	Diff Power (kW)	
	1	RAR	48.0	7.0	336	975	431	N/A	0%
	2	RAR	48.0	8.0	384	1,339	677	246	57%
	3	RAR	48.0	9.0	432	1,769	1,006	329	49%
	4	RAR	48.0	10.0	480	2,268	1,432	426	42%
Threshold	5	RAR	48.0	11.0	528	2,819	1,958	526	37%
	6	RAR	48.0	12.0	576	3,419	2,592	634	32%
	7	RAR	48.0	13.0	624	4,111	3375	783	30%
Current Fans 100% speed	8	RAR	48.0	14.0	672	4,851	4,290	915	27%
	9	RAR	48.0	15.0	720	5,643	5,346	1,056	25%
	10	RAR	48.0	16.0	768	6,486	6,554	1,208	23%

Figure 16-43 6 m x 8 m Return Air Drive Capacity

Primary Fans

Use the following system PEA duties in Table 16-18 as a basis to match suitable primary fans for the LOM flowrate requirements for the DFS. These duties do not estimate internal fan losses, which must be stated to the fan supplier during tender. All three system duties provide the performance envelope that the primary fans must meet.

Table 16-18 System Duties for Meeting PEA Flowrate Requirement

K92 System Duties for LOM							
Staging	Mine Resistance (Ns^2/m^8)	Electrical Input (kW)	System Total Pressure (Pa)	System Static Pressure (Pa)	Inlet Density (kg/m^3)	Flow (kg/s)	Flow (m^3/s)
Development Milestone							
Peak Flowrate @ LOM	0.00011	3,389	2,268	2,164	1.01	669	673
Min Flowrate @ LOM							

Entech recommends that multiple fans in parallel are used to meet these duties. This means that the wide range in the performance envelope can be met when fan speed controls alone cannot satisfy, i.e. switching fans on and off between ventilation milestones.

Modelling suggests that two 1.9 MW axial flow fans (e.g. Zitron ZVN 1-36-1900/8) in parallel would suit the fan station. This information has been verified by the supplier within the current stage 3 expansion (S3E) project construction scope. Figure 16-44 offers an overview of what an underground fan station with the two Zitron fans in parallel would look like. Fans are connected to a bulkhead on the outlet side, which offers easy access for maintenance purposes. Fan specs represented in the image are non-specific to K92ML requirements.

The PEA vent model also requires three sets of booster fans to help push return air to the main primary fans. These fans are removed for the DFS LOM plan where the mining rate is 1.2 Mt per annum and there is much less development.

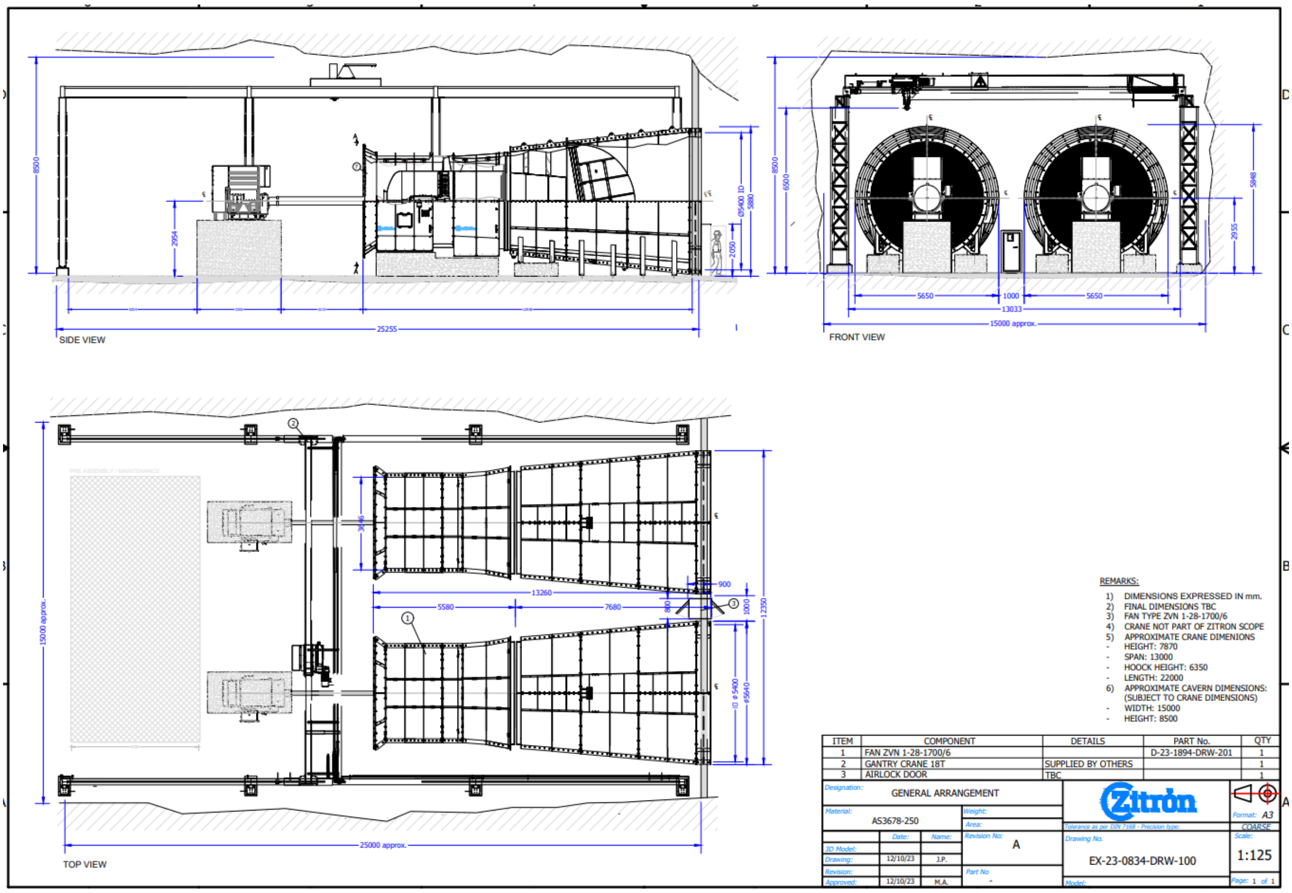


Figure 16-44 Overview of Underground Fan Station care of Zitron

16.9.8 Ventilation Control Devices

Two types of ventilation controls used in the modelling are the automatic louvred regulator and the ore pass flap. The ore pass flap is important for minimising short circuiting between levels and allows the loader free access to the pass. The automatic louvred regulator can be controlled with software and balances airflow across multiple operating levels according to changes in air pressure at the louvre. Incorporated into a ventilation on demand (VOD) system, incline flow may always be maintained to the correct flowrate.

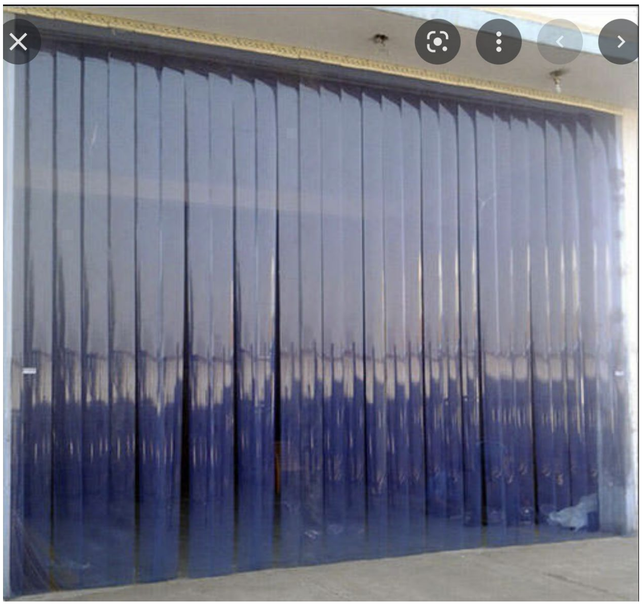


Figure 16-45 Example of Flap, or Strip Curtain that can be used at Ore Pass Access

Figure 16-46 Example of Ventilation Louvres care of Clemcorp and Wilshaw

The primary fan power across the LOM is represented in Figure 16-47 and is influenced by changes in the mine's resistance.

16.9.9 Power

The primary fan power across the LOM is represented in Figure 16-47 and is influenced by changes in the mine's activity in relation to ore production and its resistance.

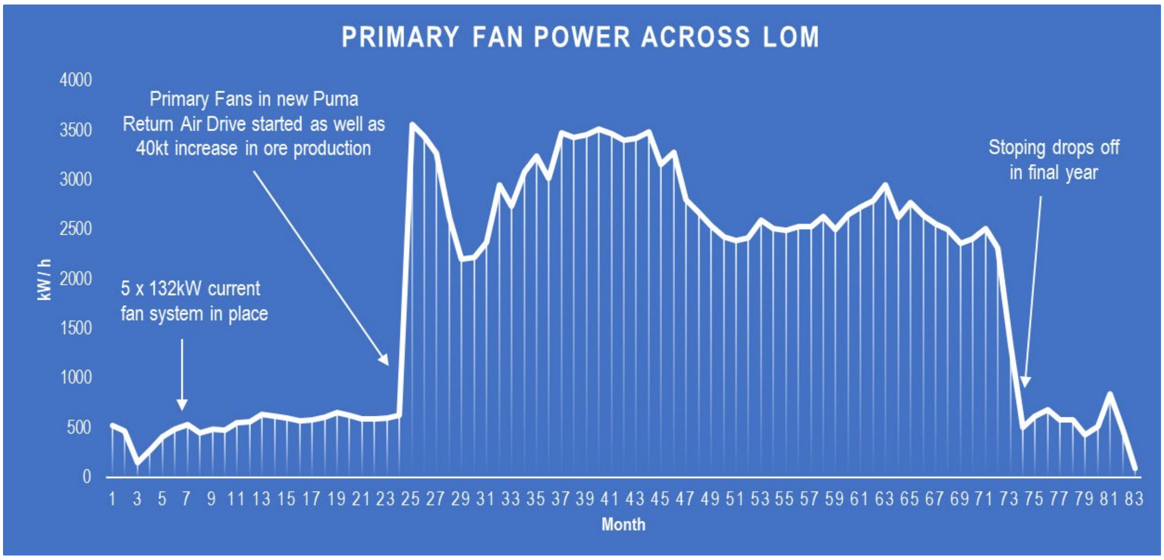


Figure 16-47 Primary Fan Power as Average kWh per Scheduled Month

16.10 Recommendations and Conclusions

Use speed controls, like VSDs, on primary fans to change the available flowrate and optimise power. This will prevent overventilation when peak flow is not required.

This data for the ventilation section has been based on the PEA ventilation model with the removal of the booster fans - as these are located in areas that are not being mined in the DFS LOM plan.

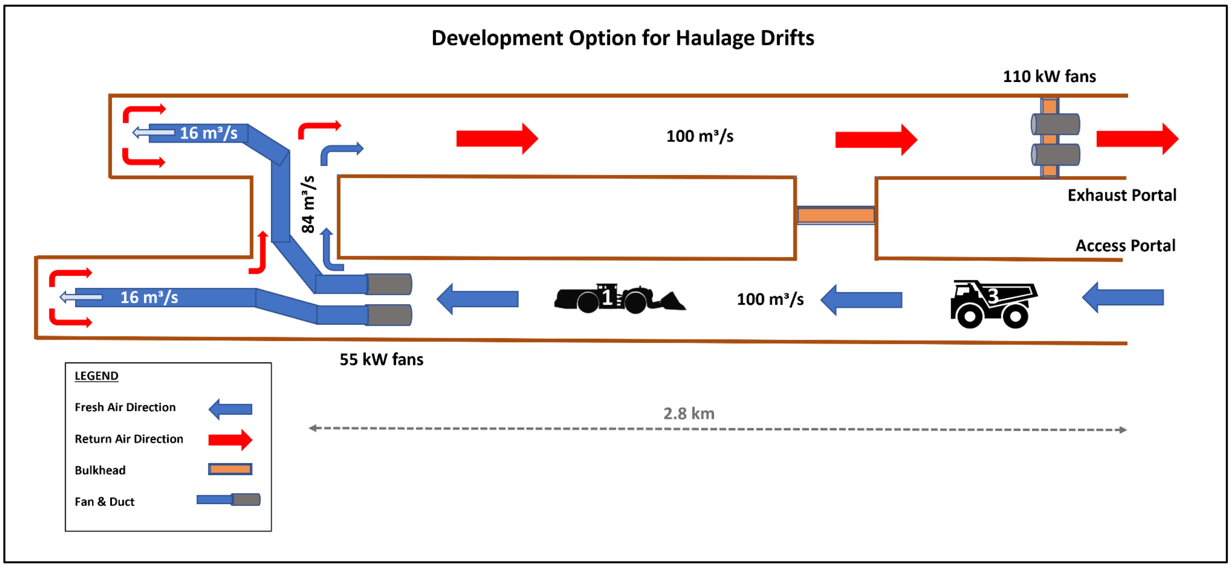


Figure 16-48 Initial Development Option for Haulage Drifts

16.11 Underground Infrastructure

16.11.1 Second Means of Egress and Refuge Chambers

A second means of egress is currently established within the mine to enable personnel to exit the mine in case that one egress (e.g. decline) becomes blocked. To ensure that all personnel can reach a decline connection drive, ladderways are installed in the raise-bored escapeway rises.

In certain cases, escaping the mine via a second means of egress in an emergency may not be possible or may not be safe. Some examples would be a mobile equipment fire that creates large amounts of smoke, which would prevent the use of the escape ladderways or blockage of a single-entry heading preventing personnel in that heading from escaping the drive.

In order to provide refuge for underground personnel in such circumstances, re-locatable refuge chambers are installed in the mine. The chambers vary in size from four to 20-man capacity. 4 to 8-man portable, battery-powered refuge chambers will be used throughout the mine development phase; these chambers are easily moved by an ITC (integrated tool carrier) into single-entry headings where there is the risk of entrapment (e.g. when a jumbo operator is working at the decline face with trucks being loaded between the jumbo and the nearest access to an escapeway, thereby creating an entrapment situation). 12-man and 20-man chambers are used in other parts of the mine. The chambers will be purchased and installed progressively as the mine is developed deeper.

Refuge chambers are located at approximately 100 m vertical intervals throughout the mine, and based on the fully developed mine, a possible arrangement of refuge chambers has been illustrated in Figure 16-46, which also depicts the escape routes to surface via the portals.

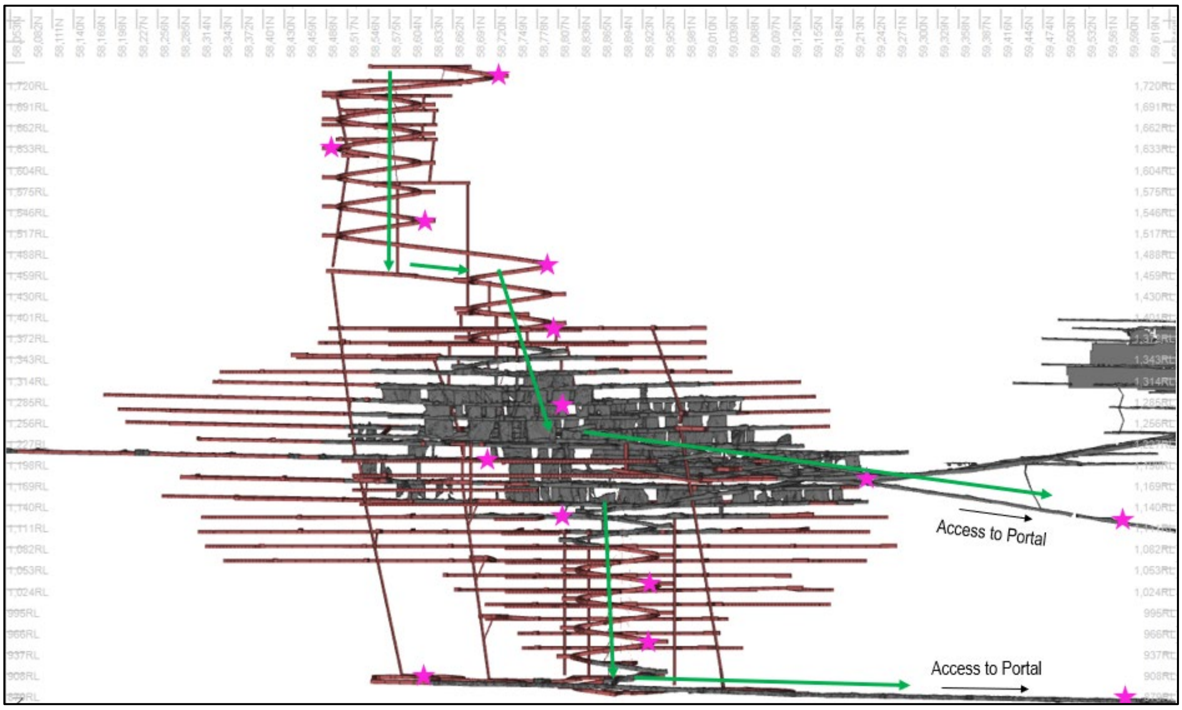


Figure 16-49 Proposed Underground Refuge Chambers (Purple) and Escapeway Route (Green arrows)

The actual locations of the chambers will change over time and will be dependent on the numbers of personnel typically working in each part of the mine.

The total number of personnel working in each part of the mine is limited to the refuge chamber capacity and will be controlled by tag board procedures.

16.11.2 Electrical Power

11 kV power is reticulated from the portal into the underground workings. From here the high voltage power is distributed to a series of 2MVA 11 kV / 1 kV step-down transformer sub stations. High voltage power is planned to be reticulated via dedicated service holes or other infrastructure such as fresh air rises. From the underground substations, 1 kV power is then reticulated to on level distribution boards, which will provide power for equipment, and infrastructure.

A depiction of the underground substation placement is shown in Figure 16-47. The number of substations has been estimated using an average of range of 100 m - 150 m vertical distance between sub stations, with some levels requiring two sub stations along strike given the strike length of the orebody and the associated voltage drop that occurs over these distances. A limit of 1,000 m was used in terms of distance from the sub-station before the voltage drop would be too high.

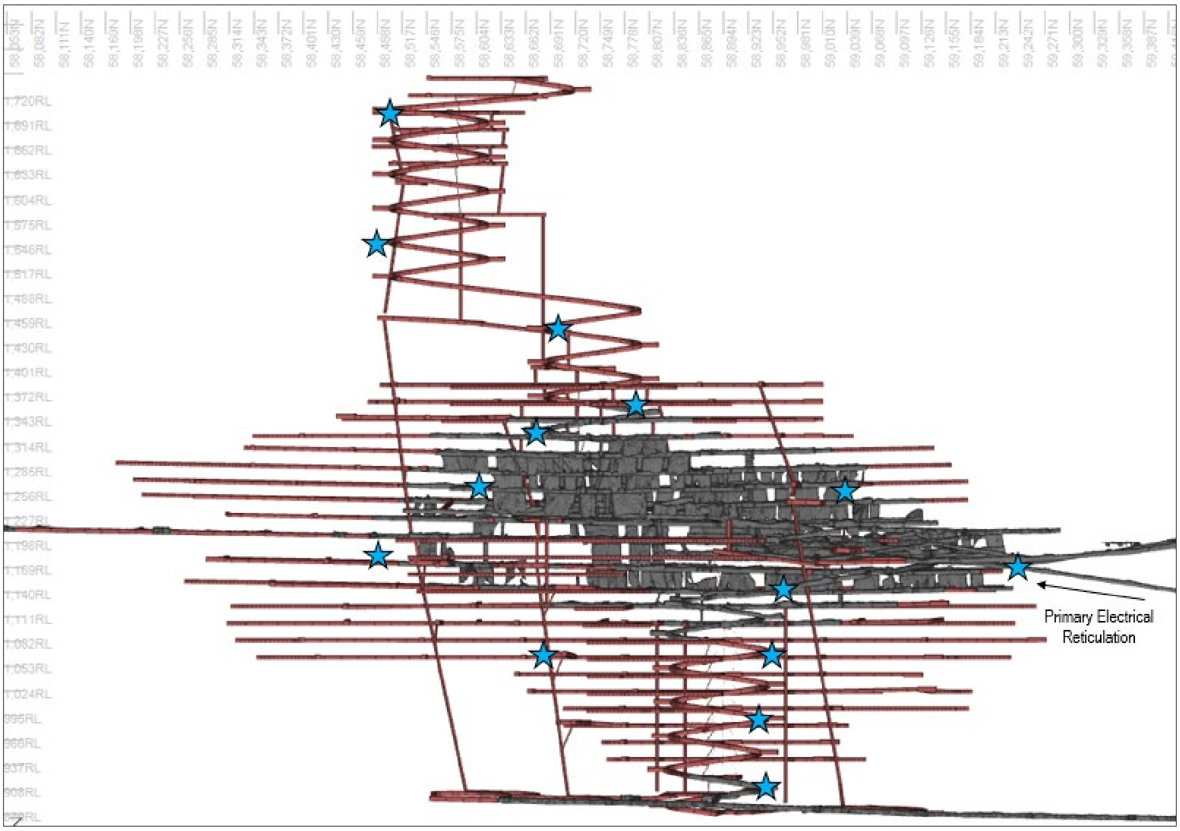


Figure 16-50 Estimated Underground Power Substation Network (Star Locations Depict Potential Placement of 2MW Substations)

A site power upgrade is planned to meet the anticipated peak power demand of approximately 6.9MW - not including the paste plant.

16.11.3 Service Water

Water captured by the dewatering system is settled and a portion is recycled for use at the underground mine. It is managed internally where it is captured and distributed to working areas.

16.11.4 Hydrogeology and Dewatering

The following is an excerpt from the 'Kora Hydrogeology Assessment' completed by EMM in December 2021:

"Hydrogeology conditions are highly heterogeneous and primarily controlled by local topography, local recharge zones and mineralisation-associated fracture trends. Locally groundwater is directly recharged via the infiltration of rainfall. Infiltration is enhanced at outcropping fracture zones, streams and anthropogenic (mine-related) features. The Kora-Kainantu fractured rock aquifer is stress-dependent, with rockmass-relaxation in the vicinity of excavations resulting in increased permeability. Higher groundwater pressures are observed in the south-eastern side of Kora, due to topographic head contrasts.

Dewatering of Kainantu underground workings is aided by topography and favourable mining-progression. The elevation of the mine access portal is below the current resource, facilitating gravity-driven mine-dewatering. Historical mining in the Irumafimpa region occurred at a higher elevation and was accessed via higher elevation portals. Subsequent mining operations have been accessed via the '800 Portal', from where all mine discharge currently exits.

Observations of mine discharge volumes indicate dewatering requirements have increased. In 2015 mine discharge was recorded at approximately 30 litres per second (L/s). In comparison, mine discharge from 2020 to 2021 is understood to have been in the order of 200-225 L/s. About 100 L/s of mine-discharge is sent to the ore processing facility, the remainder is discharged to a local creek via 'twin weirs' and associated settling ponds.

Dewatering volumes in the order of 200-225 L/s are expected for the Kora mine-expansion. Recorded data indicate a seasonal component to dewatering discharge. A 'baseflow' of about 75-100 L/s occurs year-round with an additional 100-125 L/s in the wet season, attributed to enhanced aquifer recharge. Dewatering volumes are likely to reduce if surface water recharge is regulated, including stream-diversions in areas connected to underground workings and other high-infiltration areas.

Mining operations include underground pumps and pipeline infrastructure in the existing incline's backs. Additional dewatering infrastructure will be required for mining operations below 1,185mRL. This will include an additional progressive cavity sump pump(s) and transfer pipelines. Pumps will be required to lift a maximum head of about 300 metres (m); however, multi-stage, lower-head pumping systems could also be employed.

The new twin inclines were completed in 2023, changing the nature of dewatering. The twin incline will intersect the Kora resource area at approximately 900mRL, this will become the new focus of mine-discharge collection and transfer to the 800 Portal (largely negating the need for pumping to the 1,185mRL sump/dam). However, pumping infrastructure will be required for Kora mine-development below the 900mRL. Risks associated with uncontrolled groundwater discharge will increase as underground mining depths increase.

Mining operations show evidence of acid rock drainage (ARD). The majority of ARD to date is likely to have been generated within the historical (Irumafimpa) underground workings. This apparent ARD will produce low pH conditions, elevated salinity and elevated concentrations of dissolved metals in discharged water. ARD risks may comprise:

- Environmental impacts on downstream riparian environments.
- Increased concentrations of salinity and dissolved metals.
- Human health impacts from potential mine workers' direct skin-contact (skin rashes, other impacts).
- Downstream impacts on potable water supplies.

Paste-filling of mined workings will occur after the paste fill plant is completed in July 2025. Paste backfilling will reduce mine-throughflow, and therefore, slow the effects of ARD. Paste backfilling is planned for Kora mining regions; paste backfilling may also be required in historical (Irumafimpa) underground workings to reduced ARD. 'Mining upwards' paste backfilling (whereby mine voids and subsequent paste backfilling occurs above a region that has been previously paste-backfilled) will result in superior ARD-mitigation outcomes. Additional ARD mitigation measures may also be required, such as dosing of discharge water with ARD-neutralising material (for example, calcium hydroxide ($Ca(OH)_2$ [hydrated lime / slaked lime]).

ARD will likely form a key feature of mine-closure planning. Mine-closure planning could be compromised if illegal mining operations were to occur beyond closure. Paste-backfilling is likely to have additional benefits with respect to reducing opportunities for illegal mine access and disturbance of mine-closure measures.

The Papua New Guinea Conservation and Environment Protection Authority (CEPA) has issued K92ML with Environment Permit EP-L3(34). This permit's conditions include a limit on discharge volumes from mine dewatering of 7.2 giga litres (227 L/s). Current operations appear to be within this limit; however, existing data are insufficient to properly assess the existing and future operations' compliance with these conditions. Environment Permit EP-L3(34)'s mine-discharge water quality conditions focus on major physical parameters and metals, as well as dissolved oxygen, suspended solids and oil and grease. The apparent ARD suggests that these limits may be exceeded.

Project risks are summarised in a risk matrix, including risk-mitigation measures and further works. Requirements for additional data include:

- Installation of flow meters to monitor mine-discharge volumes.
- A groundwater sampling program to characterise ARD discharge (a sampling program is provided in this report).

- Tracer tests to identify recharge locations and pathways.
- Hydrogeological mapping and monitoring of underground workings.

16.11.5 Primary Pumping

The underground mine dewatering strategy utilises the gravity advantage of having the mine portal at a lower elevation than the currently mined areas of the deposit. Mine water is pumped via secondary pumping, to a central sump location at the 1185mRL. From here the water is discharged via the pipes as shown in Figure 16-51 Overview Dewatering System where it reaches the portal and discharge to a weir system where it is treated and subjected to water monitoring.

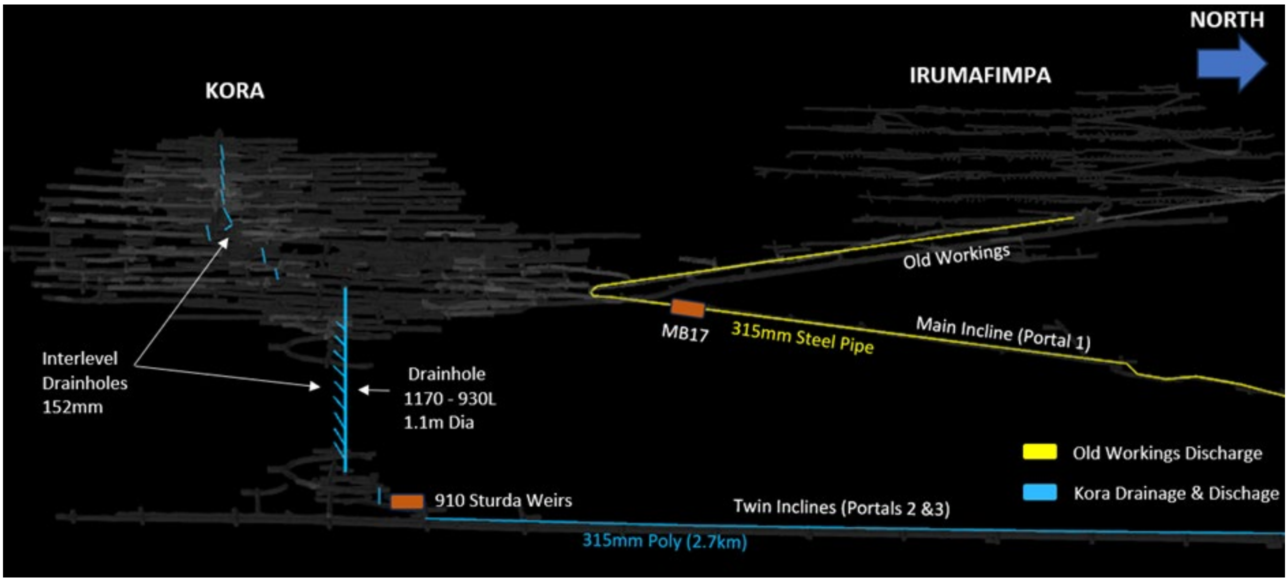


Figure 16-51 Overview Dewatering System

16.11.6 Secondary Pumping

The secondary dewatering system includes 8 kW, 20 kW and 37 kW electric submersible pumps (e.g. Flygt pumps) and a small number of re-locatable helical-rotor pumps (often referred to as 'travelling monos'). These secondary pumps will transfer water from sumps located in the active headings to the 1185mRL, until the final system is established at the completion of twin haulage drives.

16.11.7 Compressed Air

There is currently one compressor in place that supply compressed air to the underground. This will be increased to two compressors in July 2025 to account for future equipment levels.

16.11.8 Underground Communications

A leaky-feeder UHF (ultra-high frequency) radio system provides the primary means of communication within the mine and is already installed underground. The key components of the system comprise:

- Radio head-end unit located on the surface.
- Leaky-feeder cable installed throughout the mine (and associated amplifiers).
- UHF radios installed on all mobile equipment and some fixed equipment.

The system can communicate with users of the surface UHF system (e.g. mill control, surface operations) via a dedicated surface-underground channel.

16.12 Underground Operations

16.12.1 Mining Fleet

Assumptions for mine equipment requirements have been derived by K92ML based on actual site productivities and first principles' calculations. The listed equipment types have been reviewed by Entech and are appropriate for the mine design and schedule developed by Entech. Table 16-19 provides a summary of the peak number of units required for mine development and production.

This represents the equipment necessary to perform the following duties:

- Excavate the lateral and decline development in both ore and waste.
- Install all ground support including rockbolting and surface support.
- Maintain the underground road surfaces.
- Drill, charge and bog (including remote loading) all stoping ore material.
- Drill slot rises for production stoping.
- Install all underground services for development and production.

Table 16-19 Peak Fleet Numbers

Equipment	Quantity
Primary	
Development Drills (DD421-60C & DD422i-60C)	6
Loaders (LH517i & LH621i)	7
Trucks (AD45B, TH545')	7
Production Drills (DL421-15C/DL431-15C/DS 421C Bolter)	4
Slot Drill (Easer L)	1
Production Charge-up (Charjet)	2
Development Charge-up (Emulsion Vehicle and MC100 Emulsion Truck)	2
Ancillary	
Spraymec (Maxi-Jet MX3 & Maxi-Jet Sprayer)	3
Agitator (Transmixer X5 & Transmixer 5000)	2
IT CLG856H Development	3
IT CLG856H Production	3
12K Grader	1
Scissor Lift (Ultimec)	1
Light Vehicles	44
Raisebore (Rhino 400H)	1
Other Trucks	
Ore and Waste Surface Trucks (A60H)	8
Tailings Backhaul for UG Paste Plant Trucks (730ADT)	5

16.12.2 Personnel

The Personnel requirements have been calculated based on the equipment requirements for underground production. The proposed Personnel requirements are based on a mix of Papua New Guinea national employees and expatriate employees.

The typical roster for the national employees is 20 days on site and 10 days off, with a roster of 4 weeks on and 4 weeks off for expatriate employees.

The total number of employed personnel in the underground operation at peak production reaches approximately 561, with all positions detailed in Table 16-20, Table 16-21, Table 16-22, and Table 16-23.

Table 16-20 Total Mine Technical Services Personnel

Mine Technical Services	Quantity
Head of Mine Technical Services	1
Manager - Mine Technical Services	1
Superintendent - Geotechnical	1
Senior Specialist - Mining 1	1
Senior Specialist - Mining 2	1
Superintendent - Geology	1
Senior Specialist - Survey	1
Senior Specialist - Mining 3	1
Officer - Administration 2	2
Senior Engineer - Mining 2	2
Engineer - Mining 2	4
Technician - Mine Ventilation	4
Senior Engineer - Drill & Blast	1
Junior Engineer - Mining Operations	2
Graduate - Mine Engineer 2	2
Senior Engineer - Geotech	1
Engineer - Geotechnical	2
Surveyor - Mine (Registered)	1
Senior Surveyor Mine	2
Surveyor - Mine	4
Junior Surveyor - Mine	4
Assistant - Survey	6
Superintendent - Geology	1
Senior Geologist - Mine	4
Senior Geologist - Database	1
Geologist - Database	2
Geologist - Mining	6
Senior Field Technician	3
Junior Geologist - Mine	4
Graduate - Geology	2
Field Technician 1	16
Field Assistant	4
Field Technician 2	4

Table 16-21 Mining Overheads Personnel

Mining Overheads	Quantity
Head of Mining	1
Manager - Mine	1
Senior Specialist - Mining	2
Superintendent - Mining	2
Coordinator - Administration	1
Coordinator - Stores	1
Superintendent - Training	1
Coordinator - Training & Development	1
Cost Controller	2
Lamp Room Operator	4
Engineer - Mine Dispatch	3
Tool Storeman 1	2
Cleaner 1	4

Mining Overheads	Quantity
Officer - Administration 1	5
Engineer - Mining 1	2
Senior Engineer - Mining 1	2
Snr Officer - Admin	1
Trainee - Truck	2
Trainer - Mine UG 1	3

Table 16-22 Mining Operations Personnel

Mining Operations	Quantity
Superintendent - Mine	4
Foreman - General Production	3
Shift Boss - Mine	3
Driller - Longhole	9
Operator - Charge up 1	9
Heavy Equipment Operator 1	3
Multi Skilled Miner - Specialist 1	2
Multi Skilled Miner - Charging	4
Long Hole Drill Operator	6
Cable Bolt Drill Operator	1
Operator Truck 1	11
Loader Operator 1	21
Foreman - Mine (Development)	3
Jumbo Operator Driller	13
Multi Skilled Miner - All Rounder	4
Multi Skilled Miner - Live Mine	2
Multi Skilled Miner - Shotcreter	1
Operator Truck 3	11
Operator - Charge up 2	6
Operator - Charge up 3	6
Jumbo Offsider	18
Jumbo Operator	10
Loader Operator 2	3
Nipper 1	6
Shotcreter	9
Supervisor - Shotcrete	1
Foreman - Mine (Services)	3
Multi Skilled Miner - Specialist 2	1
Agi Driver - Shotcrete	6
Technician - Boilermaker UG	3
Construction Crew	6
Operator - Grader	2
Heavy Equipment Operator 2	3
IT Loader Operator (Prod IT)	9
Leading Hand - Service 1	3
Servicecrew (Dev IT)	9
Operator - Batch Plant	3
Driver - Manhaul	3
Heavy Equipment Operator 3	3
Servicecrew	3
Coordinator - Mobile Maintenance UG 1	1
Operator - Grader UG	3

Mining Operations	Quantity
Serviceman - Roadway UG	3
Operator Truck 4	3
Plant operator	3
Superintendent - Mobile Maintenance (UG) 1	3
Supervisor - Drilling (Vertical)	2
Driller	3
Technical Specialist - Maintenance	2
Raisebore Offsider	3

Table 16-23 Maintenance Personnel

Maintenance	Quantity
Head of Maintenance	1
Snr Specialist - Maintenance	2
Technical Specialist - Maintenance 1	1
Technical Specialist - Maintenance (Automated)	2
Technical Specialist - Maintenance (Workshop)	2
Technical Specialist - Maintenance 2	4
Superintendent - Mobile Maintenance (UG) 2	2
Officer - Administration 3	2
Coordinator - Mobile Maintenance UG 2	2
Maintenance Planner	2
Coordinator - Mobile Maintenance UG 3	3
Supervisor - Diamond Drill (UG) 5	3
Supervisor - Auto Electrical (UG)	3
Supervisor Development & Haulage	3
Trainer - Mine UG 2	3
Leading Hand	2
Technician - Auto Electrical UG	3
Technician - Auto Electrical	6
Technician - Fitter (Automated) UG	3
Technician - Fitter (HEF) 1	21
Technician - Boilermaker	3
Technician - Plant Fitter	3
Senior Tyre Fitter	3
Technician - Hose UG	3
Trade Assistant - HEF 1	12
Trade Assistant - HEF 2	6
Tool Storeman 2	3
Operator - Fuel/Oil Farm	3
Cleaner 2	2
Superintendent - Electrical (UG)	2
Coordinator - Electrical UG	2
Supervisor - Electrical (UG)	3
Technician - Electrical (UG)	9
Technician - Pump Fitting	3
Technician - Radio (UG) 1	1
Technician - Radio (UG) 2	3
Trade Assistant - Electrical (UG)	6
Head of Asset Management - K	1
Senior Maintenance Planner 1 - K	1
Technical Specialist - Maintenance (Workshop) - K	1

Maintenance	Quantity
Senior Specialist - CRC Maintenance - K	1
Maintenance Planner 1 - K	2
Engineer - Maintenance - K	1
Officer - Administration 1 - K	1
Officer - Administration 2 - K	2
Trainee - Pre-Vocational - K	5
Coordinator - Workshop - K	2
Supervisor - Workshop - K	2
Supervisor - Ancillary - K	1
Supervisor - Auto Electrical - K	2
Supervisor - Welding - K	2
Technician - Auto Electrical 1 - K	3
Technician - Fitter (HEF) 1 - K	4
Technician - Fitter (HEF) 2 - K	2
Technician - LV Mechanic - K	6
Technician - Fabricator - K	2
Supervisor - Tyre Fitting - K	2
Tyre Fitter - K	2
Technician - Panelbeating - K	2
Technician - Boilermaker 1 - K	2
Tool Storeman - K	2
Trade Assistant - HEF 1 - K	2
Trade Assistant - LV Mechanic - K	2
Senior Trade Assistant - Rock Drill Mechanic - K	1
Technician - Fitter (HEF) 3 - K	3
*K = Kumian	

Peak underground Personnel requirements (total employed personnel, not all on site simultaneously) by month are found in Figure 16-52.

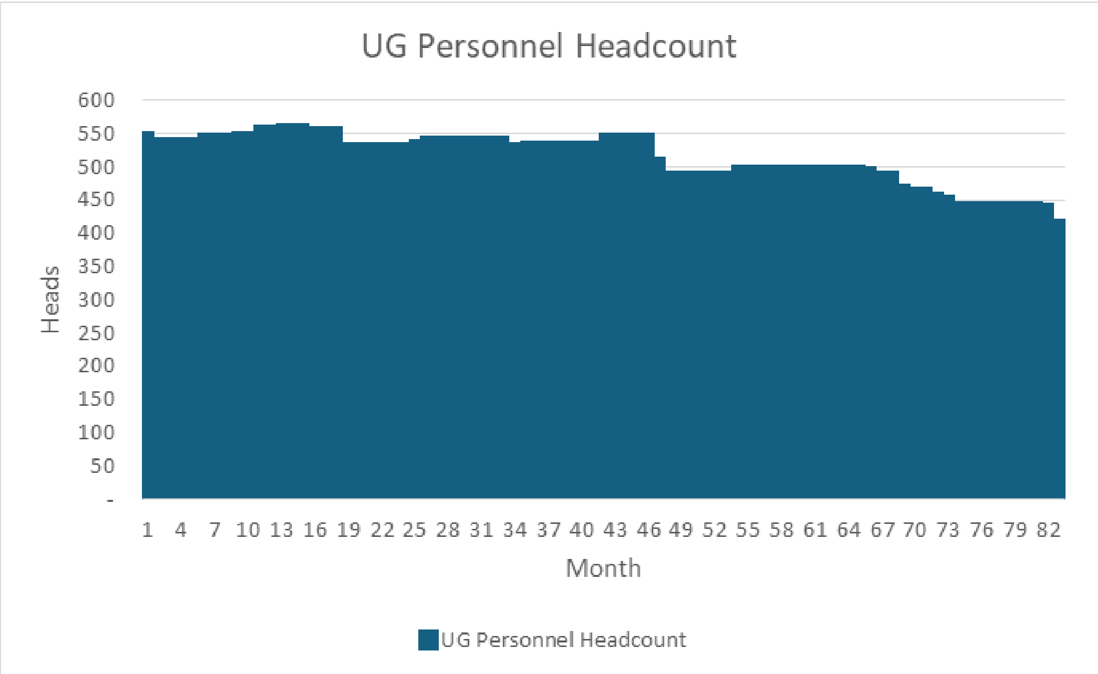


Figure 16-52 Total Underground Mining Personnel Per Month

16.12.3 Mine Facilities

The site layout is illustrated in Figure 16-53.

Figure 16-53 Overall Site Layout (~6 km From Mine Portal to Processing Plant)

16.12.4 Run of Mine (ROM) Pad

Underground ore is tipped outside the mine portal where there is a temporary ROM area. From here a surface haulage fleet takes ore to the final ROM pad. Its location proximity to the processing plant is approximately 6 km from the mine portal. Once the ore pass system is set up from July 2025 the haulage fleet will be loaded from the bottom of the ore passes on the 890L.

16.12.5 Waste Rock Storage Facility

Surplus underground waste that is not used for backfilling, is tipped outside the mine portal where there is a temporary dump area. From here a surface haulage fleet takes the waste to the Waste Rock Storage Facility which is approximately 6 km from the mine portal.

16.12.6 Fuel Storage

Diesel is stored on surface in existing licensed facilities.

16.12.7 Explosives Storage

There is an existing surface magazine compound.

17. RECOVERY METHODS

17.1 Overview

The plant has been designed on the basis of treating underground ore from the Kora and Judd deposits. The process plant design and technology is well proven in the industry and treats ore through a conventional crushing and grinding circuit incorporating both gravity recovery and flash flotation, followed by conventional flotation to produce saleable gold-silver-copper concentrate and a dorè product. The flowsheet makes use of metallurgical unit processes already used in the existing Kainantu processing plant treating these ore types.

The key project design criteria for the plant are:

- Throughput of 1.2M tpa of ore with a grind size of 80% passing (P_{80}) 106 µm.
- Major unit operations and equipment are sized with 20% design margin.
- Crushing plant utilization of 68.5% (6,000 hr/y).
- Grinding and flotation plant utilization of 91.3% (8,000 hr/y) supported by crushed ore storage, standby equipment in critical areas and back-up power from an on-site diesel power station.

Selected Process Flowsheet

The treatment facility incorporates the following unit process operations:

- Single stage crushing to produce a crushed product size of 80% passing (P_{80}) of 90 mm.
- Direct mill feed via a crushed ore surge bin with an emergency dead stockpile arrangement and reclaim feeder facilities.
- Primary semi-autogenous (SAG) milling followed by secondary ball milling in closed circuit with hydro-cyclones to produce a grinding circuit product size of 80% passing (P_{80}) of 106 µm.
- A dedicated flash flotation cell on the cyclone underflow to recover gold, silver and copper bearing minerals into a high-grade stream that will report to final concentrate.
- Gravity gold recovery from a portion of the cyclone underflow followed by gravity separation using wet shaking tables and gold room operations to produce dorè bars.
- Gold-silver-copper flotation utilising roughing, scavenging, cleaning, cleaner scavenging and recleaning stages to produce a saleable high-grade gold-copper concentrate.
- Concentrate thickening, filtration and loading of concentrates into 20 ft sea-containers for dispatch from the mine site.
- Tailings thickening and disposal to a tailings storage facility (TSF) for deposition.
- Reagent mixing, storage and distribution.
- Water and air services.

An overall process flow diagram depicting the unit operations incorporated in the selected process flowsheet is presented in Figure 17-1.

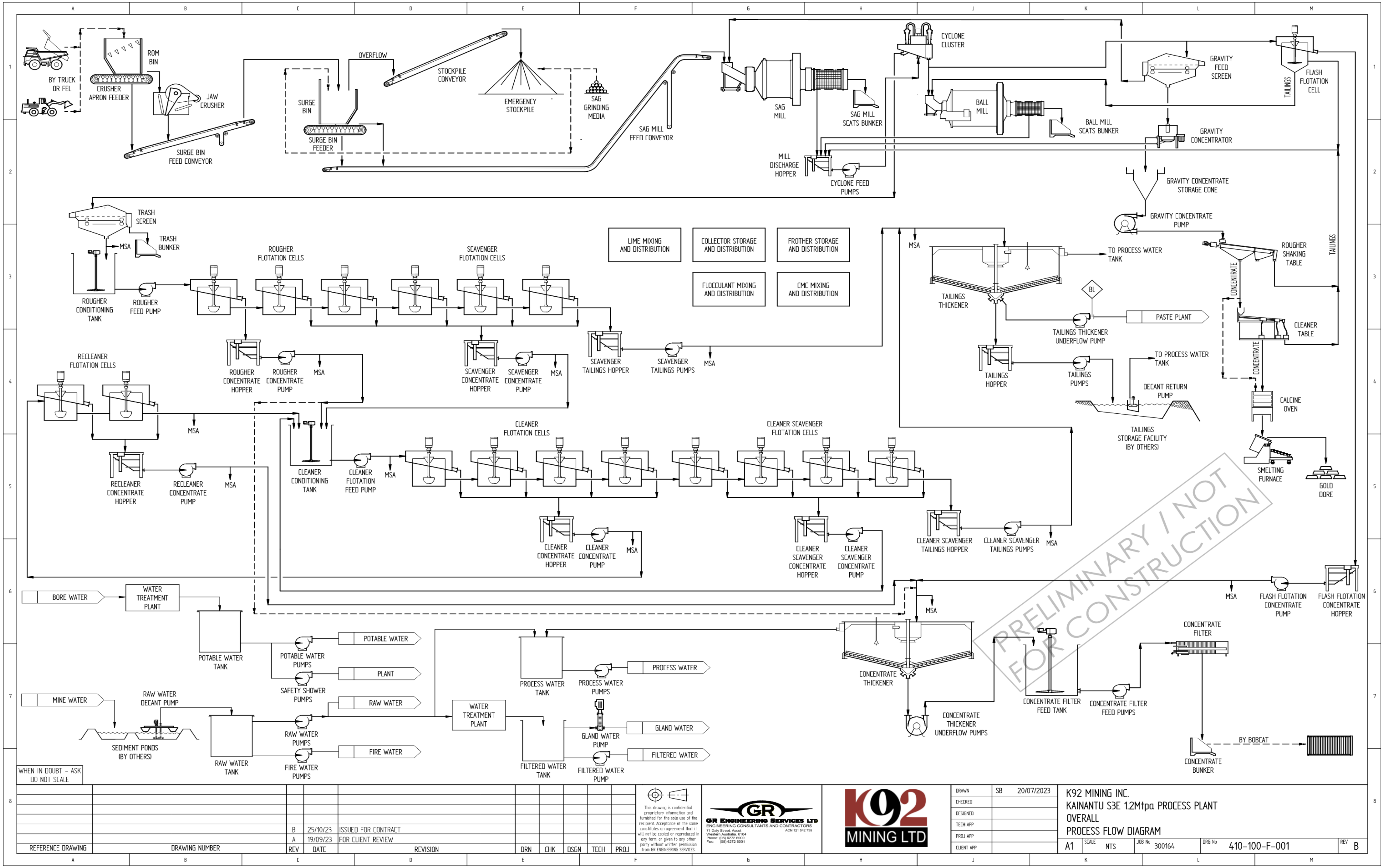


Figure 17-1 Overall Process Flow Diagram

17.1.1 Key Process Design Criteria

The process plant has used the following key process design criteria listed in Table 17-1 as the basis of design development.

Table 17-1 Summary of Key Process Design Criteria

Parameter	Units	Value
Annual Plant Throughput	M tpa dry	1.2
Ore SG - Design	t/m^3	2.78
Ore Moisture - Design	%	5
Head Grades - Design		
Gold	g/t Au	10
Copper	% Cu	1.0
Overall Gold Recovery (Gravity & Flotation) - Design	%	93.0
Gravity Gold Recovery - Design	%	15
Overall Copper Recovery - Design	%	95.0
Crushing Circuit Annual Operating Hours	hr/y	6,000
Process Plant Annual Operating Hours	hr/y	8,000
Crushing Plant Feed Rate - Design	t/hr	200
Process Plant Feed Rate - Design	t/hr	150
Crushing Work Index, CWi (85th Percentile)	kWhr/t	16.4
Bond Ball Mill Work Index, BWi (85th Percentile)	kWhr/t	16.8
SMC Test Parameter, A*b (15th Percentile)		56.4
Bond Abrasion Index, Ai (Average)		0.129
Grinding Circuit Product Size, P_{80}	µm	106
Rougher / Scavenger Flotation Feed Density - Design	% solids (w/w)	30
Cleaner Flotation Feed Density - Design	% solids (w/w)	20
Rougher / Scavenger Laboratory Flotation Time	min	10
Cleaner Laboratory Flotation Time	min	4
Cleaner Scavenger Laboratory Flotation Time	min	4
Scale-up Factor - Design		2.5
Recleaner Laboratory Flotation Time	min	4
Total Mass Pull to Concentrate - Design	%	5
Concentrate Thickener Settling Flux	t/m^2/hr	0.25
Concentrate Thickener Underflow Density	% solids (w/w)	68
Concentrate Filter Utilization - Design	%	80
Concentrate Filter Discharge Cake Moisture	%	10.5
Tailings Thickener Settling Flux	t/m^2/hr	0.7
Tailings Thickener Underflow Slurry Density	% solids (w/w)	59

17.2 Process And Plant Description

17.2.1 ROM Pad and Crushing

The single stage open circuit crushing plant will prepare Run of Mine (ROM) ore for feed to the grinding circuit. ROM ore is delivered either by front-end loader (FEL) or direct tip by haul truck into the 150 tonne capacity ROM bin. The ROM bin is equipped with a static grizzly screen with a 600 mm by 600 mm aperture to protect the crusher from oversize rock and a level transmitter for remote monitoring of level measurement and control of tipping operations at the ROM bin.

Mining will be required to supply ore at a maximum (nominal) particle size of 600 mm to minimise grizzly cleaning requirements. Any oversize ore sorted from the ROM feed will periodically require a mobile rock breaker to break the oversize rocks to a size smaller than 600 mm before feeding to the ROM bin. An apron feeder will be located directly beneath the ROM bin to feed ore into the primary crusher. The apron feeder is equipped with a variable speed drive to enable control of the crushing circuit production rate as measured by the surge bin feed conveyor weightometer. The crusher apron feeder discharge reports to the 160 kW, 810 mm x 1,370 mm primary jaw crusher which in turn discharges onto the surge bin feed conveyor. The ROM ore will be reduced in size from an F_{80} of 317 mm to a P_{80} of 90 mm by the primary jaw crushing, which operates with a closed side setting (CSS) of 100 mm.

A self-cleaning belt type tramp metal magnet is located above the conveyor to remove tramp metal from the ore stream.

A camera, situated above the crusher apron feeder, will be used to monitor the feed into the crusher.

Dust emissions in the primary crushing area are minimised by the use of appropriate skirting and a dust collector. Access will be provided for a mobile crane to undertake the lifting required for maintenance purposes.

17.2.2 Ore Storage

Ore from the crushing circuit will discharge from the surge bin feed conveyor into the surge bin. The surge bin has a maximum live capacity of 75 tonnes to provide 30 minutes of mill feed at the design throughput.

Crushed ore will be withdrawn from the surge bin via a variable speed apron feeder and delivered to the grinding circuit by the SAG mill feed conveyor. A weightometer on the SAG mill feed conveyor will indicate the instantaneous and totalised mill feed tonnage and will be used to control the speed of the surge bin apron feeder.

When the primary crusher operated at a tonnage rate higher than the SAG mill feed rate, excess material will overflow the surge bin onto the stockpile conveyor via a chute. The stockpile conveyor transfers material onto the 1,800 tonne capacity emergency static stockpile, equivalent to approximately 12 hours of mill feed storage for use during periods of crushing circuit down time.

Crushed ore on the emergency stockpile will be reclaimed by FEL during crushing circuit down time and tipped directly into the surge bin by use of an earthen access ramp. The ramp will be constructed adjacent to the emergency stockpile for easy access. The surge bin feeder will be used for both normal and emergency mill feed operations. Grinding media for the SAG mill will be added to the surge bin by FEL as required.

17.2.3 Grinding and Classification

The grinding circuit will comprise of a conventional two stage SAB milling circuit, consisting of a single grate discharge SAG mill and a single trunnion overflow ball mill. The SAG mill will operate in open circuit, whilst the ball mill will operate in closed circuit with hydro-cyclones and a flash flotation cell for recovery of gold, silver and copper minerals from the circulating load. A portion of the cyclone underflow will also be treated by a gravity separation circuit to enhance the recovery of free gold particles that concentrate in the cyclone underflow stream.

The grinding and classification circuit reduces the primary crushed feed to a product size of 80% passing 106 µm for the downstream flotation stages. The SAG Mill is a 5.8 m diameter (inside shell) by 3.35 m long (effective grinding length), mill equipped with an 1,850 kW single pinon drive started and operated by the mill variable speed drive (VSD) to compensate for the variability in ore hardness. The variable speed drive is capable of varying the SAG mill speed within a range of 60% to 80% of mill critical speed. The SAG mill will also be installed with load cells to measure the mill weight and to assist in optimising and controlling mill performance.

Process water will be added to the SAG mill feed chute, to obtain the target pulp density within the mill. Hydrated lime slurry will be added to the SAG mill feed chute for pH control prior to the downstream flotation circuits. The SAG mill contains a ball charge of approximately 10.5% balls by volume and a total charge of approximately 25%. Top up grinding balls are added to the SAG mill feed conveyor via the surge bin and surge bin feeder.

The SAG mill discharge reports to the SAG Mill trommel fitted with 10 x 20 mm slotted panels. SAG mill trommel oversize will be directed to the SAG mill scats bunker, from where scats are periodically returned to the surge bin or stockpiled by a front-end loader. Trommel screen undersize will flow by gravity to the mill discharge hopper, where it will join with the discharge slurry from the ball mill, tailings from the gravity concentrator, upper outlet tailings from the flash flotation cell, grinding and gold room area clean up sumps and will be diluted to the correct cyclone density before being pumped by one of two variable speed cyclone feed pumps to the cyclone cluster for classification. There is density and flow measurement on the cyclone feed line for recirculating load calculation. The cyclone cluster will consist of 8 by 400 mm diameter cyclones. Typically, five cyclones will be operating with three units remaining as standby.

Cyclone overflow will flow by gravity to flotation circuit via a trash screen. The horizontal vibratory trash screen is 1.2 m wide and is 4.8 m long and is fitted with polyurethane panels having 1.0 mm by 18 mm apertures and has two process water spray bars. Trash screen oversize is directed to a trash bunker at ground level for periodic manual removal. The trash screen undersize reports the rougher conditioning tank.

Cyclone underflow (the coarse fraction) gravitates to the cyclone underflow distribution box where the flow is split. The cyclone underflow distribution box has three outlets, with one feeding the flash flotation cell, one feeding the gravity scalping screen and one returning the remainder of the cyclone underflow stream to the ball mill feed chute for further grinding.

The rubber lined ball mill is 4.2 m diameter (inside shell) by 6.85 m long (effective grinding length), mill equipped with an 1,850 kW motor, started by the mill VSD and operated at a fixed speed of 75% critical speed with a ball charge of 29% by volume. The ground slurry discharges from the ball mill and flows onto the ball mill trommel. Ball mill trommel oversize will be washed by water sprays and then discharged via a chute into the ball mill scats bin. The scats bin is periodically emptied by a front-end loader. Trommel undersize from the ball mill will report to the mill discharge hopper.

Grinding media for secondary grinding is added to the ball mill via the ball mill charging hoist and the ball mill ball charging kibble.

The grinding, classification and gravity area has two floor sumps, each equipped with a vertical spindle pump for clean-up, with spillage returned to the mill discharge hopper. A cyclone davit crane and gravity concentrator hoist are provided for maintenance on the cyclones, gravity circuit and associated equipment.

17.2.4 Flash Flotation

To maximise the recovery of gold, silver and copper bearing minerals a flash flotation cell will treat a portion of the cyclone underflow stream with the flotation concentrate removed from the grinding circuit at relatively coarse particle size. The design will reduce losses associated with over-grinding sulphide minerals before conventional flotation. The flash flotation machine will also provide a capacity buffer for the conventional flotation circuit during times when high feed gold grade ores are processed.

Cyclone underflow gravitates to the cyclone underflow distribution box from where a portion of the underflow stream is directed to the flash flotation cell. An appropriately sized orifice plate in the flash flotation cell outlet will provide the required flow split to the flash flotation cell. The flash flotation cell feed will be diluted to a target pulp density and flotation reagents added to aid the flotation process via the flash flotation reagents feed box. A Metso SK240 dual outlet flash flotation cell has been selected which will be capable of treating up to 240 t/hr of feed, equivalent to 40% to 50% of the recirculating load. Low pressure air is introduced down the rotor shaft and controlled to a flow setpoint.

The concentrate is recovered over the launder lip and, with the aid of spray water, flows by gravity to the flash flotation concentrate hopper from where it is pumped to the final concentrate thickener feed box via the Multi Stream Analyser (MSA). The flash flotation cell maintains level by a dual level control system that modulates the mid-point offtake valve and lower discharge valve. The mid-point tailings offtake from the flash flotation cell discharges a low density fine slurry that gravitates to the mill discharge hopper thereby bypassing the ball mill. The bottom tailings discharge gravitates to the ball mill impingement box, where it combines with the cyclone underflow and feeds into the ball mill for further grinding.

Hydrated lime slurry will be added to the SAG mill feed chute for downstream pH control. Collector to render the valuable mineral surfaces hydrophobic will be dosed to the flash flotation cell via a flash flotation reagents feed box along with frother and Sodium Carboxymethyl Cellulose (CMC). Frother is used to create a froth phase which acts as the transport mechanism for hydrophobic minerals to report to the concentrate stream and CMC is used to depress readily floatable gangue minerals.

17.2.5 Gravity Circuit

The gravity circuit consists of a horizontal vibrating gravity screen and a single centrifugal gravity concentrator treating a portion of the cyclone underflow stream. An appropriately sized orifice plate in the gravity circuit outlet will provide the required flow split to the gravity circuit.

Feed from the gravity circuit outlet will be combined with dilution water at the gravity screen feed box to drop the density to approximately 65% solids and then fed onto the 1.5 m wide and 3.6 m long horizontal vibrating screen fitted with polyurethane panels having 2.5 mm by 18 mm apertures and two process water spray bars. Oversize from this screen will flow by gravity to the SAG mill impingement box and then report to the SAG mill feed chute for further grinding. Undersize material from the gravity screen will report to the SB1350 Falcon centrifugal concentrator where precious metals amenable to gravity recovery will be collected into a primary gravity concentrate. The gravity concentrator will be supplied with high pressure water to fluidise the bowl during operation. Gravity circuit tailings will gravitate to the mill discharge hopper.

The gravity concentrate is a batch process whereby the concentrate is discharged on a periodic regular basis from the gravity concentrator and flows by gravity to the gravity concentrate storage Cone located within the secure gold room. The concentrate storage cone has capacity to hold 24 hours concentrate solids production from the gravity concentrator when operated with a dump cycle of 40 minutes. Decanting of the excess concentrator flushing water from the cone, is done automatically at the end of every dump cycle to assist the settling of fine concentrate solids in the hopper. Gravity circuit tailings will gravitate to the mill discharge hopper.

The area will be serviced by the nearby grinding area sump pump to assist with clean-up. A gravity concentrator hoist will be provided for maintenance purposes

17.2.6 Gold Recovery

Gravity concentrates will be processed using wet shaking tables to produce a final concentrate for direct smelting to doré. The gravity concentrate storage cone and table separation circuit will be located within the secure gold room. Tabling of gravity concentrate will be a batch process performed daily.

The table separation circuits will comprise of:

- Rougher shaking table.
- Cleaner table.

The final concentrates produced from the gravity separation circuits will periodically be removed (by hand) and smelted to produce doré in the gold room.

The concentrates will be pumped from the gravity concentrate storage cone. Water will be injected into the base of the storage cone via manual valves to assist the fluidisation of the high specific gravity concentrate and achieve the required dilution for feeding onto the rougher shaking table.

The rougher table feed will report to a single deck rougher shaking table being a Holman 8000. Wash water will be added to the deck to aid the separation process. A rougher concentrate will be recovered from the table and will flow by gravity into a bucket for manual transfer to either the cleaner table feed hopper or for re-processing on the rougher shaking table. The rougher table middlings and rougher table tails will gravitate to the in-ground gold room area sump from where they will be pumped to the mill discharge hopper.

The rougher table concentrates will discharge (by hand) into the cleaner table feed box. The rougher table concentrates exit the cleaner table feed box and report to the single deck cleaner shaking table being a Gemini 250. Wash water will be added to the deck to aid the separation process. A final concentrate will be recovered from the cleaner table into a bucket for manual transfer to the gold room dry area. The cleaner table middlings and tails will gravitate to the table tailings pump and be pumped to the in-ground gold room area sump from where they are pumped to the mill discharge hopper.

The final gravity concentrates will be calcined, mixed with fluxes and smelted in a diesel fired furnace. Off-gases from the oven and barring furnace will be scrubbed in the gold room fume scrubber and vented to atmosphere. Solids captured from the off-gases will discharge into the scrubber recycle hopper, from where they will be pumped to the mill discharge hopper.

The gold room area has a floor sump, equipped with a vertical spindle pump for clean-up, with spillage returned to the mill discharge hopper.

17.2.7 Flotation Circuit

The flotation circuit produces a gold copper concentrate for downstream thickening and filtration. The flotation circuit will consist of a rougher conditioning tank, a bank of rougher / scavenger cells, cleaner conditioning tank, two stages of cleaning and one stage of cleaner scavenging as well as associated hoppers and pumps.

Trash Screen undersize will report by gravity to the agitated rougher conditioning tank. The rougher conditioning tank will act as a surge tank to reduce flow fluctuations to the rougher flotation circuit and allow contact time for flotation reagents to prepare mineral surfaces for rougher flotation. The agitated conditioning tank will have an effective volume of 47 m^3. The tank's normal operating capacity of 41 m^3 will provide 6 minutes conditioning residence time with a further 1 minute of surge capacity at the design pulp flow rate.

Reagents dosing will occur at the conditioning tank and various other points in the rougher / scavenger flotation circuit. Process water will be added to the conditioning tank via a flowmeter and flow control valve for density adjustment as required. The rougher feed pump will direct circuit feed to the rougher / scavenger flotation bank from the conditioning tank. The rougher feed pump will be equipped with a variable speed drive to control flotation feed via flow rate set point and level control at the conditioning tank.

The flotation circuit will consist of conventional, forced air addition flotation cells for the roughing / scavenging duty, two stages of cleaning and one stage of cleaner scavenging. The flotation circuit will consist of the following equipment:

- One 47 m^3 rougher conditioning tank.
- Six 40 m^3 rougher / scavenger mechanically agitated, forced air tank cells in series.
- One 20 m^3 cleaner conditioning tank.
- Four 10 m^3 cleaner mechanically agitated, forced air tank cells in series.
- Four 10 m^3 cleaner scavenger mechanically agitated, forced air tank cells in series.
- Two 5 m^3 recleaner mechanically agitated, forced air tank cells in series.

Laboratory conditioning and flotation times in the DFS test work have formed the basis for sizing of equipment with industry standard scale up factors applied and checks to ensure cell area and lip length capacities were not exceeded. The existing processing plant flotation retention times were however, also considered in the interpretation and process design. The flotation residence time was based on the aerated cell volume(s) divided by the average slurry flowrate. The effective scale-up ratio was then calculated based on the flotation time divided by the laboratory flotation time. Flotation times have effectively been scaled-up higher the industry standard of two to three times the bench scale tests as a result of the 20% design margin being applied in the basis for cell sizing. The flotation retention time are summarised in Table 17-2.

Table 17-2 Flotation Time

Flotation Stage	Effective Scale-Up Ratio	Laboratory Flotation Retention Time (minutes)	Flotation Retention Time (minutes) 1.2M tpa
Rougher / Scavenger	3.4	10	34
Cleaner	3.2	4	13
Cleaner Scavenger	3.7	4	15
Recleaner	2.9	4	12

The flotation reagent addition regime was selected based on the DFS test work. The plant design will allow space to facilitate the future installation of a xanthate reagent mixing, storage and distribution should this be necessary.

17.2.8 Rougher / Scavenger Flotation

Six cells in series were selected for the rougher / scavenger stage to minimise losses due to short-circuiting and also provide circuit flexibility. The six 40 m^3 cells will have an effective total pulp volume of 204 m^3 and a residence time of 34 minutes based on the average slurry flowrate. The rougher / scavenger bank will be set up in a 1/1/2/2 configuration.

The concentrate from the first rougher cell reports to the rougher concentrate hopper from where it will be pumped to either the cleaner conditioning tank or the concentrate thickener. Concentrate from the remaining five rougher / scavenger cells will report to the scavenger concentrate hopper from where it will be pumped to the cleaner conditioning tank. The scavenger tailings will discharge via a pinch valve to the scavenger tailings hopper. There will be duty and standby variable speed pumps that will pump the scavenger tailings to the tailings thickener.

17.2.9 Cleaner Flotation

The combined rougher / scavenger concentrate will report to the cleaner conditioning tank. The cleaner conditioning tank will act as a surge tank to reduce flow fluctuations to the cleaner flotation circuit and allow additional contact time for flotation reagents to prepare mineral surfaces for cleaner flotation. The agitated conditioning tank will have an effective volume of 20 m^3. The tank's normal operating capacity of 16 m^3 will provide a conditioning residence time of 6 minutes with a further 1 minute of surge capacity at the design pulp flow rate.

Reagents dosing will occur at the conditioning tank and various other points in the cleaner / cleaner scavenger flotation circuit. Process water will be added via a manual valve for density adjustment as required. The cleaner flotation feed pump will direct circuit feed to the cleaner flotation bank from the conditioning tank. The cleaner flotation feed pump will be equipped with a variable speed drive to control cleaner flotation feed via flow rate set point and level control at the conditioning tank.

The cleaner flotation stage has been configured as a cleaner and cleaner scavenger arrangement. The four 10 m^3 cleaner flotation cells will have an effective total pulp volume of 34 m^3 and a nominal residence time of 13 minutes based on the average slurry flowrate. The cleaner cells will be set up in a 1/1/2 configuration. The cleaner concentrate will report to the recleaner bank via the cleaner concentrate pump. The cleaner tails will discharge via a pinch valve to the copper cleaner scavenger flotation cells.

The four 10 m^3 cleaner scavenger cells will have an effective total pulp volume of 35 m^3 and a residence time of 15 minutes based on the average slurry flowrate. The cleaner scavenger cells will be set up in a 1/1/2 configuration. Cleaner scavenger concentrate will report to the cleaner scavenger concentrate hopper and will then be then recirculated to the cleaner conditioning tank via the cleaner scavenger concentrate pump. The cleaner scavenger tailings will discharge via a pinch valve to the cleaner scavenger tailings hopper. There will be duty and standby variable speed pumps that will pump the cleaner scavenger tailings to the tailings thickener.

The recleaner bank will consist of one 5 m^3 tank cell with an external peripheral launder followed by one 5 m^3 tank cell with an internal peripheral launder. The 5 m^3 cells will have an effective total pulp volume of 9 m^3 and a residence time of 12 minutes based on the average slurry flowrate. The recleaner cells will be set up in a 1/1 configuration. Concentrate from the recleaner cells reports to the recleaner concentrate hopper from where it is pumped via the recleaner concentrate pump to the concentrate thickener whilst recleaner tailings will discharge via a pinch valve to the cleaner conditioning tank. The desired mass pull from all flotation cells will be achieved by adjusting the air flow rate and froth levels. Air will be added to the individual cells to produce a froth containing concentrate which overflows the launder lip and with the aid of spray water flows to its relevant concentrate hopper. Individual cell air addition is measured by magnetic flow meters and controlled remotely by flow control valves. Individual cell froth depth is measured by ball float and is controlled by a pinch valve on the cell or bank discharge.

Frother (Dowfroth 250C), collector (Blend 1) and depressant Sodium Carboxymethyl Cellulose (CMC) will be added via dedicated dosing pumps throughout the flotation circuit. Hydrated lime (pH modifier) will be dosed throughout the circuit via a ringmain. Reagent addition rates will be controlled on a g/t of feed ore or fixed flowrate basis.

The flotation area will be serviced by the rougher area sump pump, cleaner area sump pump and the cleaner scavenger area sump pump for return of process spillage to the circuit and for clean-up.

17.2.10 Online Analysis

The flotation circuit performance will be monitored by a Multi-Stream Analyser (MSA) and shift composite samples collected for analysis by the existing on-site laboratory. This unit will require a high degree of calibration during commissioning and in the early stages of operation. Once the unit is fully calibrated and the commissioning of the sampling system is completed, the Multi-Stream Analyser (MSA) will determine percent solids and measure the copper, iron as well as gold and silver concentration where possible in key process streams with ongoing routine calibrations. The sampling system will include primary in-line samplers and sample pumps that direct streams to the MSA. Shift composite samples will be collected for analysis by the existing on-site laboratory by individual samplers that will be provided for each MSA analysis box.

The sample streams measured in the flotation area will be as follows:

- Flash Flotation Concentrate.
- Rougher Feed.
- Rougher Concentrate.
- Scavenger Concentrate.
- Cleaner Feed.
- Cleaner Concentrate.
- Recleaner Concentrate.
- Final Concentrate.
- Recleaner Tailings.
- Scavenger Tailings.
- Cleaner Scavenger Tailings.
- Final Tailings.

Concentrate Thickening, Filtration and Handling

The concentrates from the flotation circuit and the flash flotation circuit will be combined and thickened by a hi-rate thickener before reporting to a vertical plate pressure filter to produce a filter cake for shipping. Dewatered cake is stored in a storage shed before being loaded into shipping containers.

The flash flotation concentrate, and recleaner concentrate will be combined with concentrate filter filtrate in the concentrate thickener feed box from where the combined final concentrate gravitates via a final concentrate sampler into the concentrate thickener de-aeration tank. Spray water will be added to the de-aeration tank to assist in breaking down the froth prior to entry into the thickener feed well. The final concentrate sample will be pumped via the final concentrate sample pump to the MSA for analysis. The combined final concentrates will report to a 9 m diameter hi-rate concentrate thickener fitted with an auto-dilution feed system. Diluted flocculant will be added to the thickener feed well to promote settling of the solids. A bed level measuring device will be installed to monitor the thickener bed depth. The addition rate of flocculant, if required, will be controlled according to the bed depth. The thickener will be equipped with two variable speed peristaltic type underflow pumps arranged in a duty/standby configuration which will be controlled to maintain thickener bed pressure. The concentrate slurry will be thickened to approximately 68% solids and then pumped to a 100 m^3 capacity concentrate storage tank to provide 12 hours of surge capacity between the flotation and filtration unit operations. The thickener overflow will gravitate to the process water tank.

There is density and flow measurement of the thickened concentrate slurry pumped to the concentrate storage tank. Thickener torque will be maintained in a pre-set range automatically via the thickener local control panel which will raise or lower the thickener rakes according to torque reading. The thickened concentrate slurry will be pumped from the concentrate storage tank via the concentrate filter feed pumps working in a duty/standby arrangement to the batch concentrate pressure filter for dewatering. The filter will contain 50 chambers with 1.2 m by 1.2 m plates providing a total installed filter area of 115 m^2. Each filter cycle will take on average 20 minutes, including a high-pressure cloth washing cycle once every 10 filter cycles, to produce a filter cake of approximately 10.5% moisture (based on test work) and a filtrate containing minimal solids. Filtrate will be returned to the concentrate thickener via the concentrate filter filtrate pump. The dewatered filter cake will discharge into a concrete concentrate bunker beneath the filter for removal by a bobcat to a stockpile within the concentrate storage shed. Concentrate stored within the shed will be loaded by bobcat into shipping containers for transport to port. The containers will be weighed prior to leaving the area.

The concentrate filter will be serviced by the concentrate filter maintenance hoist. The concentrate thickener area sump pump and the filter area sump pump will be installed for return of process spillage and to aid clean up in each area.

17.2.11 Tailings Thickening, Disposal and Decant Return Water

The flotation tailings streams will be combined with sump pump discharges for water recover in the tailings thickener. A final tails sample will be collected and sent to the MSA for analysis. Thickener underflow slurry is pumped to the existing tailings storage facility (TSF) via the tailings transfer pumps whilst the thickener overflow water gravitates to the process water tank.

The scavenger tailings, cleaner scavenger tailings will be combined in the tailings thickener feed box and then the final tailings will be sampled separately for MSA analysis. The combined stream reports to a 25 m diameter hi-rate tailings thickener fitted with an auto-dilution feed system. Diluted flocculant will be added to the thickener feed well to promote settling of the solids. A bed level measuring device will be installed to monitor the thickener bed depth. The addition rate of flocculant, if required, will be controlled according to the bed depth. Thickener overflow gravitates to the process water tank for re-use in the plant and thickener underflow will gravitate into the tailings hopper. There are duty/standby underflow lines fitted with actuated pinch valves to regulate the discharge rate of thickener underflow to the tailings hopper as well as the capability to run the pinch valves in automatic bed pressure control using the bed pressure sensor located at the bottom of the thickener cone. The combined tailings will be thickened to approximately 59% solids and then pumped to the existing tailings storage facility (TSF) by a duty/standby, single-stage pumping arrangement. The pumps will be variable speed and ramped up and down to maintain a consistent level in the tailings hopper as part of the level control loop. The thickener underflow to the TSF will be discharged into the existing dam by spigot arrangement. Water from the supernatant pond on the TSF is collected by the decant pumps and returned to the process water tank for re-use in the plant. Excess water solutions from the wet season and when the tailings filtration plant is running will be pumped to the existing site anaerobic pond. The tailings thickener area is serviced by the tailings thickener area sump pump to assist with clean-up. The tailings area sump pump will discharge to the tailings thickener feed box or the tailings hopper.

17.2.12 Reagent Mixing, Storage and Distribution

The reagents used in the processing plant will include:

- Hydrated Lime.
- Collector: Blend 1 collector (Aero3477, Aero6697 and Aero5100 with mass ratio of 64:16:20).
- Frother: Dowfroth 250C or equivalent.
- Sodium Carboxymethyl Cellulose (CMC).
- Flocculant.
- Smelting Fluxes.

17.2.13 Hydrated Lime Slurry

Lime slurry will be used as the pH modifier in the flotation process. Hydrated lime ($CaOH_2$) will be delivered to site as a powder in 1 tonne bulk bags and will be manually unloaded into lime storage bin using the lime hoist.

The lime will be discharged from the bottom of the lime storage bin via the lime screw feeder and wetted in the lime mixing vortex mixer prior to discharging into the lime mixing tank. There will be an activator fitted to the bottom of the Lime Storage Bin which is activated at regular intervals to prevent cone blockages.

A milk of lime suspension at approximately 20% solids (w/v) will be mixed in batches with raw water dilution added into the agitated 5 m^3 capacity lime mixing tank. After a pre-set mixing time, the lime will be periodically transferred to the agitated 20 m^3 capacity lime storage tank via the lime transfer pump. The lime distribution pumps will be used to distribute the milk of lime throughout the circuit via a ring main. For all addition points, a remotely actuated on / off pinch valve on a timed sequence will be provided to achieve the desired target pH in that area.

The area will be serviced by the lime area sump pump to assist with clean-up.

17.2.14 Collector

Blend 1 collector or equivalent will be used as the collector in the flotation process. The Blend 1 collector is a proprietary blend of other collectors, Aero® 3477, Aero® 6697 and Aero® 5100. The reagent will be delivered to site in 1,000 litre IBC as a neat solution. The IBC will be transferred via the air operated collector transfer pump to the 1.5 m^3 storage tank. The reagent will then be dosed via individual dosing pumps to the following locations:

- Flash flotation reagents tundish.
- Rougher conditioning tank.
- Rougher flotation cell 3.
- Cleaner conditioning tank.
- Cleaner scavenger flotation cell 1.
- Cleaner concentrate hopper.

All dosing pumps are controlled on a g/t of ore feed basis or by manual addition. The collector area will be serviced by with the collector sump pump to assist with clean-up.

17.2.15 Frother

Frother will be used in the flotation process to increase stability and drainage of the froth, improving flotation recovery and grade. Dowfroth 250C will be used as the frother in the flotation process. The reagent will be delivered to site in 1,000 litre IBC as a neat solution. The IBC will be transferred via the air operated frother transfer pump to the 1.5 m^3 storage tank. The reagent will then be dosed via individual dosing pumps:

- Flash flotation reagents tundish.
- Rougher flotation cell 1.
- Scavenger flotation cell 1.
- Cleaner flotation cell 1.
- Cleaner scavenger cell 1.

- Recleaner flotation cell 1.

All dosing pumps will be controlled on a g/t of ore feed basis or by manual addition. The area will be serviced by the cleaner area sump pump to assist with clean-up.

17.2.16 Depressant Sodium Carboxymethyl Cellulose (CMC)

Sodium Carboxymethyl Cellulose (CMC) will be used as a depressant for silicate type minerals in the flotation process. CMC will be delivered to site in 700 kg bulk bags and will be manually unloaded into the mixing system using the flocculant mixing hoist that is shared with the CMC system. The bags will be lifted into the CMC bag splitter and the contained CMC will be emptied into the 1.5 m^3 CMC storage bin.

The CMC mixing system will be a proprietary packaged plant consisting of the CMC storage bin, powder feed and wetting system and 6 m^3 capacity CMC mixing tank with agitator. CMC will be mixed automatically with raw water on a batch basis to generate a 1.0% (w/v) solution. After a suitable hydration period, the CMC solution will be automatically transferred from the mixing tank to the 30 m^3 capacity CMC storage tank.

The CMC will then dosed via individual dosing pumps to the following locations:

- Flash Flotation Reagents Tundish.
- Rougher Conditioning Tank.
- Scavenger Flotation Cell 1.
- Cleaner Conditioning Tank.
- Cleaner Scavenger Flotation Cell 1.

All dosing pumps will be controlled on a g/t of ore feed basis or by manual addition. The CMC area will be serviced by the flocculant sump pump to assist with clean-up.

17.2.17 Flocculant

Flocculants are long chain molecules that aid solids settling by causing individual particles to stick together thereby forming larger, heavier particles. Flocculant will be delivered to site in 750 kg bulk bags and will be manually unloaded into the mixing system using the flocculant mixing hoist. The bags will be lifted into the flocculant bag splitter and the contained flocculant will be emptied into the 1.5 m^3 flocculant storage bin.

The flocculant mixing system will be a proprietary packaged plant consisting of the flocculant storage bin, powder feed and wetting system and 6 m^3 capacity flocculant mixing tank with agitator. Flocculant will be mixed automatically with raw water on a batch basis to generate a 0.25% (w/v) solution. After a suitable hydration period, the flocculant solution will be automatically transferred from the mixing tank to the 30 m^3 capacity flocculant storage tank.

The flocculant will then be dosed via individual dosing pumps to the to the concentrate thickener and tailings thickener to control the bed level in each thickener.

The flocculant area will be serviced by the flocculant sump pump to assist with clean-up.

17.2.18 Gold Room Smelting Fluxes

The gold room smelting flux constituents will be:

- Borax ($Na2B4O_2$).
- Soda Ash ($Na2CO_3$).
- Sodium Nitrate (Nitre or $NaNO_3$).
- Silica Flour (SiO_2).

The gold room smelting fluxes will be delivered to the plant in 25 kg bags.

17.2.19 Consumables

Grinding Media

SAG and ball mill grinding media will be delivered to site in bags or drums with a nominal weight of 1 tonne. The two sizes to be used will be 125 mm and 50 mm but this may vary after mill optimisation is performed. Grinding media for the SAG mill will be added to the surge bin by FEL as required. The quantity of SAG mill grinding media to be added daily will be based on the SAG mill power draw and ore properties. Media will be added to the surge bin containing ore or pre-mixed with ore to prevent damage to the surge bin feeder.

Grinding media for secondary grinding will be added to the ball mill via the ball mill charging hoist and the ball mill ball charging kibble. Ball mill grinding media to be added daily will be based on the ball mill power draw and ore properties.

Diesel

Diesel will be pumped from existing site diesel storage facility to a self-bunded diesel day tank located outside the gold room via a tie in to the existing plant diesel supply. Diesel fuel will be pumped from diesel day tank to the gold furnace in the gold room during smelting.

Water Storage and Reticulation

The process plant will utilize raw water, filtered water, process water and potable water. Make-up water requirements for process water will be pumped from the raw water tank.

Rain water and site run off will report to existing site run-off pond.

17.2.20 Raw Water

Raw water will be supplied from the existing underground mine to the raw water sediment ponds and then pumped to the process plant by the raw water decant pump. At the process plant, raw water will be fed into the 1,000 m^3 capacity raw water tank. Two raw water pumps arranged in a duty-standby configuration will be located at the raw water tank. These pumps will distribute the raw water throughout the processing facility.

The raw water will be used for the following:

- Gravity concentrator fluidisation water.
- Reagent mixing.
- Table wash water.
- Furnace fume scrubber.
- Process water make-up.

The lower portion of the raw water tank will provide a dedicated fire water reservoir for the fire water system. The fire water system will include three pumps, an electric fire water pump, a diesel driven fire water pump and a fire water jockey pump to ensure a continuous supply of water to the fire system in the event of a power failure.

17.2.21 Filtered Water

The raw water tank will also supply the 300 m^3 capacity filtered water tank via the filtered water treatment plant which will be a vendor controlled system to produce filtered water. Two filtered water pumps arranged in a duty-standby configuration will be located at the filtered water tank. In addition, there will be two dedicated pumps, arranged in a duty-standby configuration, for gland seal water.

The filtered water will be used for the following:

- Cloth washing and membrane squeeze in filtration.
- Multi-Stream Analyser (MSA) sprays.
- Gland seal water.

17.2.22 Process Water

Process Water will be used extensively throughout the processing plant and will be supplied by concentrate thickener overflow water, tailings thickener overflow water and decant return water from the TSF which will report to the 470 m^3 capacity process water tank along with raw water make-up as required. Two process water pumps arranged in a duty-standby configuration will be located at the process water tank. These pumps will distribute the process water throughout the processing facility.

The process water will be used for the following:

- Grinding area dilution water.
- Screen sprays.
- Flotation area dilution and knock down water.
- Flocculant dilution water.
- Tailings line flushing water.
- General purpose hose down points.

17.2.23 Potable Water

Potable water will be supplied from the potable water treatment plant. The potable water treatment will be supplied by an existing bore PBW/PW1 to feed the plant. The potable water treatment plant will be a vendor controlled package that stops and starts based on the level in the 10 m^3 capacity potable water tank.

Potable water will be distributed via a ring main network using the safety shower water pump to all the safety showers. A back-up diesel safety shower water pump will provide potable water to safety showers during power outage events. Potable water will be also distributed using the potable water pumps arranged in a duty and standby configuration to the plant control room, gold room and to top up the mill cooling water chiller.

17.2.24 Air Services

High pressure air for the process plant will be supplied by two 75 kW air compressors which operate in a lead/lag arrangement. The compressed air will then be fed into the 5 m^3 capacity plant air receiver for general use in the plant at a supply pressure of approximately 750 kPa. All HPA is dried by the plant air drier before subsequent distribution.

An additional dedicated air compressor will provide air services to the concentrate filter air as higher pressure air for the cake drying and core blow cycles and a separate receiver will be housed at the filter building for this duty.

Low pressure air is used in the flotation process to provide air to the cells for valuable mineral recovery. The low pressure air required is supplied by two flotation blowers arranged in a duty standby configuration. Each blower is equipped with a fixed speed drive and a variable air intake valve. A pressure transmitter is fitted to the blower discharge which monitors the pressure generated by the blowers.

17.3 Projected Energy, Water And Process Material Requirements

17.3.1 Power Consumption

The power for the processing plant will be grid power supplied from an overhead line with back-up power from an on-site diesel power station. The projected average power demand is summarized in Table 17-3. The average power demand does not reflect the instantaneous power demand for equipment start-up and electrical system capacity sizing.

Table 17-3 Average Power Demand Summary

Area	Installed Power (kW)	Average Power Consumption (kW)	Annual Power Consumption (kWh/year)
Crushing	269	179	1,104,529
Coarse Ore Storage & Handling	70	35	235,250
Grinding, Classification & Flash Flotation	4,916	3,422	26,905,937
Gravity Recovery	25	18	128,443
Flotation	964	619	4,866,867
Concentrate Thickening, Filtration & Handling	273	157	514,342
Gold Recovery	131	81	500,177
Reagent Mixing, Storage & Distribution	70	32	139,481
Water Storage & Reticulation	654	304	2,403,090
Air Services	486	218	1,379,707
Tailings Thickening, Disposal & Decant Return Water Administration	404	298	1,567,320
Total (per year)	**8,282**	**5,377**	**39,806,064**

17.3.2 Water Consumption

A water balance for the process plant has been completed. Water from the thickeners overflow streams will be recycled within the process plant to reduce the external water requirement. Approximately 419 m^3/hr of combined overflow water from the concentrate and tailings thickeners will be transferred back to the process water tank along with nominally 40 m^3/hr decant return water from the existing TSF.

During steady state operation at design throughput, it is estimated that the processing facility will consume approximately 12 m^3/hr of make-up water based on an average operating hour of the process plant with tailings filtration plant off-line, however this will vary between the wet and dry seasons. During steady state operation at design throughput, when the tailings filtration plant is on-line, it is estimated that there will be approximately 30 m^3/hr of excess process water produced from the processing facility based on an average operating hour of the process plant. The excess water solutions when the tailings filtration plant is running will be pumped to the existing site anaerobic pond via the existing TSF.

17.3.3 Reagent and Consumable Consumption

Reagent storage, mixing and distribution facilities will be provided for all reagents for the process plant. Table 17-4 provides a summary of the projected wear liner consumables and reagents for the design plant throughput of 1.2M tpa as specified in the operating cost estimate.

Table 17-4 Annual Reagent and Consumable Consumption

Item	Annual Usage
Grinding Media (125 mm balls)	372 t
Grinding Media (50 mm balls)	516 t
Frother - Dowfroth 250 or equivalent	18 t
Collector - Blend 1	126 t
Depressant - CMC	90 t
Flocculant - Magnafloc 155 equiv	60 t
Hydrated Lime	1,800 t
Caustic	
Borax	1.4 t
Silica	0.5 t

Item	Annual Usage
Sodium Nitrate	0.2 t
Soda Ash	0.2 t
Primary Crusher Fixed Jaw Die	4 sets
Primary Crusher Swing Jaw Die	2.5 sets
Primary Crusher Upper Cheek Plate	2 sets
Primary Crusher Lower Cheek Plate	1 set
SAG Mill Linings	2 sets
Ball Mill Linings	1 set
Filter Cloths	13 sets
Smelting Crucibles	10 each
Diesel	686 kL

17.4 Process Control System

The process plant control system will be a PLC based system. The HMI will utilise standard personal computers running Siemens software to provide control. The plant control room will be located in the flotation area with a subsidiary control room in the crushing area.

From the main control room, the plant status can be monitored via two PC based Supervisory Control and Data Acquisitioning (SCADA) screens via Siemens software. Two additional visual display units will also be installed in the main control room to present the operator with graphical process information in the form of trends, mimic pages, alarm summaries, logs and reports. Plant equipment will be remotely started and stopped from the operator screens via start and stop sequences or by manually starting or stopping individual equipment.

The plant will be designed to allow the operator to control and monitor the entire plant. Motor starters, main isolators and distribution board feeders will be located in the substations. Each substation will also contain a programmable logic controller (PLC) which, along with the SCADA system, makes up the Process Control System (PCS). The PCS will provide the interface between drives and instrumentation and the operators.

Vendor supplied packages will typically use vendor standard control systems. Vendor packages will generally have limited interfaces with the PCS. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the HMI. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

18. PROJECT INFRASTRUCTURE

18.1 Power

The supply of electric power will be via grid connection to the Papua New Guinea Power Limited (PPL) network, 22 kV overhead line (OHL) from the Gusap Substation located 24 km away. A new diesel back-up power station forms part of the site-wide power reticulation for the project.

Generators that are located on the existing power system at the 800 Portal, existing Kainantu process plant, and camp will operate independently as satellite systems until the main back-up power plant becomes fully operational.

K92ML constructed and installed a 4.8 MW power station which became fully operational at the beginning of 2024. The main purpose of the 4.8 MW power station is to provide back-up power supply for the underground mine at the 800 Portal via existing 11 kV OHL, prior to the completion of the main back-up power station.

A new 800 Portal Switchroom including 12 MVA and 1.5 MVA associated transformers are scheduled to be on site by end of 2024, which will require conversion of the existing 11 kV OHL to 22 kV. At that point, the 4.8 MW power station transformers will be adjusted to step-up the voltage to 22 kV.

Following is a summary of the back up power supply for the Kainantu operation:

- Existing satellite gensets: These units are spread across the site in various locations: existing Kainantu process plant (3 x 1.25 kVA and 1 x 0.8 kVA generators), existing camp (1 x 1.25 kVA generator), new camp (1 x 1.25 kVA generator to be commissioned in 2025), and 800 Portal (2 x 1.25 kVA). These units act as back up to the current power supply to the site.
- 4.8 MW power station: This was constructed and commissioned during 2024, utilising some of the existing gensets on site in addition to new units, totalling to 6 x 1.25 kVA generators. The main purpose of this power station is to act as back up for 800 Portal and underground loads; as it was expected that underground load will increase prior to installation of the main back-up power station. The 4.8 MW power station will be re-purpose to be connected to 22 kV network;
- New 8.8/20 MW power station: this power station will act as back up for the entire Kainantu Mine once it becomes operational. The new power station will be built in two (2) stages; each having 7 x 1.63 MW generators. This will result in total installed power of 14 x 1.63 MW unit after building the second stage. All the other power stations and satellite units will become redundant.

The first stage of the new diesel back-up power station has been sized based on the maximum demand requirement of the new 1.2M tpa treatment plant, and the tailings filtration plant which forms part of the paste back-fill system. The diesel generators have a prime power output rating of 1.63 MW. The back-up power station will be constructed and installed in two separate stages; each comprising 7 x 1.63 MW generators. This will result in total installed power of 14 x 1.63 MW unit after installation of the second stage.

The first stage will provide 8.8 MW power continuously using six (6) units with one being off-line for maintenance.

Initially all of the electrical loads which are connected to the 22 kV OHL at the 800 Portal and underground areas will be fed from the 4.8 MW power station; which is purposely built to be able to be connected to the 22 kV network. Currently this 4.8 MW power station is connected to the 11 kV OHL which runs to the 800 Portal area; 5.5 km away.

Currently all underground loads are fed from 11 kV OHL - which runs from the process plant to the 800 Portal -and back-up generators at 800 Portal. The 11 kV OHL has been inspected and fixed to suit 22 kV network. Eventfully this OHL will be used as 22 kV; which will be fed from 4.8 MW power station initially until it is connected permanently to the main 22 kV Switchboard in the main back-up power station.

The 22 kV Switchboard - which is part of the back-up power station design - will be the main power source for the entire Kainantu mine. The board will receive power from both power station and future 22 kV OHL - which is being design by PPL to replace the current 22 kV OHL - and will adjust power consumption to ensure the

22 kV OHL is feeding majority of the loads; especially when the current 22 kV OHL is still in operation. It should be noted that the current 22 kV OHL can only provide maximum of 8 MW to the mine due to the voltage drop.

Provision for future expansion of the power station has been made with the sizing of the power station switchroom to accommodate the additional generator HV circuit breakers and control panels.

Capacitor banks on the 22 kV supply have been included in the design. The basis for these capacitor banks is the key outcome from a power study done as part of the overall study. Without some form of reactive compensation, the powerline to the site will not have the capacity to supply the load as the acceptable voltage limits will be exceeded. All the other 22 kV switchgear and transformers are included as part of the capital estimate.

Table 18-1 Electric Power Consumption for the Project

Area	Installed Power (kW)	Average Demand (kW)	Peak Power (kW)	Comments
Camp	1,500	1,000	1,100	Utilize existing site genset and new genset for independent / satellite power back-up
1.2M tpa Plant	8,096	5,443	6,329	The new 8.8 MW Power Station and existing PPL OHL will feed the loads
Tailings Filtration Plant	750	510	610	The new 8.8 MW power Station and existing PPL OHL will feed the loads.
Bus Stop Area	370	290	340	Initially will be fed from 4.8 MW power station and existing OHL.
Underground Paste-Backfill Plant	2,452	1,145	1,366	Initially will be fed from 4.8 MW power station and existing OHL.
Mine	7,887	4,598	6,950	Utilize existing site gensets for independent / satellite power back-up. After upgrading the current 11 kV OHL to 22 kV OHL, during power outage 4.8 MW power station will be brought online to feed the loads.
Total	21,005	12,986	16,695	

Note 1: The above table does not include following loads:

- *Existing Kainantu process plant;*
- *Camp upgrade;*
- *U/G mine expansions which are estimated to peak at 10 MVA in the future; and hence the new design for the 800 Portal Switchroom and 12 MVA 22/11 kV power transformer has considered this maximum demand.*

18.2 Water

Raw water for the process plant will be sourced from mine dewatering water from the 800 Portal. The water from the 800 Portal will make its way to the plant site via an existing HDPE pipe and report to sediment pond one and then overflow to sediment pond two. Raw water is then pumped from sediment pond two into the raw water tank and used as required.

Surface run-off from the plant site will be collected in a storm water pond, where it then overflows to the creek.

18.3 Non-Process Infrastructure

Several non-process plant buildings will be built or extended as part of the project. The location of the new process plant means that the existing fixed plant maintenance workshop and crib room will need to be demolished and rebuilt.

- New fixed plant maintenance workshop and crib room.
- New warehouse.
- Extension to the existing admin office.

18.4 Paste Fill Plant Project Infrastructure

18.4.1 Paste Fill Plant Binder Consumption

Based on the target strength requirements from Section 13.17.1 and the results from the laboratory UCS test program, as outlined in Section 13.16.7 a summary of binder requirements is provided in Table 18-2. Binder requirements are estimated for two different paste consistencies, 175 mm and 250 mm slump. Typically, a higher slump paste is used to reach stope locations that are further away from the paste pump, which would otherwise require an excessive amount of pressure to deliver if it was a lower slump paste.

Table 18-2 Summary of Estimated Binder Requirements Based on Paste Design Strength

Exposure Type	Exposure width (m)	Hydration Period (days)	Design Strength kPa (FoS = 1.5)	90:10 Slag Cement Required (175mm slump)	90:10 Slag Cement Required (250mm slump)
Simultaneous Underhand and Vertical	0-5	28	250	3	4
	5-10	28	300	3.5	4.5
	10-15	28	450	4	5
	15-20	28	500	4	5
Vertical Exposure	0-5	14	150	3.5	4.5
	5-10	14	150	3.5	4.5
	10-15	14	200	4	5
	15-20	14	200	4	5
	20-25	14	200	4	5
	25-30	14	250	4.5	5.5

18.4.2 Real Estate and Site Layouts

To improve underground geotechnical stability, facilitate increased ore extraction, and to dispose of approximately 74% of the Kainantu tailings underground, cemented paste backfill manufactured from thickened concentrator tailings is used as the mine backfilling method. The system is designed to use approximately 74% of the instantaneous available tailings, with a nominal paste production rate of 108 m^3/h and a design rate of 148 m^3/h to facilitate sprint production when required. The nominal production rate is expected to satisfy the peak underground backfill demand of 46,298 m^3/month, at a utilization rate of 59%. The utilization rate is typical for paste backfill systems and includes downtime for reticulation system changes, filter cake stockpile buildup, and regular maintenance.

The average stope size (~2,400 m^3) is expected to be filled in a single pour campaign taking into account stope bulkheads and exclusion zones, which will not require downtime associated with fill cure periods and which will accelerate stope turn-around times. Larger sized stopes may require a series of pour campaigns to complete but these are expected to be relatively few compared to the amount of average sized stopes.

As part of the stope extraction sequence, paste is to be exposed both vertically and horizontally. Due to the narrow stope widths, relatively low paste strengths (typically less than 550 kPa) are typically required. A paste cure period of 28 days is used to achieve this relatively low strength requirement and indicates a binder content of 5% should be used. Binder content may be required to vary up to 9% for certain mining areas and pumping distances. Optimization studies show the optimal binder product to be predominantly ground granulated blast furnace slag (slag) with GP cement addition, mixed at a mass ratio of 90:10 slag:cement. Being a waste product, the predominant use of slag (in the binding agent) presents a favourable solution from an environmental as well as cost perspective.

The operating philosophy of the Paste Fill Infrastructure was modified from the original feasibility design which pumped thickened slurry from the mill area to a processing facility located at the mine portal. The basic engineering design shifted to trucking pressure filtered tailings from the mill area leveraging the back haul cycle of the ore haul fleet to a staging area near the village located near the mine. The staging area colloquially known as the "bus stop" area will act as a tailings stockpile to decouple the operation of the mill from the capability to produce paste backfill. Decoupling the backfill system from mill and filtration operations will allow operations to improve plant availability / operability. The bus stop will serve as a staging area for both filter cake and binder storage to service the backfill plant operation.

From the bus stop a dedicated fleet of tailings haul trucks will transport the filter cake to the underground paste plant. In the underground plant the tailings will be mixed with water and binder slurry to form the backfill. The mixed backfill will be pumped to the stope using twin piston diaphragm pumps with one operating and one on standby.

This design approach requires three distinct infrastructure areas to produce paste backfill:

- Tailings filtration.
- Bus Stop Area.
- Underground Backfill Plant.
- And Underground Distribution System.

Tailings Filtration

With the change to implementing haul truck transportation of the filtered tailings to feed the backfill system a pressure filtration plant is a necessary inclusion as part of the system. WSP engaged in analysis of the K92ML tailings stream and determined that filter cake consistency of nominally 88% solids is necessary to achieve transportability without liquefaction of the tailings from road transport induced vibrations in the truck bed. Such a high solids content was not achievable with vacuum filtration equipment thus necessitating the use of the more expensive pressure filtration technology which will produce nominally 86-88% solids filter cake.

The filtration building is designed to receive tailings from the tailings thickener underflow pumps located at the mill into the filter feed tank. The filter feed tank supplies two pressure filters with feed from a pressurized feed loop which returns to the filter feed tank for pressure control. The pressure filters discharge cake into storage bunkers which a loader will retrieve filter cake for loading onto haul trucks for back haul to the bus stop storage area.

The filtration plan will include a dedicated motor control center, control room, compressed air system, water containment and distributions system, and waste storage and discharge systems. This facility will be located near the mill. An isometric view of the of the filtration plant is present in Figure 18-1 and Figure 18-2.

The filtration plant is designed with a processing throughput of 106 tph which supplies sufficient filtered tailings for paste backfill production based on 85% pressure filter utilisation. Filtered tailings will need to be stockpiled at the bus stop area to maximum capacity prior to each pour to mitigate the possibility of tailings shortage. This is because the underground paste plant operates at a higher throughput compared to the filtered tailings plant. However, the paste plant operates for shorter periods of time by comparison, allowing the filtered tailings plant to catch up. In this way, the production capacity of the filtered tailings plant is optimised, tailored to the needs of the paste plant.

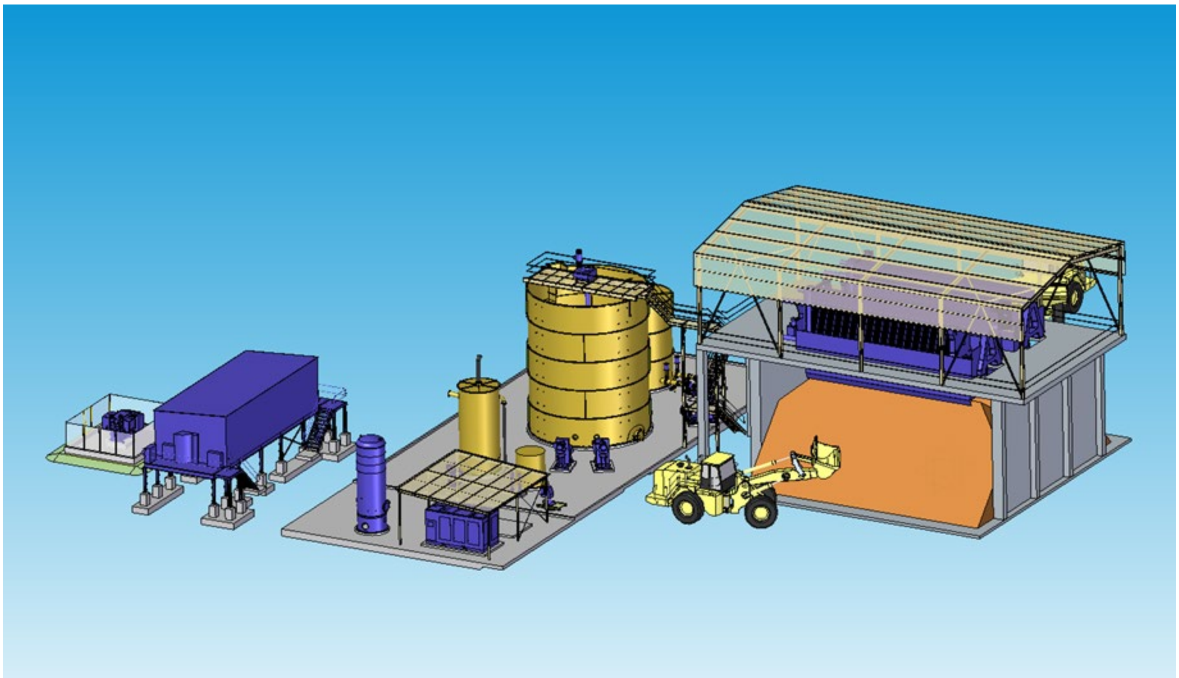


Figure 18-1 Filtration Plant 3D Models View From SE

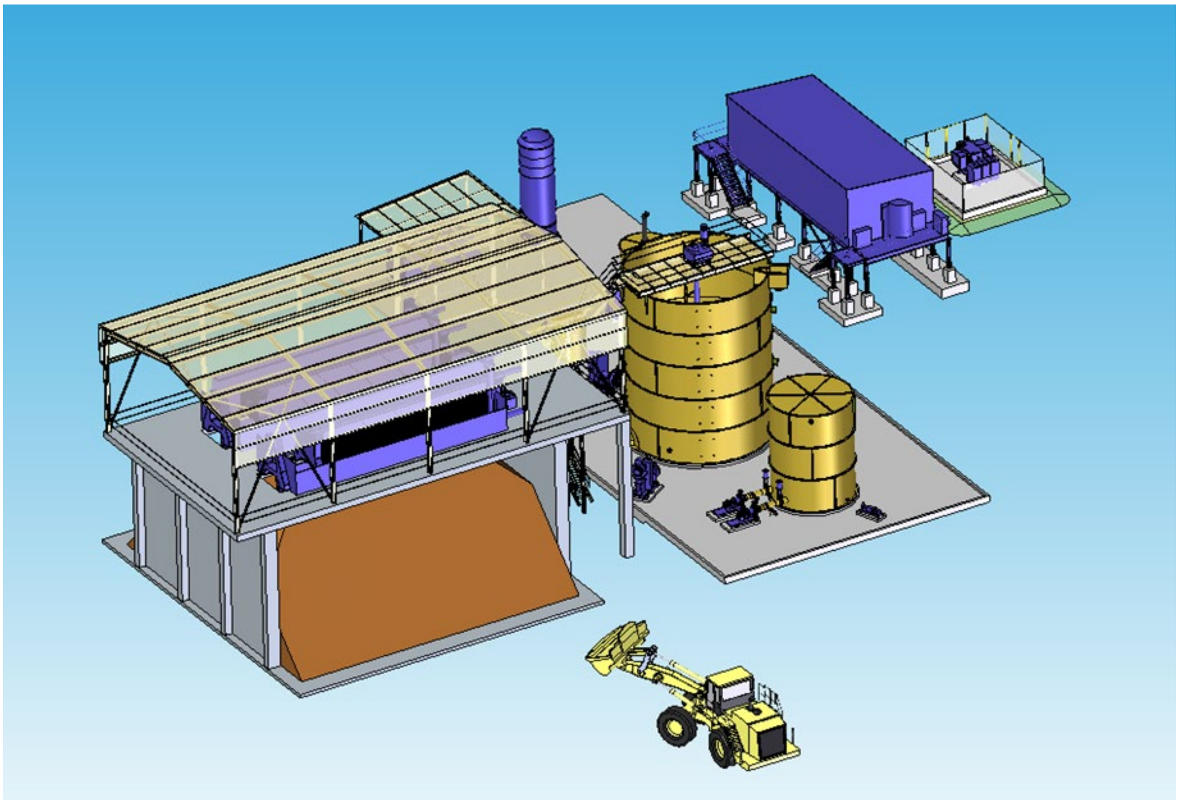


Figure 18-2 Filtration Plant 3d Model View from NW

Tailings Transfer Area ("Kokomo Bus Stop Area")

The Kokomo Bus Stop Area operates as a staging area for the underground backfill plant. This design minimizes the amount of underground infrastructure required and decouples the underground operation from the mill and filtration plants. There are two main functions to the bus stop which require careful traffic management and planning to ensure that they do not interfere with one another. These functions are filter cake storage and handling with haul truck washing, and binder storage for distribution to underground.

Haul trucks loaded with filter cake will enter the Bus Stop from the East towards the North end of the facility. The trucks will offload the filter cake into a covered storage area where a front loader will shape the stockpile into 3-meter-high piles. Upon existing the bus stop the haul trucks will pass through a wash bay which will rinse remaining filter cake from the truck bed to reduce fouling of the ore feed stream into the mill.

Another front-end loader will draw filter cake from the South side of the stockpile to load haul trucks which will enter the facility from the East near the South corner of the area. These haul trucks will transport the filter cake from the stockpile to the underground backfill plant. Separating the load and off load functions of the pile in a North South fashion will reduce vehicle traffic conflicts and improve operational efficiency and safety.

The binder storage facility will be in the West end of the bust stop area. Binder delivery trucks will enter the area from the same point as the filter cake delivery haul trucks. The binder trucks will continue West past the cake stockpile and park next to the binder system for pneumatic offload of the binder. Binder will be delivered in two forms, slag and normal Portland cement. The two constituents will be delivered to separate silos and the system will be configured to mix both parts in a 90:10 ratio of slag to cement.

To the south of the binder system a parking lot for idle trailer mounted isotainers will be created. Operators will take empty isotainers and move them to the S side of the binder system where they will be filled with the 90:10 mixture. Once filled the isotainer will be moved back to the parking lot where it will wait for operators with a trailer truck will retrieve them for transportation to the underground backfill plant. The truck operators will bring empty isotainer trailers back to the bus stop area for refilling on their return trip from the plant. Isotainer trailer operators will use the Eastern entrance to the bus stop area for their haul route.

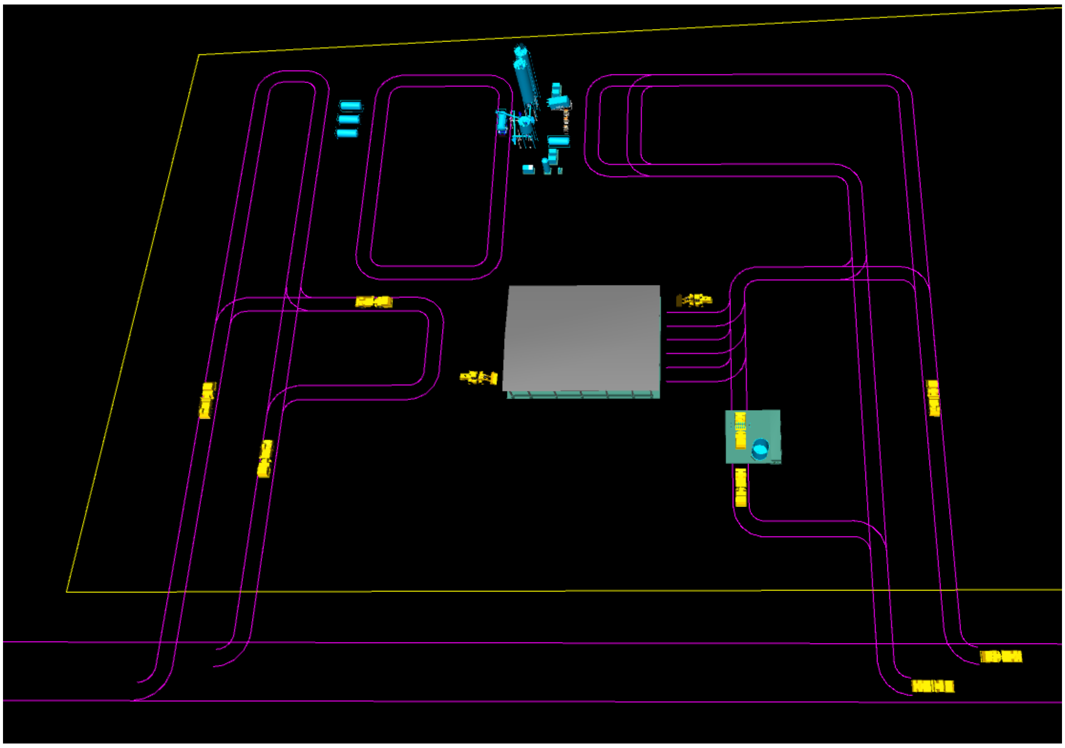


Figure 18-3 Kokomo Bus Stop Area Layout with Traffic Routes

Underground Backfill Plant

Filter cake will be delivered to the underground backfill plant on the 1225 level to a raise bored storage excavation. Cake will discharge through a Kamengo style reciprocating feeder system to a weigh conveyor. From the weight conveyor material will drop into the paste mixer where it will be mixed with water and binder slurry to create the backfill.

The binder slurry will be mixed in a colloidal mixer system. The colloidal mixer system will be comprised of a horizontal binder silo, two (2) colloidal mills and pumps. The binder will be transferred pneumatically to the horizontal silo from isotainers which will be parked in a nearby cut out. The colloidal mills will be used to create a 50:50 binder to water mixed slurry delivered to the paste mixer using the colloidal mill pumps.

The paste mixer will operate in a overflow condition where mixed backfill overflows from the discharge end of the mixer to a paste hopper. The paste hopper will be used to direct the paste to a borehole which will connect the mixing level to the pumping level below. A diverter valve will direct the paste to one of two piston diaphragm pumps which will feed the Underground Distribution System (UDS).

In addition to the main process circuit the plant will be equipped with a water storage sump on the upper level which will function as a process water tank. The plant will also have a stand-alone compressed air system. Waste material from the plan will be collected in a dual sump system on the lower level. One sump will be kept in active state while the other sump is allowed to consolidate and settle. Settled solids will be excavated from the inactive sump after water is decanted and then the sump will be returned to service. Waste solids are expected to be disposed of in noncritical stopes.

The backfill plant is designed to mix paste for slumps ranging from 175 mm to 250 mm with typical binder contents between 5 to 9%.

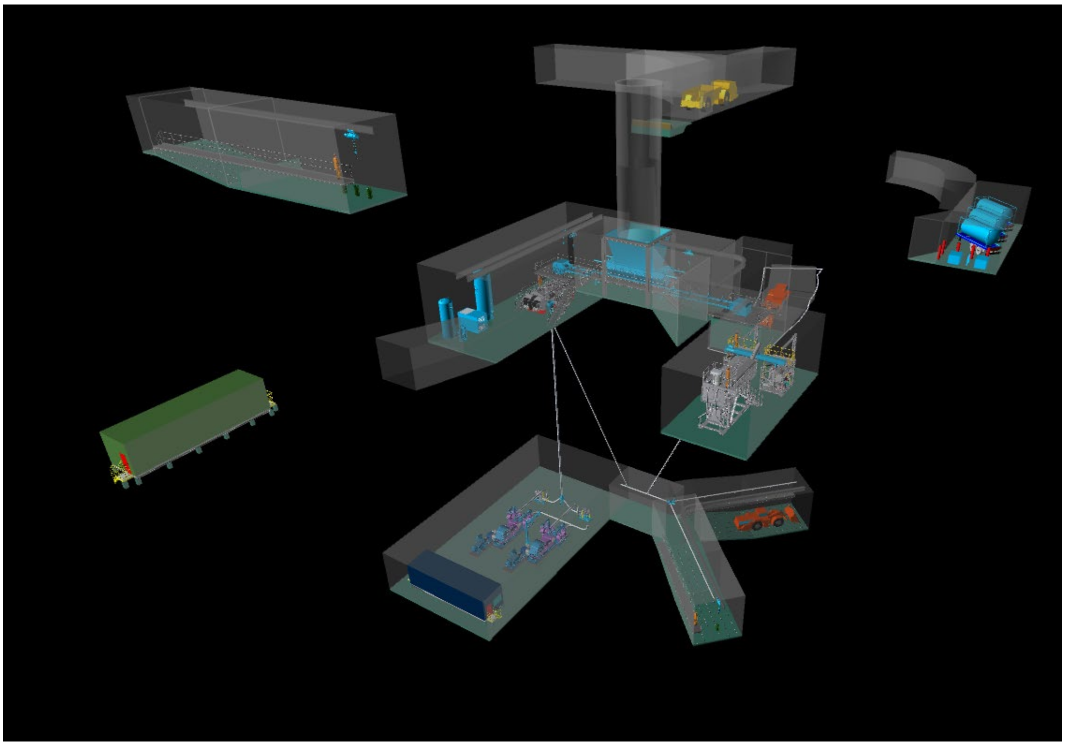


Figure 18-4 Underground Backfill Plant View from SE (without access drifts shown)

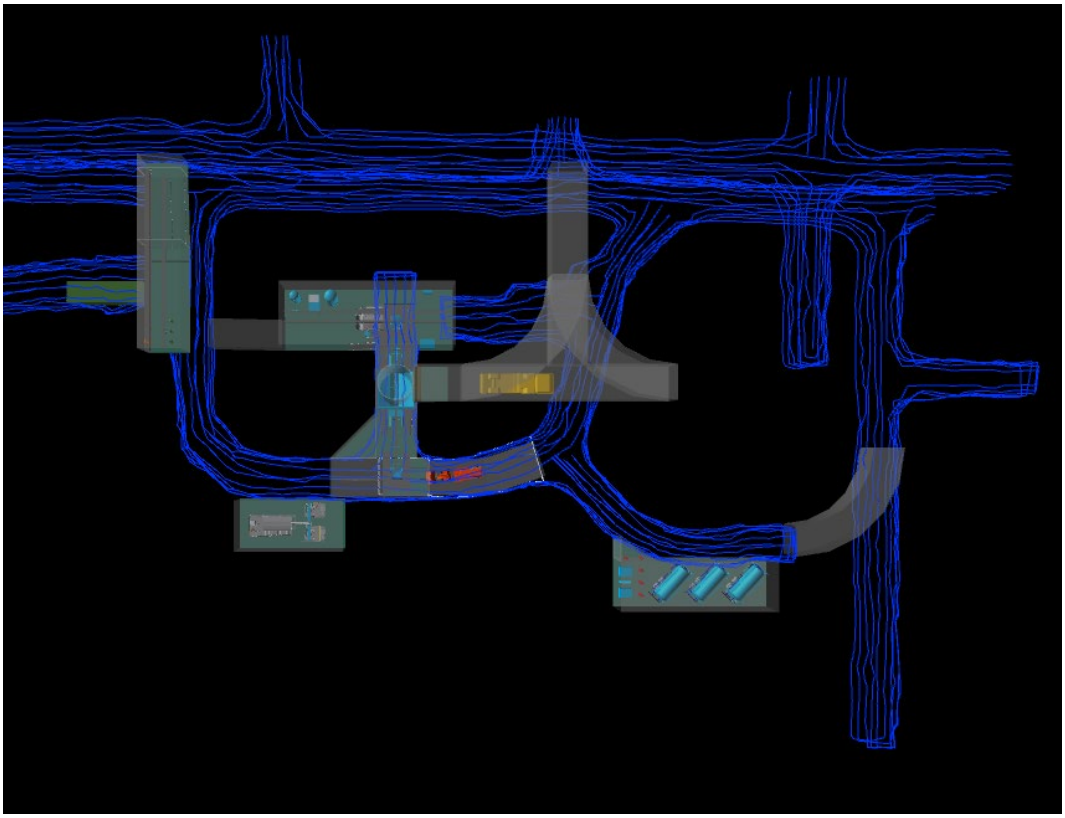


Figure 18-5 Paste Plant Infrastructure Top View (with access drifts shown)

Underground Distribution System (UDS)

The UDS has been designed to optimize pipe installation requirements while achieving reasonable wear rates in boreholes and other primary trunk portions of the UDS system. A primarily 200 mm nominal diameter pipe system has been specified with the use of 250 mm nominal diameter boreholes for rising trunklines and 200 mm nominal diameter for descending trunklines. The rising trunklines were upsized to minimize piping pressure losses and allow for the system to only need one booster station to service the upper portions of the orebody with backfill. Initial calculations show that the entirety of the proposed DFS mine layout can be serviced with the pumps located in the paste plant and a single booster station located on level 1170.

18.4.3 Major Equipment Specifications

Each section of the facility, as described in the sections above, has been fully developed with procurement ready specifications for all major pieces of equipment. Procurement activities are ongoing, however most major pieces of equipment have been selected and procurement contracts are in place or in negotiations. The remaining equipment is expected to be procured during the detailed engineering phase of the project design cycle.

Table 18-3 outlines the list of equipment for the project by area code. Note that the area codes are presented as follows:

- 2511 - Filtration Plant
- 2512 - Bus Stop Area
- 2513 - Underground Paste Plant

Table 18-3 Paste Plant Equipment List

Equipment Number			Equipment Name
Area Code	Equip. Code	No.	
2511	FL	001	Pressure Filter 1
2511	FL	002	Pressure Filter 2
2511	HU	001	Hydraulic Pack 1
2511	HU	002	Hydraulic Pack 2
2511	TK	001	Filter Feed Tank
2511	AG	001	Filter Feed Tank Agitator
2511	PP	001	Filter Feed Pump 1
2511	PP	002	Filter Feed Pump 2
2511	TK	002	Process Water Tank
2511	PP	003	Flood Wash Pump 1
2511	PP	004	Flood Wash Pump 2
2511	PP	007	Process Water Pump 1
2511	PP	008	Process Water Pump 2
2511	TK	003	Seal Water Tank
2511	FL	004	Seal Water Filter 1
2511	FL	005	Seal Water Filter 2
2511	PP	009	Seal Water Pump 1
2511	PP	010	Seal Water Pump 2
2511	TK	004	Waste Water Pump Box
2511	PP	011	Waste Water Pump 1
2511	PP	012	Waste Water Pump 2
2511	PP	013	Filtrate Sump Pump 1
2511	PP	014	Filtrate Sump Pump 2
2511	PP	015	Filter Wash Sump Pump 1
2511	PP	016	Filter Wash Sump Pump 2
2511	PP	017	Tank Area Sump Pump
2511	CP	001	Filter Air Compressor 1
2511	VP	002	Filter Air Receiver 2
2511	DR	001	Instrument Air Dryer
2511	VP	004	Instrument Air Receiver
2512	BL	001	Slag Blower 1
2512	BL	002	Slag Blower 2
2512	BL	005	Cement Blower 1
2512	BL	006	Cement Blower 2
2512	SI	001	Slag Silo 1
2512	SI	002	Slag Silo 2
2512	SI	003	Cement Silo
2512	DC	001	Slag Silo Bin Vent 1
2512	DC	002	Slag Silo Bin Vent 2
2512	DC	003	Cement Silo Bin Vent
2512	VA	001	Slag Silo Spheri Valve 1
2512	VA	002	Slag Silo Spheri Valve 2
2512	VA	003	Cement Silo Spheri Valve
2512	CV	005	Slag Screw Conveyor
2512	CV	006	Slag Incline Screw Conveyor
2512	CV	007	Cement Screw Conveyor
2512	CV	008	Cement Incline Screw Conveyor
2512	FE	001	Slag Mass Flow Meter

Equipment Number			Equipment Name
Area Code	Equip. Code	No.	
2512	FE	002	Cement Mass Flow Meter
2512	DC	004	Binder Area Dust Collector
2512	VA	004	Binder Area Dust Collector Rotary Valve
2512	EF	001	Binder Area Exhaust Fan
2512	XM	001	Container Tipper
2512	HU	005	Container Tipper Hydraulic Pack
2512	CP	004	Binder Area Air Compressor
2512	DR	002	Binder Area Air Dryer
2512	VP	005	Instrument Air Receiver
2512	TK	005	Wash Bay Tank
2512	FL	006	Wash Bay Water Filter
2512	PP	025	High Pressure Wash Pump 1
2512	PP	034	High Pressure Wash Pump 2
2512	PP	036	Wash Bay Sump Pump 1
2512	PP	037	Wash Bay Sump Pump 2
2513	BL	003	Isotainer Offload Blower 1
2513	BL	004	Isotainer Offload Blower 2
2513	SI	004	Horizontal Binder Silo
2513	CV	009	Reversible Screw Conveyor
2513	CV	010	Incline Scew Conveyor
2513	MX	002	Colloidal Mixer 1
2513	MX	003	Colloidal Mixer 2
2513	DC	005	Paste Area Dust Collector
2513	LB	001	Live Bottom Feeder
2513	HU	004	Live Bottom Feeder Hydraulic Pack
2513	CV	011	Feeder Conveyor
2513	PP	018	Process Water Pump 1
2513	PP	019	Process Water Pump 2
2513	PP	031	Flush Water Pump
2513	MX	001	Mixer
2513	PP	020	High Pressure Wash Pump
2513	HP	003	Paste Hopper
2513	PP	028	Charge Pump 1
2513	PP	029	Charge Pump 2
2513	PP	023	PD Paste Pump 1
2513	PP	024	PD Paste Pump 2
2513	PP	027	Paste Pump Area Sump Pump
2513	CP	005	Paste Area Air Compressor
2513	AD	003	Paste Area Air Dryer
2513	AR	006	Paste Area Instrument Air Receiver
2513	AR	007	Paste Area Plant Air Receiver
2513	HV	001	Paste Diverter Valve
2513	HV	003	Sump Diverter Valve
2513	CN	003	Mixer Crane 1
2513	CN	004	Mixer Crane 2
2513	CN	005	Paste Pump Area Sump Crane
2513	CN	006	Feeder Crane
2513	CN	008	Process Water Sump Crane

18.5 Tailings Storage Facility

The Kora Expansion DFS scenario forecasts that approximately 5.89 million tonnes of tailings will be generated from 2024 to 2030, as detailed in Table 18-4. By end-2025, the Paste Backfill Plant will begin operations, utilizing approximately 35% of the tailings to fill underground voids in its first year. Starting in 2026, when the plant reaches full operational capacity, it is projected that over 70% of the tailings will be used for this purpose. The remaining will continue to be deposited in the existing TSF. In the last years, a portion of the tailings will be stockpiled to be used as dry backfill into the remaining underground voids in the final year of the LOM.

Table 18-5 details the total tonnes and volume of tailings required to be stored in the TSF from 2024 to 2030.

Table 18-4 Kora Expansion DFS Production Forecast

Year	Mill Production (t)	Tailings generated (t)	Tailings to TSF (t)	Tailings to Paste Backfill Plant (*) (t)
2024	473,004	450,419	450,419	0
2025	799,723	767,068	502,005	265,062
2026	1,000,725	960,518	280,738	679,779
2027	1,200,000	1,140,144	227,206	912,939
2028	1,200,000	1,126,617	35,025	1,066,203
2029	1,200,000	1,166,262	0	1,166,261
2030	302,268	280,010	60,010	0
Total	6,175,720	5,891,037	1,555,404	4,090,244

(*) Full operational capacity of the Paste Backfill Plant is anticipated by 2026.

Note: *The cut-off grade scenario for the Tailings Storage Facility (TSF) has been assessed at* ***3.5 g/t****, which serves as the base scenario. Scenarios with lower grades require reevaluation as they may exhaust the TSF capacity before the planned production end of the Definitive Feasibility Study (DFS).*

Table 18-5 DFS Forecast Tailings Production Into the TSF

Year	Tailings to TSF (tonnes)	Volume of Tailings to TSF (m^3) (*)
2024	450,419	333,644
2025	502,005	371,856
2026	280,738	207,954
2027	227,206	168,300
2028	35,025	25,944
2029	0	0
2030	60,010	44,452
Total	**1,555,404**	**1,152,151**

(*) Calculations assume an average settled density of tailings of 1.35 t/m^3.

The existing TSF, situated downstream of the process plant adjacent to Kumian Creek, which flows into the Baupa River, as illustrated in Figure 18-6, has already accommodated approximately 1.15 Mm^3 of tailings. Future raises of the TSF embankment, along with their corresponding volume and remaining capacity for tailings storage, are presented in Table 18-6.

The detailed engineering design to raise the embankment of the Tailings Storage Facility to Stage 2, reaching an elevation of 520mRL, has been completed. In contrast, the design for raising the embankment to Stage 3, which would reach 530mRL, is still at the conceptual stage.

Figure 18-6 Existing TSF

Table 18-6 Capacity for Tailings Volume with Embankment Raises

TSF Raise Stage	Crest Level (RL m)	Volume Capacity (m^3)	Remaining capacity (m^3)
1C	517	1,530,000	384,480
2	520	1,529,959	724,480
Interim stage	525	2,500,000	1,354,521
3 (Conceptual)	530	3,150,000	2,004,480

18.5.1 Tailings Storage Capacity for DFS Scenario of Kora Expansion Project

In the DFS scenario for the Kora Expansion project, it is proposed to raise the embankment crest level of the existing TSF to 525mRL to increase the facility's capacity. This configuration will accommodate all tailings generated from the production plan outlined in this scenario and allows for potential additional raises up to Stage 3 (530mRL), that is the maximum elevation currently considered as feasible.

The existing TSF, with its embankment raised to Stage 2 - design currently at a detailed level- will not be sufficient to accommodate the forecasted tailings volume from the DFS scenario production plan. However, a further increase in capacity from raising the embankment to 525mRL will meet the storage needs throughout the mine's operational lifespan, as shown in Figure 18-7. The expected annual volume of tailings and the incremental storage capacity provided by each stage of the embankment raise are detailed in Figure 18-7.

Key assumptions for extending the tailings capacity to 525mRL are as follows:

- **Underground Void Storage:** More than 70% of the tailings generated post- 2025 will be stored in underground voids, leveraging the capacity of the newly commissioned paste backfill plant.
- **Tailings Particle Size:** There will be no particle size selection for tailings directed to the TSF or the paste backfill.
- **Slurry Deposition Method:** Tailings directed to the TSF will continue to be deposited via the conventional slurry method.
- **Settled Density**: The average settled density of the tailings should be maintained at the average of 1.35 t/m^3.
- **Borrow Material Availability:** Sufficient borrow material (rockfill) is available for constructing the dam up to Stage 3 (530mRL).

Construction Sequence

During the DFS production period, embankment raises should be continuous, with a yearly increase in the crest level of the dam to accommodate the increasing volume of tailings. Figure 18-7 depicts the sequence and timing of the embankment raises, aligned with the production schedule and tailings volume forecasts.

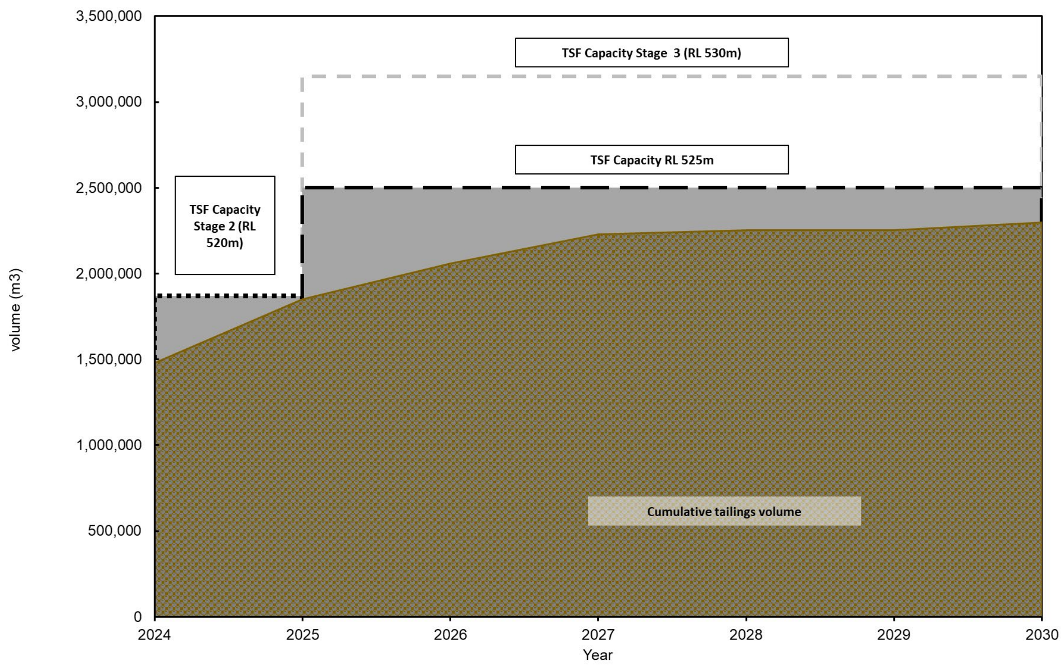


Figure 18-7 Tailings Volume to TSF and Storage Capacities

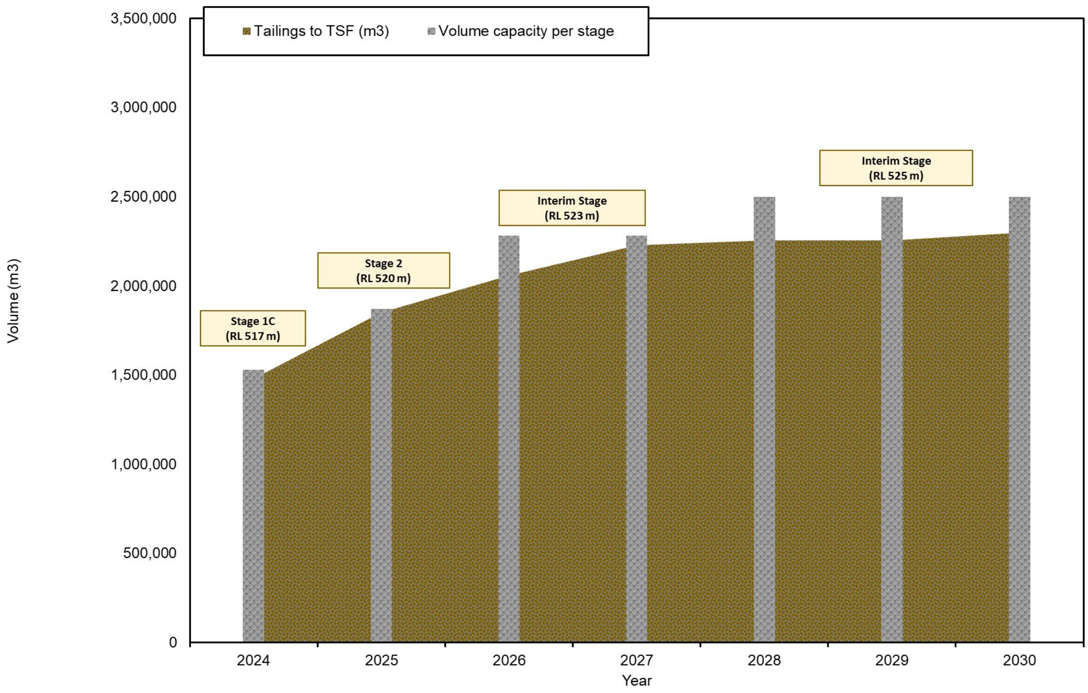


Figure 18-8 TSF Embankment Raises for DFS Production

18.5.2 Tailings Management

The tailings placed in the existing TSF primarily consist of silty, sand-size particles from the flotation circuit. These are largely composed of inert materials such as quartz and waste rock sand, with a minor fraction of sulfur-bearing minerals. Despite possessing acid-producing potential, routine inspections indicate that water quality remains within acceptable pH parameters. This stability is attributed to the site's excessive rainfall and the addition of lime in the processing plant, which raises the pH to 10.

The current tailings handling practices consider tailings from the last scavenger flotation cell are transported to the TSF through an HDPE pipeline at an average of 34% solids content by weight. They are deposited via a spigot pipeline positioned at 25-meter intervals along the dam crest. An average settled density of 1.35 t/m^3 is anticipated. However, variability in this density is expected due to several factors, including initial solids concentration, rainfall, seismic activity, and overall tailings management practices.

The water management strategy for the TSF ensures that water released from the tailings due to settling and consolidation is recycled back to the process plant. In conditions of water surplus, typically following rainfall, excess water is pumped to a dual-pond overflow wetland system-comprising anaerobic and aerobic ponds-before being safely discharged into Kumian Creek.

The embankment designs are developed to accommodate maximum levels of tailings and water, ensuring stability under all operational conditions. These designs aim to maintain operational loads within the safe limits specified in the design criteria.

Key principles of tailings management include:

- Reduction of Free Water: Minimizing free water within the impoundment and keeping it away from the embankments to ensure the tailings remain unsaturated.
- Maximization of Settled Density: Implementing various strategies to increase the density of settled tailings, such as:
  - Effective spigot management and deposition control to reduce water ponding.
  - Enhancing settling behavior through tailored management of particle sorting along the beaches of the TSF.

- Periodic evaluations for the potential installation of wick and carpet drains to facilitate water drainage and speed up consolidation.
- Considering implementing in-line dosing of coagulants and flocculants to improve the settling behavior of tailings.
- Considering the use of engineered water evaporators to enhance water loss during the dry season, thereby aiding further consolidation of tailings and reducing water levels before the rainy season.

The TSF is licensed to operate under an amended Environment Permit EP-L3(34) issued by the Papua New Guinea Conservation and Environment Protection Authority (CEPA) in November 2023. The Environment Permit provides guidance on operation of structures to industry standards, including completion of Annual Dam Safety Reviews.

18.6 River Crossings

Three river crossings over the Baupa river will be upgraded as part of the Stage 3 expansion project scope. The crossings are on the current haul road from the process plant up to the 800 Portal, located within the mine lease.

- Kokomo bridge.
- Kasese crossing.
- Lower Baupa bridge.

18.7 Accommodation Camp

The Kumian Camp could originally accommodate 850 personnel after additional accommodation units for both COVID-19 quarantine capability and increased personnel were installed and constructed during 2019 and 2020 along with the construction of a new kitchen and mess. The camp also contained a health / first aid clinic for the benefit of all onsite personnel. In addition, there are recreation facilities including gym, covered basketball court and sports grounds.

Camp upgrades have been made as part of the Stage 3 expansion which includes the construction of five additional accommodation units, two twenty person single ensuites, a HR training and induction centre. A new community affairs building, an OHS department building, extension to the mess and kitchen, improving the recreational facility and upgrading the camps water, power, communications and sewage systems are currently underway due for completion in 2025.

These upgrades will aid the increase the camp capacity from 850 in 2022 to more than 1,850 beds in 2025.

19. MARKET STUDIES AND CONTRACTS

During 2019, K92ML entered into an offtake agreement with Trafigura covering a 9-year period ending 11 February 2028 or until 165,000 dry metric tonnes ('DMT') of concentrate have been delivered. If the minimum DMT has been delivered during the 9-year period, K92ML is only required to sell 50% of its annual production until the end of the term. The terms provide for payment of gold, silver and copper contained in the concentrate.

During 2024, K92ML entered into a new offtake agreement with Trafigura, that will supersede the above agreement beginning January 1, 2026, covering a 7-year period ending 31 December 2032 or until 600,000 DMT of concentrate have been delivered, whichever occurs sooner. The terms provide for payment of gold, silver and copper in the concentrate.

Under the new offtake agreement, Trafigura will purchase gold and copper concentrates from the Kainantu Gold Mine in Papua New Guinea at London Metals Exchange spot prices with terms more favourable than those in the current offtake agreement at a reduced cost.

20. ENVIRONMENT STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

The environment and community aspects relating to the Kora Expansion Project are discussed in this section. This includes a concise analysis of key environmental and community aspects associated with the project, including relevant statutory requirements and a summary of environmental and community impacts.

All findings, conclusions or recommendations set out in this section are an interpretation of the information and data available at the time of writing using professional judgement and opinion having due regard to the assumptions that can be reasonably made. This section should be reproduced in full. No responsibility is accepted for use of any part of this section in any other context or for any other purpose or by third parties.

20.2 Legal and Regulatory Requirements

The environment and community assessment framework applicable to the Kora Expansion Project Definitive Feasibility Study ("DFS") includes regulatory and policy requirements of Papua New Guinea ("PNG"). National Government regulatory bodies, such as the Mineral Resources Authority ("MRA") and the Conservation and Environment Protection Authority ("CEPA") have granted various approvals, permits and licenses required by K92MLto operate the Kainantu Gold Mine, and these approvals have been subject to ongoing review and amendment for the approval of the Kora Expansion Project.

20.2.1 Mining Act 1992

The Mining Act 1992 ("Mining Act") is the principal regulatory instrument governing the mining industry in PNG and is administered by the MRA. The Mining Act vests ownership of all minerals in or below the surface of land (or under the sea) with the national government, and governs the exploration, development, processing, and transport of minerals.

K92ML existing mining lease (ML150) contains approximately two-thirds of the resource to be mined as part of the Kora Expansion Project. To mine the remaining resource, an extended or new mining license will be required for issuance by the MRA. Per the mining schedule outlined for the Stage 3 Expansion, this is scheduled to be obtained by 1 January 2027. The Company plans to apply for this extended or new mining lease in H1 2025.

There was an amendment to the Mining Act on 10 June 2020. The Minister for Mining presented the Mining (Amendment) Bill 2020 (the "Bill") and the Bill was passed by Parliament. The purpose of the Bill is to provide a legal basis for the PNG Government to apply for a tenement and develop a mine. The key amendments are related to the PNG Government's rights over reserved land as well as live

data reporting obligations by operators of mines to report daily production and earnings results.

The Bill was certified on the 26 June 2020 and is now an Act of Parliament and is in force.

20.2.2 Environment Act 2000

CEPA administers the Environment Act 2000 (the "Environment Act"), the primary legal and regulatory framework for environmental protection in PNG, including regulating the environmental effects of mining (and other) projects. Environmental permits are issued and administered under the associated Environment (Permits) Regulation 2002 and the Environment (Prescribed Activities) Regulation 2002. Permits are granted according to various levels, ranging from Level 1 ("L1", limited potential impacts) to Level 3 ("L3", greater potential impacts). Large-scale mining activities are authorized under L3 environmental permits, for which an Environmental Impact Statement ("EIS") is required for submission to CEPA for approval. The results of the EIS form the basis of associated environmental permit conditions.

K92ML has operated under Environment Permit EP-L3(34) for ongoing mining and construction activities at the Kainantu Gold Mine since it acquired the mine in 2015. The permit conditions were transferred to K92ML from the previous operators of the mining operation and have been subject to various amendments based on operational changes over time.

Under Section 71 of the Environment Act, a proponent may submit an environmental permit amendment application to address changes to its permitted activities. This was determined to be a requirement for the Kora Expansion Project given project changes and expected increases in associated environmental and social impacts. Following submission of the amendment application, the Director of CEPA determines whether the change constitutes a 'major amendment' or a 'minor amendment', which, in turn, enables the determination of the scope of an associated EIS. The Environment (Permits and Transitional) Regulation 2002 indicates that:

- A 'major amendment' is for an activity that is likely to result in a substantial increase in the risk of serious environmental harm under the amended permit because of a substantial change in (a) the quantity or quality of contaminant permitted to be released into the environment; or (b) the results of the release of a quantity or quality of contaminant permitted to be released into the environment.
- A 'minor amendment' is any other amendment.

20.2.3 EIS Preparation, Approval Process and Status

To support the determination by CEPA of the required environmental permit amendments for the Kora Expansion Project, K92ML was required to submit an Environmental Inception Report ("EIR") under Section 52 of the Environment Act, the objectives of which were to:

- Identify the potential environmental and community issues of developing the [expansion] project.
- Describe the scope of the EIS to address these issues.
- Initiate the formal process of stakeholder consultation.
- Enable CEPA to review the proposed EIS scope and provide feedback regarding perceived shortcomings.
- Submit an appropriate EIS under Section 53 of the Environment Act.

The Kora Expansion Project considers a brownfield expansion to underground mining at Kainantu with on-site processing of mine material by conventional milling, gravity, and flotation recovery through a standalone 1.2-million-tonne-per annum ("M tpa") process plant. The existing TSF is also proposed to sequentially raise the height of the TSF embankment beyond the original nominal design crest elevation of 509mRL to approximately 530mRL. A new paste plant, which will include re-purposing approximately 70% of tailings generated, is also scheduled for commissioning in mid-2025.

In early 2023, K92ML commissioned an expert third-party to develop the EIR, which was then submitted to CEPA in April 2023. Following review and approval of the EIR, CEPA determined that, due to the brownfield nature of the proposed Stage 3 Kora Expansion Project, no major amendments were required to the existing environmental permit (i.e. EP-L3(34)).

K92ML then commissioned the development of an EIS commensurate to the scope stipulated by CEPA based on the EIR. The EIS was prepared according to the Guideline for Conduct of Environmental Impact Assessment and Preparation of Environmental Impact Statement (DEC, 2004). It included an assessment of potential environmental and community impacts of the Kora Expansion Project and to describe how K92ML intends to avoid, manage, or mitigate these impacts.

This EIS was completed in H2 2023 for submission to CEPA and was subject to external review. Following this process, CEPA approved the EIS, which informed minor amendments to EP-L3(34). A revised EP-L3(34) was then issued to K92ML in November 2023 providing environmental authorization for the Stage 3 Kora Expansion Project. All existing environmental permits for the Kainantu Gold Mine have now been consolidated into EP-L3(34).

As part of the issuance of the revised EP-L3(34), CEPA advised that the Company should begin preliminary work for the permitting and construction of a new tailings facility. This is expected to include the application of a new Lease for Mining Purposes ("LMP") as well as a full EIS to support a major amendment to EP-L3(34).

The new tailings facility is proposed for the Stage 4 Expansion and is expected to be required in mid 2029. The environmental approval process is outlined in Figure 20-1 below.

The Kainantu Gold Mine currently operates under an Environmental Management and Monitoring Plan ("EMMP"). A revised EMMP is required for submission to CEPA for review and approval subsequent to the receipt of an amended environmental permit. As of the time of the release of this report, K92ML is developing the revised EMMP based on the results of the completed EIS and issuance of the revised EP-L3(34). Additional details related to the revised EMMP are provided in Section 20.6.

Figure 20-1 below provides a summary of the primary environmental permitting process in PNG.

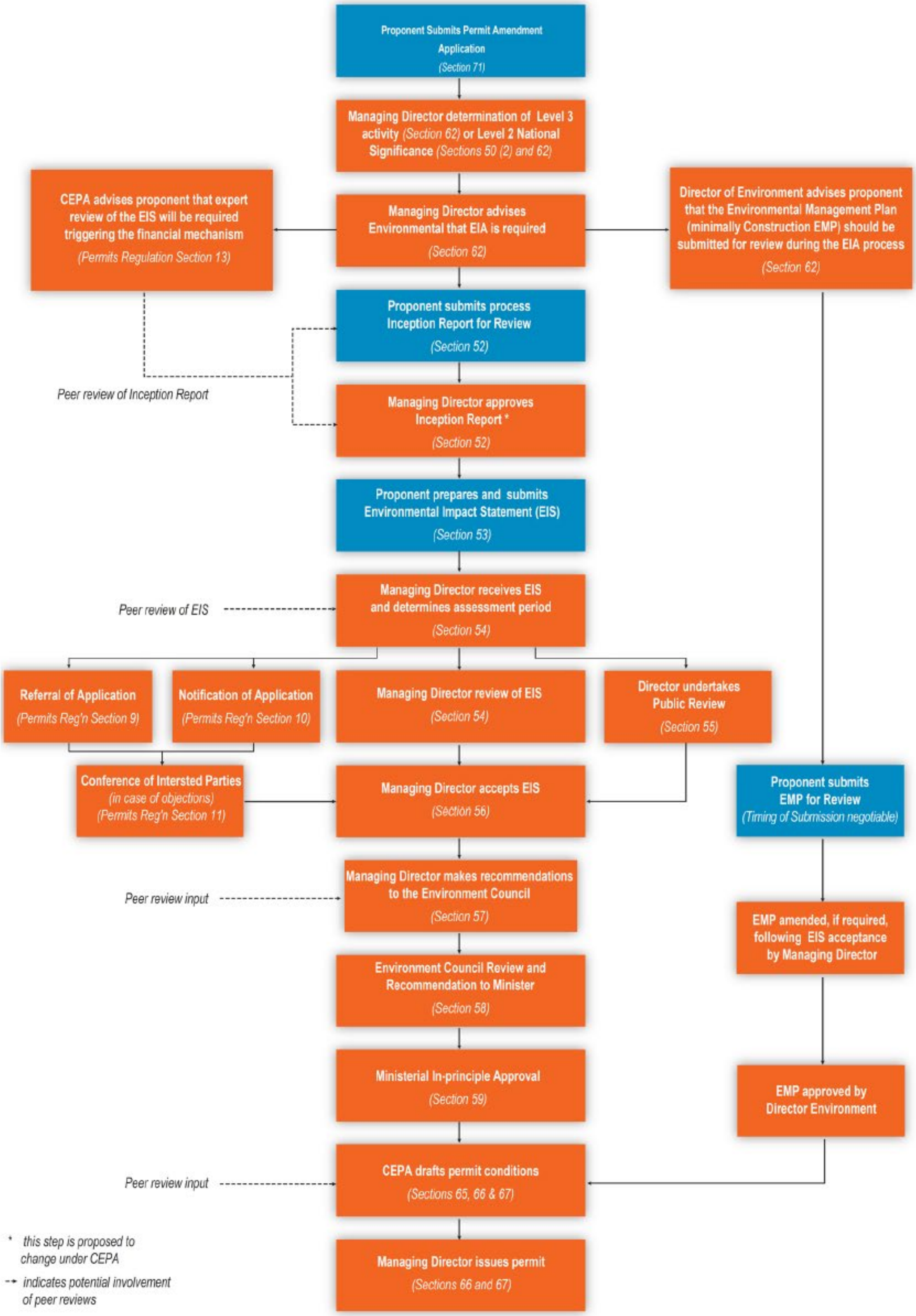


Figure 20-1 Summary of EIS Preparation and Approval Process

20.3 Environment and Community Setting

This section provides an overview of the key biophysical and community aspects within the existing Kainantu Gold Mine area. The existing operation is in an environment that has been modified through existing mining activities.

20.3.1 Geography and Climate

The Kainantu Gold Mine is in Kainantu District in the Eastern Highlands Province of PNG. It is accessible via the mine access road that turns off towards the Eastern Highlands from the Madang - Lae Highway at Gusap, Morobe Province. The access road crosses the Ramu River alluvial plain. The mine camp site, TSF and processing plant are in this area along the access road. The access road continues southwest into an area of more rugged mountainous landscape of ridge and ravine landforms with elevations from 400 m to 1,000 m above sea level (Loffler, 1974). The access road ends where the mine portals are located.

Kainantu has a cool, highlands climate, with an average monthly temperature of 13.5°C to 25°C. The warmer months are in October to May, and cooler months are in June to September. Kainantu has an average monthly rainfall of 102 mm to 317 mm. The wetter months are in October to April and drier months in May to September. Average monthly wind speeds at Kainantu range from 3.1 km/hr to 4.7 km/hr, with higher wind speeds experienced in May to October and lower wind speeds in November to April. The average visibility is 7 km to 9 km (Weather Atlas, 2021).

20.3.2 Surface Water

Much of the existing project infrastructure is located alongside Baupa and Kumian Creeks. Baupa Creek joins the adjacent Pumasi River and flows out to Ramu River. Surrounding small streams form a dendritic drainage system from surrounding hills and mountains draining into the Baupa Creek, making up the Baupa catchment. Kumian Creek joins Maniape Creek before it flows out to Ramu River. Kumian Creek is part of the Maniape catchment (Figure 20-2).

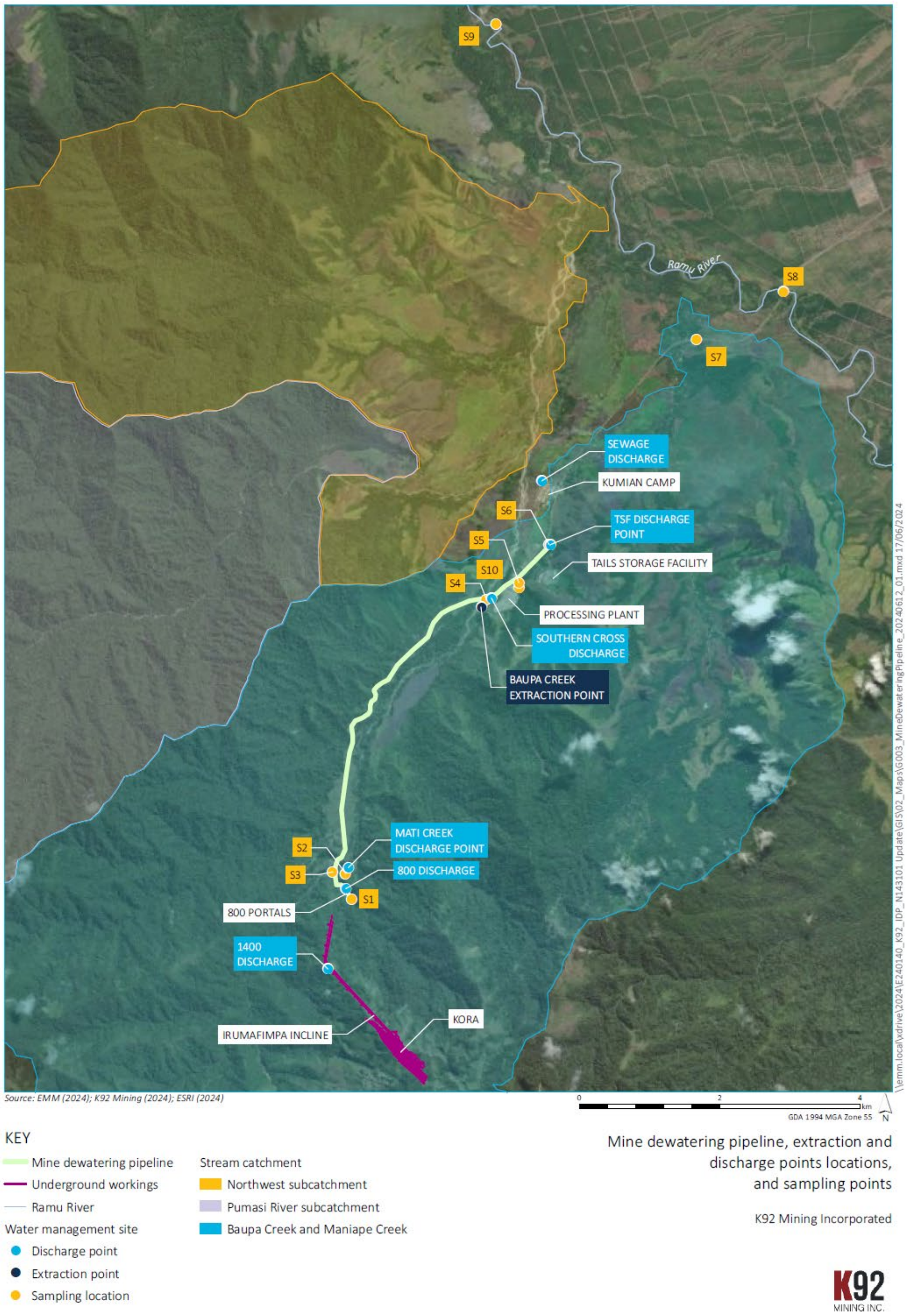


Figure 20-2 Surface Water Catchments and Water Quality Sampling Sites

Surface Water Quality

Water quality monitoring locations for the Kainantu Gold Mine are situated in three catchment areas - Baupa catchment, Maniape Catchment and at Ramu River where the access road crosses. Water quality monitoring per the mine's EMPP has been undertaken or commissioned by K92ML. Results have been reported in annual reports from 2004 to the present and will continue as per license conditions.

The annual mean concentrations of most trace metals for the 2023 monitoring period were within the permissible limits described under the existing Environment Permit EP-L3(34) (K92ML, 2023). However, Mati downstream and Baupa downstream had higher manganese concentrations, many of which exceeded the criteria of 0.5 mg/L. The manganese concentration recorded at Mati Creek was attributed to the mobilization of metals from the mineralized zones from the underground mine. Mati Creek is the recipient of the mine water discharge from the underground mine at the 800 RL portal location (K92ML, 2020). Dissolved metal concentrations for underground mine water were elevated at the 800RL portal monitoring point compared to the freshwater criteria in Schedule 1 of the PNG water quality guidelines. However, concentrations become diluted at downstream monitoring points prior to reaching the main Baupa River system where metal concentrations are below detection limits.

Water quality monitoring is also conducted for potable water and for the Tailings Storage Facility (TSF). Water quality for potable water is sampled from the site's two bores at the camp and at the process plant. Potable water sampled and tested at the process plant and camp sites in 2023 had no presence of the bacteria Escherichia coli (E. coli) (K92ML, 2023). According to the existing Environment Permit EP-L3(34) the maximum permissible level for E. coli (per 100 ml sample) is none present and for total coliforms (per 100 ml sample) is less than three if E. coli is absent.

In the 2023 environmental monitoring, TSF water quality (including both downstream treatment anaerobic and aerobic ponds) indicated that dissolved metal concentrations of copper, zinc, iron and mercury were lower than the PNG water quality criteria stipulated in EP-L3(34). The dissolved metal concentrations of manganese and lead were detected at, or near, their limit of detection. Metals were below limit of detection downstream of Kumian Creek. The pH of TSF water remained neutral and within 6.5 to 9 mg/L (EP criteria) throughout the year (K92ML, 2023).

20.3.3 Terrestrial and Aquatic Flora and Fauna

The vegetation surrounding the mine area is mainly of irregular hill forest. The forests in this area have irregular canopy and secondary species are common. The area has low intensity shifting agriculture surrounded by tall secondary forest. The access road crosses grassland and low altitude forest on plains and fans and enters lower montane forest (Turia, et al., 2019).

Several terrestrial flora and fauna species in the greater region (i.e. within the bordering Ramu floodplains) are IUCN red listed threatened species, some of which are endemic to the surrounding area (IUCN, 2021).

Baseline surveys completed in 2001 confirmed twenty-eight mammal species to be within the existing project area, with a further 37 likely to occur in the area. Fauna known to be present in the area include: echidnas, bandicoots (three species), tree kangaroos (three species), cuscus (four species), and nine other types of marsupials and several native rodent species. Some 110 species of birds were confirmed as present at or near the mine site. Bird species include pigeons, raptors, birds-of-paradise, parrots, megapodes as well as waterbirds of the floodplain. While there is limited information on frogs and lizards in the area, six species of snakes are known to occur within the project area (NSR Environmental Consultants Pty Ltd, 2001).

The upland tributaries of the Ramu River are known to support 23 fish species, 12 of which were collected from Baupa Creek. Two prawn (Macrobrachium) species, one shrimp (Caridina) species and one crab (Geelvinkia) species are also known to occur in the upland tributaries (NSR Environmental Consultants Pty Ltd, 2001).

20.3.4 Groundwater

The local topography and geology mean the hydrogeological conditions are complex, as described below. Local groundwater systems, localized recharge areas and inter-connectivity with regional aquifer systems determine recharge of groundwater in the Project area (Woodward, Tear, Desoe, & Park, 2020).

Groundwater Flows and Levels

Associated structures are yet to be fully mapped to estimate the inflow potential into the underground infrastructure. Due to surface outcropping of the ore body and potential fracturing associated with this unit, this can act as a direct recharge area. Creek beds intersect the outcropping ore body on the surface, and these zones could be considered as elevated influx zones. Seasonal inflow along these preferential pathways have been noted by site staff and chemical tracing should be completed to assist in verifying the source of inflow water. Inter-connectivity between regional aquifers has not been characterized to date.

Transmissivity and Storage

The ore zone represents a complex hydrogeological environment due to the presence of vuggy zones (higher permeability zones of irregular cavities inside rocks formed by dissolution) and other structural units that could rapidly transmit water. In addition, the presence of fines and weak claylike material was observed that would hinder flow along the periphery of the structure. These clay units and vuggy zones are variable in transmissivity capacity which is likely due to the deposition environment of the ore body. The presence of fracture zones and faults can also result in an asymmetric groundwater setting with preferential flow paths (Woodward, Tear, Desoe, & Park, 2020). Host rock adjacent to the ore body has a reduced permeability and could represent a confining unit. In sections, drilling through this host rock resulted in the production of water which indicates the presence of water bearing structures. However, due to the complex geological setting of the ore body, localized no-flow zones cannot be excluded (Woodward, Tear, Desoe, & Park, 2020).

20.3.5 Air and Noise

Mine management objectives are to maintain dust, air emissions and noise to below nuisance levels, with regular application of water to the road, particularly during the dry season.

Most of the air and noise emissions are mostly restricted to mobile plant in the mine area and the fixed sources in the ore processing plant. The process plant will produce noise levels that are not expected to pose a significant issue as it will be located a long distance from established villages and is not expected to produce any noise and air quality nuisance. The process plant operates for 24 hours a day operation, 365 days a year. Mining operates 24 hours a day, 365 days a year.

Regarding the expansion project, construction activities will produce dust and noise which may impact on nearby receptors. In the context of the operating mine, these emissions are expected to be negligible compared to dust and noise emissions from existing mining and processing activities. Water carts are used for dust suppression as required.

20.4 Assessed Environmental Impacts

The Kora Expansion Project proposes to continue underground mining using the long hole stoping method (Woodward, Tear, Desoe, & Park, 2020).

A new, standalone process plant is currently being constructed to increase throughput to 1.2M tpa. Discharge of treated wastewater from the TSF will be minimal or remain unchanged since it is proposed that excess process water will be reused in the process plant. In addition, approximately 70% of mine tailings will be co-disposal via an underground paste fill plant reducing the volume of tailings to be stored in the TSF. Thickened tails from the process plant are proposed to be pumped to the paste fill plant near the mine portal whereby produced tails paste is proposed to be used as backfill for the mine stopes.

Based on the proposed developments as part of the Kora Expansion Project the key potential environmental impacts are likely to be:

- Acid and metalliferous drainage from mine waste.
- Adverse changes to surface water quality and hydrology.
- Adverse changes to groundwater quality and flow.
- Reduction in downstream ecological values.

20.4.1 Tailings and Waste Rock Acid Producing Potential

A preliminary geochemical assessment of rock likely to be classified as waste rock during operation has indicated a high total sulfur content (0.09% to 8.3%) (NSR Environmental Consultants Pty Ltd, 2001). Approximately half the waste rock samples tested had sufficient acid-neutralising capacity (ANC) to be classified non-acid-forming ("NAF") and are unlikely to be a source of AMD. The remaining half of the samples were classified as potentially acid forming ("PAF"), which means that they may, in the future, become a source of AMD. However, the results of net acid generation ("NAG") tests indicated that the PAF material is likely to exhibit a relatively long lag time (i.e. at least a year) before acid conditions occur.

A geochemical characterization of drill core samples from the mineralized zone indicated that the mineralized samples had little, if any acid-neutralising capacity ("ANC") and high to very high total sulfur contents (0.04 to 29.3%) (NSR Environmental Consultants Pty Ltd, 2001). The absence of significant ANC suggests that mineralized rock is PAF and is expected to generate AMD (poor quality, low pH drainage) almost as soon as any sulfide oxidation takes place. At the time of writing results for the geochemical and metallurgical properties of both the flotation and leach tailing types through test work undertaken as part of the development were not yet available. The initial metallurgical test work undertaken prior to development of the Kainantu Gold Mine had shown that the flotation tailing contained around 0.1% sulfur, which has been confirmed to be non-acid forming during operations. An enhanced geochemical sampling program is recommended to better understand potential, long-term risks associated with AMD.

20.4.2 Water Quality and Hydrology

The expansion project will increase gold production, and this will result in higher volumes of processing waste. However, water discharge volumes from the TSF into the receiving environment (i.e. Kumian Creek) will be minimal or remain unchanged given excess process water will be reused. A small portion of tailings water is discharged during the wet season following treatment through aerobic and anaerobic treatment ponds. The utilization of the paste plant will also reduce the amount of tailings discharged into the TSF, with 70% of tailings re-purposed as paste for underground backfill.

Uncontrolled runoff from disturbed areas associated with the TSF expansion may contribute to increased turbidity, heavy metals and sediments to downstream rivers. In addition, unplanned spills and leaks of hazardous chemicals could also result in reduced water quality in the receiving aquatic environment. Minor surface water contamination may be expected during surface clearing works for expanding the process plant which shall be managed by the main contractor and project site representative.

Groundwater discharge volumes from the mine portal are expected to increase due to water inflows as the underground mine workings expand and this may adversely impact the quality and flow of receiving surface water. Water balance modelling is being developed at the time of writing that will provide further information on the groundwater inflows and outflows. Flow meters are being installed to enhance measurement of the volume of mine water discharge.

The aquatic ecology of streams within the project area may be impacted by changes to the stream dynamics due to water use and water quality changes during the mining operation and beyond. Any possible changes in water levels, temperature, turbidity, and instream water quality can impact the aquatic habitats and life forms in the streams. Changes can be brought about with surface and ground water use and disturbance, as well as inflow of discharge and stormwater holding increased levels of sediments, effluence, or toxic components during an incident or at normal operation (NSR Environmental Consultants Pty Ltd, 2001; IFC, 2007).

Post-closure impacts on surface water due to exposed landforms erosion is proposed in the conceptual mine closure plan to be mitigated via revegetating exposed sites to minimize run-off. The conceptual mine closure plan (K92ML, 2022) states that the TSF will remain in a structurally sound and maintenance-free state with a spillway designed and constructed to meet 1:100-year 2-hour average recurrence interval event. General site inspections, surface water monitoring and rehabilitation monitoring is proposed to continue post-closure.

20.4.3 Greenhouse Gas Emissions

Scope 1 and Scope 2 greenhouse gas (GHG) emissions associated with the Kora Expansion Project are expected to increase on an absolute basis. However, GHG emissions are expected to decline significantly based on an emissions per tonne milled and an emissions per gold-equivalent ounces produced basis.

In June 2023, K92ML set an emissions reduction target of a 25% reduction in GHG emissions by 2030 against a business-as-usual forecast, which is defined as a reduction in GHG emissions against an established baseline forecast assuming no intervention measures have been taken to reduce carbon emissions. A key component of the Company's strategy to achieve this target is further enhancing the mine's access to grid electricity, and in doing so, reducing its reliance on diesel-powered electricity generation. Hydroelectricity comprises a significant amount of the local grid electricity, which is a cleaner alternative to diesel-powered generators.

20.5 Social and Community

The following subsections present key information related to social and community aspects related to the Kainantu Gold Mine. The results of key socioeconomic studies are summarized, including ongoing socioeconomic data collection as well as key social and community programs implemented by the Company.

20.5.1 Socioeconomic Baseline Studies

The Government of PNG has designated four "impacted communities" in the project area, namely the villages of Bilimoia (comprised of three sub-villages), Unantu, Pomasi, and Sakimaniap (Watarais/Marawasa). As per previous socioeconomic studies conducted by the Company (K92ML, 2018), the populations of Bilimoia, Unantu, Pomasi are classified as "Uplanders" in the Eastern Highlands Province, whereas the Sakimaniap population is classified as "Lowlanders" in the Morobe and Madang Provinces. There are numerous other informal settlements (i.e. hamlets) within the vicinity of the mine that have been established as mining operations have progressed.

The K92ML External Affairs and Sustainable Development department conducts ongoing social baseline and mapping surveys to collect demographic and socioeconomic indicators related to local communities. The surveys also act as a form of a census. Local youth from impacted communities are employed to collect the data, which supports ongoing stakeholder engagement initiatives, including to plan and roll out community investment projects.

Figure 20-3 below provides an overview of local impacted communities and their proximity to the operations and key project infrastructure. Table 20-1 lists the communities with key population data, which are based on survey and social mapping exercises collected by K92ML in 2023.

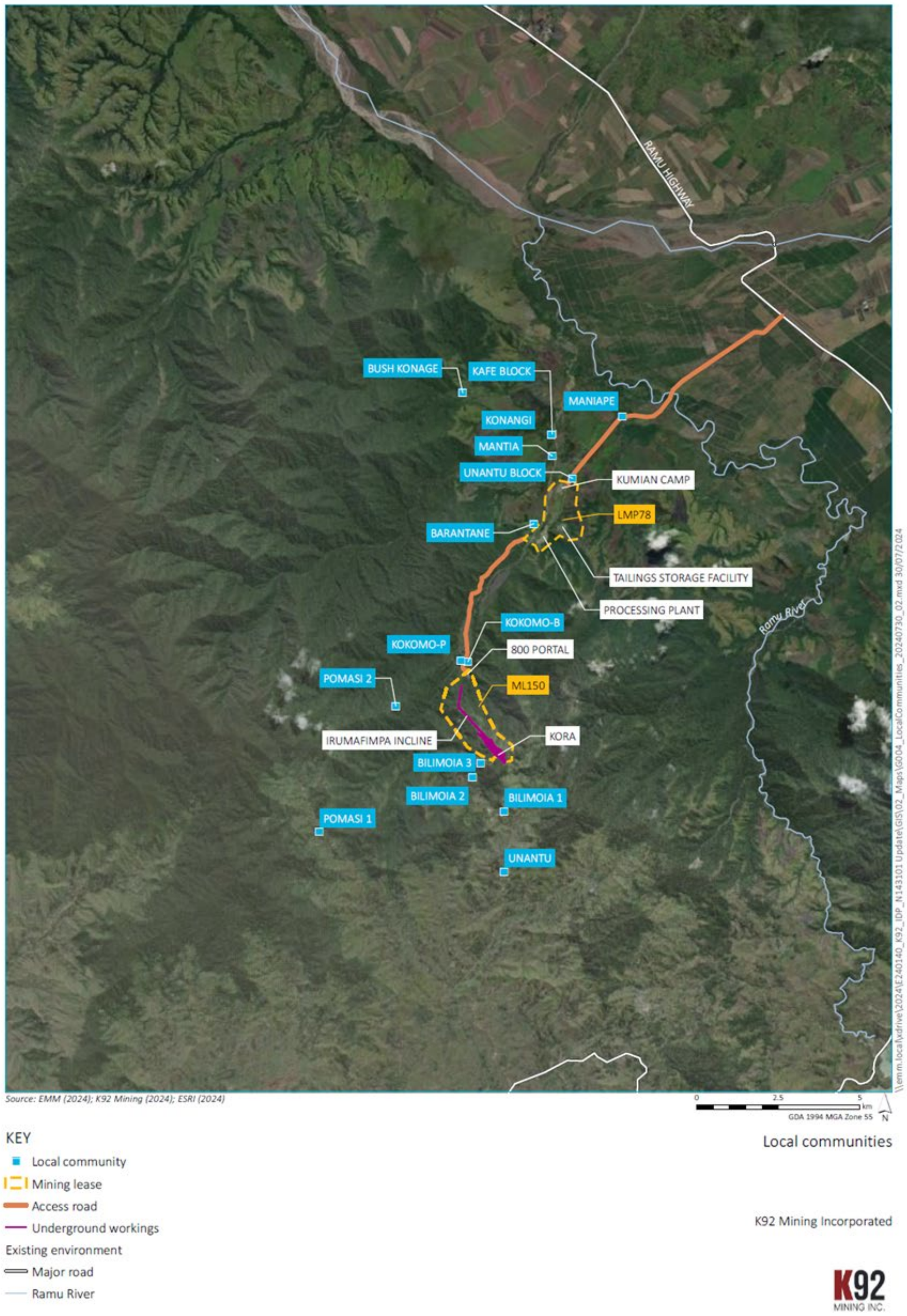


Figure 20-3 Local Communities

Table 20-1 Local Impacted Communities

Community	District & Province	Population
Bilimoia	Kainantu District, Eastern Highlands Province	2,842
Unantu	Kainantu District, Eastern Highlands Province	2,037
Pomasi	Kainantu District, Eastern Highlands Province	1,516
Sakimaniap (Watarais/Marawasa)	Markham District, Morobe Province	2,323

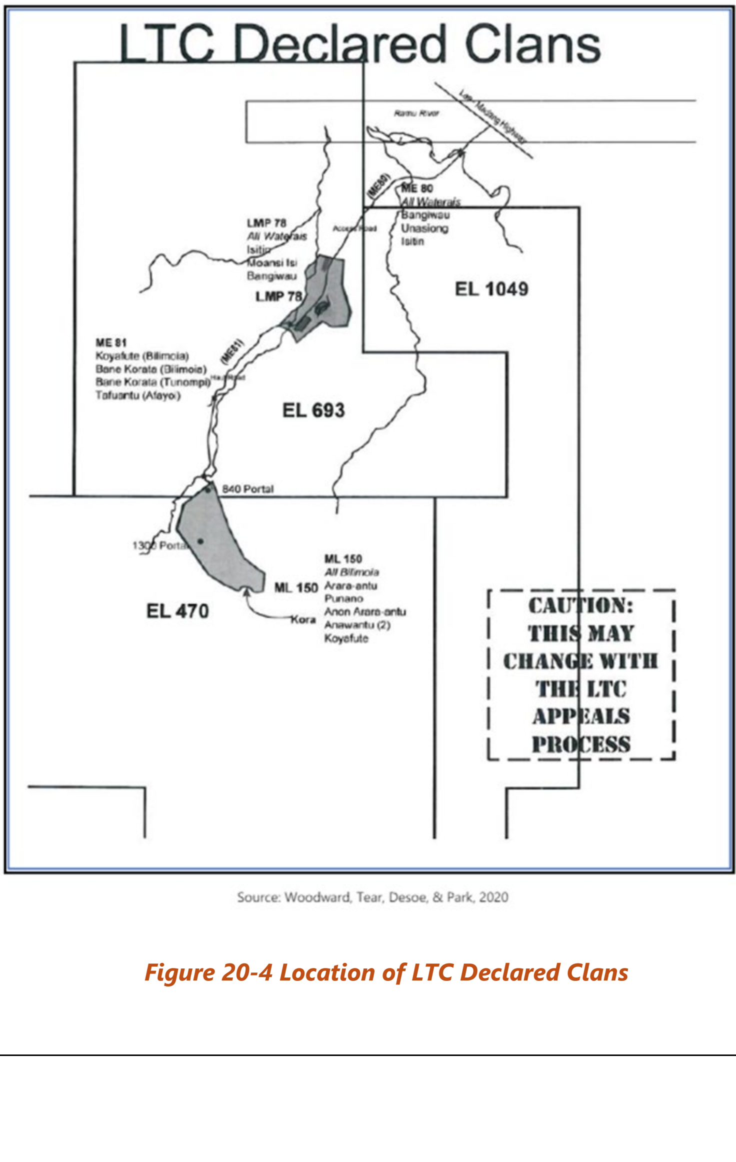
20.5.2 Memorandum of Agreement

The original tenement holder, Highlands Pacific Limited ("HPL") signed a Memorandum of Agreement ("MOA") with the State, the Eastern Highlands Province ("EHP") Government, the Kainantu LLG, the Bilimoia Landowners Association ("BLA"), and Associated Landowners on 11 November 2003. This MOA provides a framework for the relationship between the Company and local stakeholders as well as for the allocation and use of the royalties derived from the project for the benefit of the provincial and local governments and the landowners.

The original MOA was reviewed over 13-17 July 2020. The most significant documents to emerge from the 2020 Review are K92ML's Supply and Procurement Plan ("SPP") and Business Development Plan ("BDP") produced by K92ML's External Affairs and Sustainable Development department. The SPP indicates the type and range of possible contracts and joint ventures for properly qualified and able landowner entities to take part in, while the BDP is a business training and instruction curriculum and schedule for landowners.

Part of the 2020 MOA renegotiation was an offer of 5% equity in the mine, including to the Provincial Government (1.5%), the Bilimoia landowners (3%), and the Associated Landowners (i.e. Sakimaniap, Unantu and Pomasi peoples) (0.5%). This portioning is different from the 2003 equity offer which was this same 5% offered only to the Bilimoians. Shareholding would be paid for from dividends on the basis of the value of K92ML's sunk costs. Equity issue is, however, dependent on the final signing of the MOA by all parties and acceptance of the terms by those to whom the offer has been made.

Follow-up stakeholder consultations were sponsored and facilitated by the MRA in 2023; however, the final signing of the MOA still awaits approval from the National Executive Council of the PNG Government pending the results of the PNG Land Titles Commission appeals process (see Figure 20-4). Importantly, the original MOA will remain in place until a new agreement is signed. As such, the absence of a revised, signed MOA will not preclude the Kora Expansion Project from proceeding, which is a common occurrence for mining and other resource projects in PNG.



Source: Woodward, Tear, Desoe, & Park, 2020

Figure 20-4 Location of LTC Declared Clans

20.5.3 Stakeholder Engagement

The Company implements ongoing community consultation and engagement initiatives tailored to the local operating environment in PNG, given the unique sociocultural context within the country. The Company maintains a strong presence in local communities, including through Village Liaison Officers ("VLOs"), who are present in local communities on a daily basis to enable frequent, two-way communication with local stakeholders. The Company also maintains a regional external affairs office in Kainantu, the district capital of the Kainantu District, Eastern Highlands Province.

The Company's community engagement efforts are led by its External Affairs and Sustainable Development team, which is composed of approximately 40 community relations practitioners, including nine VLOs as well as six women in specialist roles focused on female empowerment initiatives.

The Company maintains a grievance mechanism to obtain feedback and resolve complaints from local communities. Given the modest, if not minimal, literacy level in the region, people with grievances are not expected to write them out and formally submit them for the convenience of the Company. Local community members do raise grievances as a part of regular engagements, and grievances are raised in a range of forms, but they are usually verbal, with the occasional letter. Verbal grievances are dealt with on-the-spot when possible and, where necessary, management is engaged in real time to resolve them. These and other grievances that are not instantly solvable are covered in daily written reports to unit management by field officers and are recorded to the extent practicable in a daily grievance record.

Social and Community Programs

To support its social license to operate, K92ML has a range of community and social development initiatives in place relating to the supply of services, infrastructure and maintenance for local communities, water supply, health programs, education assistance, livelihood programs and business development, which are summarized below.

Local Procurement

The Company has created multiple business opportunities for the local communities in the vicinity of the mine to benefit from its operation. This currently includes four major joint venture contracts between local landowner associations and PNG companies for the provision of services, as well as numerous smaller contracts with local businesses. The major contracts include catering and camp management, security, road transportation, and ancillary mobile services. In 2022 and 2023, these contracts generated USD $21.3M and USD $24.5M in revenue, respectively, for the local landowner associations.

Infrastructure Tax Credit Scheme

As at end-2023, the Company is registered as a participant in the Infrastructure Tax Credit Scheme ("ITCS") of the PNG Government, through which up to two percent of the Company's assessable income can now be allocated by the Company for spending on approved community projects, including local infrastructure, health programs, and educational initiatives, and deducted from future corporate tax payable. This is in addition to the Company's various community and social programs. The first project for implementation was formally approved by the PNG Department of National Planning in December 2023 for local road upgrades. Construction works will commence in 2024. Additional projects are being scoped in partnership with local communities and government stakeholders for future implementation according to local development needs.

Sustainable Agriculture Livelihoods Program

Launched in 2020, K92's Sustainable Agriculture Livelihoods Program began as on-the-job training with K92ML setting up model farms in and near local communities. By 2023, the program has grown to include nearly 180 farmers, nearly 80% of whom are female. This program is supported by multiple K92ML community officers who provide training, mentorship, and logistics support.

Local farmers participating in the program now supply a variety of local crops to K92ML exploration camps as well as the mess hall at the mine camp, which is operated through a local JV. K92ML has also supported the farmers in accessing regional markets through retailers and wholesalers in Lae and Port Moresby.

Adult Literacy Program

Since 2019, the K92's External Affairs and Sustainable Development team has implemented a unique Adult Literacy Program in partnership with local communities. The program offers three levels of English and Tok Pisin, a predominant local language in PNG, for those who cannot read or write. An average of approximately 150 people have graduated from the program annually since its inception in 2019, 90% of whom are female.

Women in Mining Program

In December 2023, the Company was recognized by the PNG Chamber of Resources and Energy ("CORE") with an award for Outstanding Community Humanitarian Initiative. The award recognizes the Company's contributions and achievements related to its Women in Mining program, which focuses on female-targeted community investment programs in local communities, including training initiatives, preventative health programs, and support for small enterprises.

Educational Initiatives

The Company supports education and skills development of the mining industry in PNG through a variety of programs and initiatives. The Company has established the K92 Mining Tertiary Scholarship Program, which includes six tertiary scholarships for 2023 in the fields of mineral processing, mine engineering, geology, and agriculture. The Company has also partnered with a local company to award two scholarships for Women in Mining to students studying logistics, commercial, or business management. A total of 43 scholarships were awarded in 2023.

In addition, the Company has signed multiple memoranda of understanding with universities in PNG to support areas of mutual benefit. These partnerships include financial support for the university, work experience for students and undergraduates, technical cooperation, and project generation.

A variety of strategic training initiatives are implemented by the Company, including its Industrial Trainee Program, Apprenticeship Program, Graduate Development Program, and Pre-Vocational Program (among others) to help support a robust pipeline of local available talent.

The Company continues to implement a program to assist employees with school enrolment fees. The Company pays 50% of school fees for primary and secondary schools for children of employees on condition that the employee contributes the remaining 50%.

Share Participation Plan

In 2023, K92ML introduced the K92 Share Participation Plan (SPP) as a long-term incentive plan for PNG national employees occupying roles up to the coordinator level. The plan provides for the employees to share in the success of the Company and to reward the achievement of long-term strategic goals and improvement in the Company's performance. Unlike traditional long-term incentive programs, under the K92 SPP, shares are issued to a trust which holds them on behalf of the employees on an annual basis and a percentage are sold each year, and the proceeds from the sale of the shares are distributed to the employees.

20.5.4 Security

K92ML uses the services of a contractor security company with a staffing level of 51 Guards who patrol various locations across the ML150 to ensure the Company employees and assets are secured.

The Asset Protection Department ("APD") of K92ML consists of a Guarding division (contractor company), Dog Unit and a Transport division to help ensure all employees and contractors are safely transported daily to work and back on site. The APD deploys police and security officers to all exploration camps run by the Company.

An on-site police post is manned by public police officers from the region and is under the command of the Eastern Highlands Provincial Police Commander ("PPC"). K92ML has donated a 40-man police barracks and officer accommodation blocks to support the force. The barracks are utilized for police accommodation as required. Approximately 25 officers are attached to the operations to ensure the safety and security of its employees, contractors, and assets, as well as to provide community policing.

20.5.5 Assessed Community Impacts

There will be community impacts associated with the Kora Expansion Project. With the proposed expansion, this will substantially increase the mines workforce and employment for the local communities. There will be continued enhancement of the communities through provision of livelihood trainings, educational assistance for children, improved access to clean and safe drinking water, community investment initiatives, and creation of business opportunities. In using underground mining methods, the surface disruption to landowners is minimized; however, an extended or new mining license to the south of ML150 to cover the extension of the Kora deposit into EL470 will require the necessary permitting, according to PNG Mining Act, prior to mining. Part of the process involves a compensation agreement with the landowner group concerned.

Landowner

While Landowner identification and social mapping would be required for a new or extended ML to access the portion of the resource outside the existing lease, it is anticipated that there would be no change to the recognized Landowners. It is noted that the outstanding LTC determination of landholdings within the existing project will also apply to the extension of the ML. Currently, compensation payments, including distribution of a portion of royalties, are being accrued pending the determination from the TLC.

Construction Impacts

Construction activities will produce dust and noise which may impact on nearby receptors. In the context of the operating mine, these emissions are expected to be negligible compared to dust and noise emissions from existing mining and processing activities. Water carts will be used for dust suppression during construction if required.

20.6 Management Plans

K92 has established an Environmental Management System ("EMS"). Key elements of the EMS include:

- Defined resources, roles, and responsibilities for environmental management.
- A risk assessment process to identify environmental hazards.
- Objectives and targets.
- Training and awareness initiatives.
- Standard operating procedures, including for emergency preparedness and response.
- Audit and compliance programs.
- A clear process for management of non-conformance, and corrective/preventative actions.

Additionally, K92ML's environmental department implements a site-wide, government-approved Environmental Management and Monitoring Plan ("EMMP") covering key environmental risks and impacts, including sub-plans for water management, land management (including mine closure), biodiversity, air quality, hazardous materials and waste, and environmental compliance (among others).

The EMMP is subject to renewal in 2024 following the receipt of the revised environmental permit for the Kora Expansion Project in late-2023. As of the time of writing this report, the mine's Environmental team was preparing the revised EMMP for expected submission to CEPA at the end of 2024 ahead of key expansion commissioning in 2025. As stipulated by CEPA, the revised EMMP should merge all existing EMMPs and should also capture all management and mitigation measures in relation to the transport of gold and copper concentrate from the mine site to the Lae wharf. In addition, CEPA has stipulated that the revised EMMP should include (but not be limited to) outlining evacuation methods, alarm systems and other measures put in place to avoid catastrophic effects in the event of a tailings dam failure.

20.6.1 Environmental Performance Monitoring

Environmental monitoring is conducted daily on-site by the site environmental team, and reporting is produced on a weekly and monthly basis for internal management review, and on an annual basis for external, regulatory and compliance review. Annual reports are provided to CEPA include:

- Records and results of environmental inspection and monitoring programs.
- Summary of results including any exceeding of environmental requirements.
- Remedial measures implemented for continuous improvement.
- Status of environmental compliance with permit conditions and other relevant information.

Internal environmental auditing of the operation is conducted by HSE Department on an annual basis. In addition, periodic external audits of the mine are carried out by CEPA or other independent environmental contractors/consultants to verify whether operations follow the EMP, and the EMS is being implemented.

An audit program is maintained by the site Environmental Coordinator to:

- Maintain a forward schedule of planned audits.
- Conduct the audits or plan for the audits to be carried out.
- Maintain records of all audit findings.
- Communicate findings of the audit to relevant departments or interested parties.
- Ensures that corrective actions are implemented.

20.7 Mine Closure

The Kainantu Gold Mine has a conceptual mine closure plan, and it will need to be updated to include closure and rehabilitation aspects that will arise from the proposed expansion project. Associated costs for the updated plan are expected to be $0.15M.

Based on the current conceptual mine closure plan, K92ML has outlined the formation and responsibilities of the mine closure management team. Pre- and post- closure activities described in the mine closure plan include:

- Engineered rehabilitation of sections of the mine site, including campsite, portals, exploration yard and tailing storage facility.
- Demolishment, dismantling and removal of mine related infrastructure.
- Cleaning and disposal of contaminated soils.
- Public vehicular access past the mine shall be maintained.
- Community exit strategy which includes transition plans to maintain sustainability of impacted communities post-mine closure.
- The updated conceptual mine closure plan will need to meet the requirements of the new 'Mining Project Rehabilitation and Closure Guidelines - Papua New Guinea'.

The key aspects that require inclusion in the updated mine closure plan that relate to the expansion project are:

- Additional infrastructure, including the processing plant, paste plant, and ancillary infrastructure.
- Maintaining TSF integrity.

20.8 Key Recommendations

- Geochemical characterisation of the ore body and waste rock should be reviewed, and an ore/waste acid mine drainage potential block model developed. With an extensive exploration program underway, drill core sampling is a cost-effective way to sample, and feed into a predictive ore/waste model. Potential acid generating wastes can be managed at the underground source rather than retrospectively on surface.
- Timely resolution of the 2020 MOA renegotiation with the Provincial Government and Landowners, with signing of the MOA still awaits approval from the National Executive Council of the PNG Government pending the results of the PNG Land Titles Commission appeals process. This will result in flow of royalties (held in escrow) to the landowners and provide the basis for future negotiations as the mine develops.
- The EMS and statutory EMMP should continue to be updated to include the project expansion new aspects and impacts. Environmental program specific management plans and SOPs should include new permit requirements.
- The mine closure plan will need to be reviewed and updated to reflect material changes associated with the expansion.

21. CAPITAL AND OPERATING COSTS

21.1 Mine Costing Basis of Estimate

Entech was engaged by K92ML to assist with the mining cost estimation for the DFS mine plan. The mine costing estimations were built up using a fixed and variable cost format. The mining cost estimations assume K92ML will be executing all mining activities to meet the DFS LOM schedule, except for vertical development and surface haulage which will be completed by mining contractors. Based on current site productivities, K92ML utilized Entech's mine plan physicals to estimate the fleet and manning requirements. Entech reviewed all K92ML equipment and manning estimates and found them appropriate for the mine plan.

Fleet and manning costs comprise the fixed cost component of the estimate, with all mining and maintenance consumables captured in the variable component. The variable cost component was built up from current site actual costs, with manning and equipment costs built up from current site actual salaries and equipment purchase cost quotations. Cost estimates for other major infrastructure such as primary ventilation fans, paste plant and dewatering pumps were sourced from manufacturer quotations.

The K92ML and Entech cost estimate includes allowances for the following items:

- Personnel and equipment mobilization.
- Tooling for the workshop.
- Manpower (including operations, administrative and supervisory personnel).
- All mobile mining machinery.
- All consumables required to complete the mining activities.
- Raise drilling of vertical development.
- Low voltage (LV) electrical cable, distribution boards, starter boxes and miscellaneous electrical infrastructure.
- Secondary pumping system (to enable dewatering from the working headings to the primary pumping system).
- Secondary ventilation system.
- Supply and installation of HV cabling and step-down transformers.
- Personal protective equipment (PPE).
- Supply and installation of escapeway network.
- Supply and installation of primary pumping system.
- Supply and installation of primary ventilation system.
- Supply and installation of pastefill system.
- Supply and installation of the high voltage (HV) electrical power system (K92ML have captured the cost of supply of power to the surface portal point separate to the mining section of this study).
- Refuelling facilities and supply of diesel fuel for mobile equipment.
- Management and technical staff.
- All flights, accommodation and messing.
- General owner costs including management relevant to mining (non-mining management captured in other sections of the study).

21.2 Mining Capital Cost

This section provides a summary of the underground mining capital costs.

21.2.1 Capital Decline / Incline Development

The decline/incline development costs include allowances for the following:

- Costs associated with jumbo development.
- Costs associated with ground support.
- Costs associated with blasting.
- Costs associated with the loading and hauling of the decline/incline waste material.

- An allocation of diesel fuel and power based on these activities.

21.2.2 Capital Lateral Development

Lateral capital development includes the following types of development drives:

- Stockpiles and sumps.
- Level access drives.
- Footwall access development.
- Ventilation drives.
- Escapeway accesses.
- Electrical cuddies.
- Ore / waste pass accesses.
- Pastefill cuddies.

The associated costs include:

- Costs associated with jumbo development.
- Costs associated with ground support.
- Costs associated with blasting.
- Costs associated with the loading and hauling of waste material.
- An allocation of diesel fuel and power based on these activities.

21.2.3 Capital Vertical Development

The vertical capital development includes drilling of 0.25 m - 5.0 m diameter raise-drilled ventilation rises, escapeways, ore/waste passes, service holes and fresh air rises (inclusive of setup, mobilization / demobilization, etc).

21.2.4 Mine Services

Costs included in mine services include the capital proportion of:

- The diesel cost.
- The underground power generation cost.

21.2.5 Mine Overheads

Mine overheads related to the underground mine include the capital proportion of:

- All underground management and technical staff salaries including on-costs.
- Allocation of messing, accommodation, and FIFO costs.
- General consumables.
- Personal protective equipment (PPE).
- Safety and training.

21.2.6 Infrastructure Capital

Infrastructure capital includes supply and installation of infrastructure items to support the underground operation. Table 21-1 shows the breakdown of the infrastructure capital. Paste plant capital costs are captured separately in Section 21.6.

Table 21-1 Infrastructure Capital

Description	Unit	Value
Safety		
Refuge Chambers (10x20-man)	USD (M)	$1.02
Refuge Chambers (4x4-man)	USD (M)	$0.35
Ventilation		
Primary Fans (Zitron)	USD (M)	$4.5
Primary Boster fan	USD (M)	$0.2
Electrical		
WT114/WTX3/90 kW Flygt pumps	USD (M)	$1.5
55 kW fans	USD (M)	$1.02
11 kV Ventilation Reticulation	USD (M)	$2.1
11 kV Paste Reticulation	USD (M)	$1.2
Sub-Station Purchases	USD (M)	$7.0
Infrastructure		
Escapeway Ladders	USD (M)	$1.22
Ventilation Walls	USD (M)	$0.32
Ore Pass Grizzly	USD (M)	$6.75
Total	**USD (M)**	**$27.18**

21.2.7 Summary of Capital Costs

The estimated capital costs are summarized in Table 21-2.

Table 21-2 Mining Capital Cost Totals

Description	Unit	Value
Infrastructure and Sustaining	USD (M)	$94.5
Decline	USD (M)	$13.1
Cap Access	USD (M)	$4.8
Ventilation	USD (M)	$13.3
Escapeway	USD (M)	$1.1
Other Lateral Development	USD (M)	$39.6
Fleet	USD (M)	$26.6
Operators	USD (M)	$56.0
Capital Mine Services	USD (M)	$20.1
Capital Mine Overheads	USD (M)	$15.6
Total Capital	**USD (M)**	**$284.6**

A breakdown of the capital unit costs is shown in Table 21-3.

Table 21-3 Mining Capital Unit Costs

Description	Unit	Value
Infrastructure and Sustaining	USD / t ore	$15.30
Decline	USD / t ore	$2.12
Cap Access	USD / t ore	$0.77
Ventilation	USD / t ore	$2.15
Escapeway	USD / t ore	$0.18
Other Lateral Development	USD / t ore	$6.41
Fleet	USD / t ore	$4.31
Operator	USD / t ore	$9.07
Capital Mine Services	USD / t ore	$3.25
Capital Mine Overheads	USD / t ore	$2.52
Total Capital Cost	**USD / t ore**	**$46.09**

21.2.8 Capital Costs for TSF Expansion

The total estimated capital cost required to expand the existing Tailings Storage Facility (TSF) to accommodate the demands of the Definitive Feasibility Study (DFS) scenario for the Kora Expansion project is USD $7.9 million. This estimate includes a contingency allowance of ±15% to account for potential variations in embankment construction costs.

21.3 Mining Operating Cost

This section provides a summary of the underground mining operating costs. Operating costs for other facilities such as the process / paste plant and tailings facility are captured separately in Section 21.5 - 21.9.

21.3.1 Operating Lateral Development

Lateral operating development includes the following types of development drives:

- All ore drives.
- Waste development off the footwall access drives.
- Level stockpiles.

The associated costs include:

- Costs associated with jumbo development.
- Costs associated with ground support.
- Costs associated with blasting.
- Costs associated with the loading and hauling of ore and waste material.
- An allocation of diesel fuel based on these activities.

21.3.2 Operating Vertical Development

The operating vertical development includes development of raise-bored slot raises for stoping.

21.3.3 Stoping

Costs included in ore stoping include variable costs associated with stope drilling, blasting, backfilling, and loading and haulage.

21.3.4 Mine Services

Costs included in mine services include the operating proportion of:

- The diesel cost.
- The underground power generation cost.

21.3.5 Mine Overheads

Mine overheads related to the underground mine include the operating proportion of:

- All underground management and technical staff salaries including on-costs.
- Allocation of messing, accommodation, and FIFO costs.
- General consumables.
- Personal protective equipment (PPE).
- Safety and training.

21.3.6 Summary of Operating Costs

The estimated operating costs are summarized in Table 21-4.

Table 21-4 Mining Operating Cost Totals

Description	Unit	Value
Operating Access	USD (M)	$12.0
Ore Drive	USD (M)	$41.3
Stope	USD (M)	$108.1
Operators	USD (M)	$126.0
Operating Mine Services	USD (M)	$49.4
Operating Mine Overheads	USD (M)	$38.5
Surface Haulage	USD (M)	$33.1
Grade Control	USD (M)	$11.8
Total Operating	**USD (M)**	**$420.2**

A breakdown of the operating unit costs is shown in Table 21-5.

Table 21-5 Mining Operating Unit Costs

Description	Unit	Value
Operating Access	USD / t ore	$1.95
Ore Drive	USD / t ore	$6.68
Stope	USD / t ore	$17.51
Operators	USD / t ore	$20.40
Operating Mine Services	USD / t ore	$7.99
Operating Mine Overheads	USD / t ore	$6.24
Surface Haulage	USD / t ore	$5.37
Grade Control	USD / t ore	$1.91
Total Operating	**USD / t ore**	**$68.05**

21.3.7 Operating Costs for TSF Expansion

The estimated annual operating cost for managing the tailings facility is set at an average of USD $0.6M. This estimate does not take into account the rate of discount and reflects the ongoing cost of operations. Significant components of these operating expenses include fees for consultant activities, which are crucial for ensuring compliance and optimizing tailings management strategies.

21.4 Process Plant and Infrastructure Capital Costs

21.4.1 Capital Cost Estimate Work Breakdown Structure

The capital cost estimate for the new process plant and infrastructure has been developed by the following contributors:

- GR Engineering Services Limited: process plant, paste plant and compilation of overall capital estimate.
- AVID Group Ltd: Power Station.
- Papua New Guinea Power Limited; Overhead Power Line (OHPL).
- CHEC (China): River Crossings.
- K92ML: Electrical infrastructure and Owner's costs.

The capital cost estimate is based on the Project scope as described in this report and has been peer reviewed for acceptance by the project team. All costs are presented in United States Dollars (USD) unless otherwise stated and based predominantly on fixed contract pricing and 2Q23 pricing. The estimate is considered to have an accuracy of +15% / -5%.

21.4.2 Capital Estimate Summary

The capital cost estimate is summarized in Table 21-6. The estimate is presented for the initial capital cost based on a 1.2M tpa throughput.

Table 21-6 Capital Estimate Summary (USD, 1Q24, +15/-5%)

Main Area	USD (M)
Process Plant	$90.3
Paste Plant	$42.5
River Crossing	$14.4
Power Station	$10.3
Electrical Infrastructure	$6.9
OHPL	$1.9
Maintenance Facilities	$3.0
Warehouse	$0.9
Owner's Project Costs	$20.1
Camp Upgrade	$4.2
Total	**$194.4**

The capital cost for the new process plant was prepared in accordance with GRES standard estimating procedures and practices based on an engineering, procurement and construction (EPC) contract or turnkey design and construction approach to the provision of the project management, engineering, drafting, procurement, construction and commissioning services required to construct and commission the processing plant and associated infrastructure. Within the EPC contract, the capital cost for the process plant and associated services and infrastructure within the scope of work is a fixed lump sum price. A high level breakdown of the EPC contract pricing for the design, construction and commissioning of the new process plant facilities before the commencement of production is presented in Table 21-7 including the long lead equipment items free issued to the Project by K92 Mining Inc.

Table 21-7 Process Plant Contract Price Summary (USD, 1Q24, +15/-5%)

Process Plant	USD (M)
Total Direct costs	$43.7
Total Indirect costs	$36.5
Subtotal	**$80.2**
Free Issued Equipment	$10.1
Total	**$90.3**

21.4.3 Estimate Currency and Base Date

The estimate is expressed in USD based on prices and market conditions as of first quarter 2024 (Q1 2024).

Foreign currency exposure is shown in Table 21-8 below.

Table 21-8 Foreign Currency Exposure

Currency	Exchange Rates
USD	1.00
USD to AUD	1.42
PGK to USD	0.24

21.4.4 Estimate Basis

The capital cost estimate presented in this study relate to capital works required to design & construct a new processing plant and support infrastructure facilities.

Design criteria and flow sheets for the process plant and infrastructure were developed using previous studies, historical data, metallurgical test work data, and in-house experience.

Plant equipment selections were made, and plant layouts were developed. Sufficient engineering design was undertaken to ensure the functionality of the proposed layouts, suitability of equipment specifications and to enable construction material quantities to be estimated to an accuracy appropriate for fixed cost contracts.

The estimate is prepared on a commodity basis and reported by area.

Current market pricing for equipment, labour and bulk rates are incorporated into the estimate. The installation rates include all charges necessary to deliver the requirements of the Project.

21.4.5 General Estimating Methodology

The Project capital cost estimate has been developed from the Engineering, Procurement and Construction contractors and vendors that were engaged by K92ML in late 2023 and early 2024.

The costs are based upon an EPC approach for the construction and commissioning of the project facilities with large vendor equipment items purchased and free issued by K92. The scope includes the processing plant and minor infrastructure from the receival of ROM ore through to saleable Concentrate and or gold doré.

The estimate includes all the necessary costs associated with process engineering, design engineering and drafting, procurement, construction and construction management, commissioning of the operational facilities and their associated infrastructure. Also included are first fills of reagents, commissioning consumables and spare parts to design, procure, construct and commission all of the facilities required to establish the Project.

The estimate is based upon fixed costs for engineering, equipment, commodities and installation are therefore able to meet the Class 3 estimate accuracy.

The costs also include contingencies that reflect the risk management by the contractors for items such as logistics and standard weather.

Earthworks

Plant and infrastructure site earthworks quantities were estimated off the plant layout. An overall bill of quantities for the Project was compiled and a costed. The basis assumes a single sub-contractor on multiple work fronts performing all the nominated earthworks for the Project.

Concrete

The concrete quantities were calculated for each area from the design drawings, layout drawings and preliminary designs developed for the Project.

These designs formed the competitive tender scope of works for a sub-contractor. The scope included the requisite concrete batching plant however excluded the supply of the raw concrete materials that were to be free issued.

The concrete subcontract included all relevant overheads and time related costs for the proposed duration.

Structural Steelwork and Platework

Quantities were calculated on an area by area basis, from the general arrangement drawings and layout drawings developed for the Project.

The design drawings and quantities formed the basis of fabrication tenders for fixed rates.

The rates utilized for structural steelwork includes heavy, medium, and light members, conveyor gantries, and trestles. The rates utilized for plate work includes small bis-alloy lined bins and chutes, rubber lined chutes, hoppers and various other. Separate rates are used for grid mesh flooring, hand railing, and stair treads.

The supply rates include materials supply, shop detailing, fabrication, surface preparation, final painting in the shop, and identification tagging.

Equipment

The process design criteria were used to develop the mechanical equipment list that defines the requirements and sizes of all the mechanical equipment, plate work and tank items. Specifications and data sheets were developed for all major equipment. Some major packages were purchased by K92ML and free issued including:

- Sag & Ball Mills.
- Primary Crusher.
- Flotation Cells.
- Thickeners.
- Concentrate Filter.

Written quotations from enquiries accompanied by engineering specifications and data sheets were requested from recognized suppliers for all the major equipment in the plant.

Piping

The piping estimate was based on the design layout and drawings, utilised for fabrication tenders and fixed rates.

Electrical and Instrumentation

The electrical, instrumentation and control quantities have been compiled from the Project scope, single line diagrams, layouts, equipment list, and load list. The instrument list was developed from P&ID’s.

Cable and material quantities were based on layout, switch room locations, equipment specific requirements and drive requirements, and schedules produced.

Tendered pricing has been used for all electrical components and commodities.

Transport and Freight

Major equipment suppliers were requested to provide pricing to transport equipment to either ex-works or site. Where suppliers have not included transport, a freight forwarding contractor was engaged to provide fixed costs.

Sea freight and packing charges have been included to transport all imported materials and equipment from place of manufacture to site. These costs are included in the supply cost of the item.

Transport for fabricated steel items has been allowed based on quantities expected to be loaded per truck and anticipated truck price between selected suppliers and site. Where possible, fabricated steel components will be suitably packed onto skids and containerized to minimize handling costs.

Taxes and duties associated with entering PNG are not included in the plant capital costs.

Installation and Labour

Installation was based on detailed scope of works tendered to experienced subcontractors. There were two main subcontracts, Structural Mechanical & Piping (SMP) and Electrical & Instrumentation (E&I).

These tenders provided fixed rates for the installation labour, site cranage, site offices and any overheads. The contracts also included consumables and some commodity supply.

The scope included transport of personnel however the accommodation and meals were free issued by K92.

The tender scopes included consideration of experienced supervision and labour as well as the costs of onboarding personnel to an operating site.

Labour crew rates were built-up including an appropriate mix of supervision, skilled and unskilled personnel. Each crew rate included the costs of mandatory meetings and breaks, small tools, statutory labour costs, PPE, and clothing.

Plant Services

The capital cost estimates compiled for the plant services / infrastructure components of the project are based on requirements dictated by the current process plant design / capacity and plant layout.

The main scope of work items covered by the capital cost estimate are summarized as follows:

- Process plant office.
- Remote control room at Crushing.
- Central plant control room.

Commissioning

The plant commissioning cost estimate included the relevant management, engineering experts, supervision and construction crew to attend any testing or minor modifications.

The team will consist of engineers from various disciplines, the relevant vendor commissioning representatives and a team of tradesman selected form the construction workforce.

Certain items of major equipment such as the crushers, mills, flotation cells, filters, and the thickeners will require vendor representation on site during final commissioning. A vendor commissioning cost and travel have been included in the estimate for this purpose. This expenditure will ensure that equipment warranties are preserved and any early operating issues are resolved ahead of hand over.

21.4.6 Construction Infrastructure

Project construction offices and establishment, construction services, power, water, PPE, communications, computers, IT services, servers, and telephones are all included in the capital estimate.

A heavy lift crane of 250 t capacity has been included in the estimate for a six-month duration.

21.4.7 Engineering, Procurement and Construction Contract (EPC) Services

The EPCM cost estimate is based on the Project being implemented using an EPCM approach, whereby the EPCM Engineer will provide design, procurement, and construction management services on behalf of the Owner based on the Project schedule.

The EPCM services cost estimate includes head office support and site staffing, sub-consultants, office consumables, equipment, and associated project travel. The cost of a fully equipped home design office and all project computing requirements are included under Management costs.

21.4.8 Owner Costs

As part of the project development, K92ML will be required to provide the Owner's project management team.

The Owner's construction team will interact closely with operations management personnel recruited during the construction phase of the Project.

In addition to the above, the following allowances have been made in the estimate:

- Geotechnical assessments and any ground improvement.
- Project Contingencies.
- Import duties and taxes.
- Pre-production costs.
- First fills (lubricants, fuel, and reagents).
- Opening stocks.
- Plant mobile equipment.
- Project spares.
- Vendor representative and training costs for the process plant.

21.4.9 Spares

A value for the Project spares has been estimated by GRES and included in the estimate.

21.4.10 Duties, Taxes, and Insurances

Government Duties and Taxes

The capital estimate excludes government taxes and duties.

21.4.11 Contingency

The contractors have included contingency associated with their fixed costs and schedule. This does not include scope changes or impacts beyond their contract allowances or obligations.

21.4.12 Escalation

There is no allowance for project escalation in the capital estimate.

21.4.13 Qualifications and Assumptions

The capital estimate is qualified by the following assumptions:

- Potable water and construction raw water supply will be provided by the client and available at site for the use by contractors.

- The bulk commodities for earthworks that include imported material, assume that suitable construction / fill materials will generally be available from borrow pits within 2 km of the work fronts, other than roads that will likely have longer haulage distances. Concrete imported materials have been included in the concrete installation rates by the contractor.
- Concrete aggregates, sand, concrete & water free issued by K92.
- There is no allowance for blasting in the bulk earthworks.
- Meals and accommodation for all EPCM and any sub-contractors free issued by K92.
- No allowance for security infrastructure or personnel, assumed provided by Owner.

21.4.14 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Owner's Team Project and expenses (Numbers provided by K92).
- Working capital (included directly in the financial model).
- Permits and licences.
- Project sunk costs.
- Taxes and duties.
- Exchange rate variations.
- Escalation.

21.5 Process Plant and Administration Operating Costs

21.5.1 General

The process plant operating costs have been compiled by GRES. The new treatment plant operating costs have been developed on the basis of a process plant feed tonnage of 1.2M tpa and include all direct processing costs associated with producing doré and a saleable concentrate.

The new treatment processing costs have been developed from the plant parameters specified in the process design criteria and have been compiled from a variety of sources, including:

- Data supplied by K92ML.
- Production schedule starting in Q1 2024.
- Labour costs applicable for PNG based mining and processing operations provided by K92ML.
- Consumables prices from supplier quotations.
- Consultant reports.
- Reagent consumptions based on metallurgical test work results, existing Kainantu plant operating data and vendor advice.
- First principle estimates.
- GRES data based on similar operations.

The operating cost estimate is considered to have an accuracy of +15% / -10%. The costs are presented in United States Dollars and are based on prices in effect during the first half of 2024 (1H24). The source data was collected in United States Dollars (USD), Papua New Guinean Kina (PGK) or Australian Dollars (AUD). Exchange rates used to develop the operating cost estimate were as follows:

- USD $1.00 = AUD $1.42.
- PGK 1.00 = USD $0.24.

21.5.2 Summary

The high level breakdown of the new treatment plant processing cost estimate is presented by cost centre in Table 21-9.

Table 21-9 New 1.2M tpa Treatment Plant Operating Cost Summary by Cost Centre

Cost Centre	Kora Ore - 1.2M tpa		
	USD/t	Fixed	Variable
		USD'000/y	USD/t
Power	$5.17	$1,863	$3.62
Operating Consumables	$3.28	-	$3.28
Process and Maintenance Labour	$6.85	$8,216	-
Maintenance Materials & Other	$2.25	$1,824	$0.73
Laboratory	$0.57	$577	$0.09
Total	**$18.12**	**$12,480**	**$7.72**

The high-level breakdown of the processing cost estimate on an annual basis is summarised in including Life of Mine (LOM) weighted average processing costs by cost centre.

Table 21-10 Annual Operating Cost Summary by Cost Centre for Kora Ore Treatment

	2024	2025	2026	2027	2028	2029	2030	LOM
Production	**kt**	**kt**	**kt**	**kt**	**kt**	**kt**	**kt**	**kt**
New Treatment Plant	-	380	1,001	1,200	1,200	1,200	302	5,283
Existing Plant	483	409	-	-	-	-	-	892
Total	**483**	**789**	**1,001**	**1,200**	**1,200**	**1,200**	**302**	**6,175**
Cost Centre	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t	USD/t
Power	$5.68	$5.61	$5.48	$5.17	$5.17	$5.17	$5.68	$5.34
Operating Consumables	$4.42	$3.87	$3.28	$3.28	$3.28	$3.28	$3.28	$3.44
Process & Maintenance Labour	$7.16	$7.07	$8.21	$6.85	$6.85	$6.85	$9.06	$7.23
Maintenance Materials & Other	$4.88	$3.77	$2.55	$2.25	$2.25	$2.25	$2.74	$2.73
Laboratory	$1.30	$0.91	$0.67	$0.57	$0.57	$0.57	$0.72	$0.69
Total	**$23.44**	**$21.24**	**$20.19**	**$18.12**	**$18.12**	**$18.12**	**$21.48**	**$19.43**

Process and maintenance labour is the largest cost item for the new treatment representing $6.85/t of ore processed for a full production year. Power is the second largest cost item representing $5.17 /t of ore. Over the life of mine the operating costs vary with higher operating costs achieved during the ramp-up period and first twelve months of operation of the new treatment plant at approximately 80% of the design production rate.

21.5.3 Summary Battery Limits

The battery limits for the processing operating costs are as follows:

- Ore on the ROM pad.
- Tailings discharge to existing TSF or tailings filtration plant.
- Gold doré in vault on site.
- Saleable concentrate in 20 ft shipping containers on site.
- Raw Water in the Raw Water Sediment Ponds.

21.5.4 Qualifications and Exclusions

The following qualifications apply to the operating cost estimate:

- The power cost estimate for the new treatment plant has been based on grid power, with an electricity cost of USD 0.156/kWh. The average power draw for each drive has been calculated from the installed power and the application of anticipated power draw factors.

- The plant maintenance spares and consumables cost for the new treatment plant has been estimated by factoring spares and consumables from the from the capital cost estimate for each section of the plant using factors from the GRES database.
- Labour costs for the new treatment plant have been based on a manning estimate and salaries applicable for PNG based mining and processing operations provided by K92ML There will be a total of 63 operations, 89 maintenance and 40 contract laboratory personnel including 16 supervisory or management personnel once the new treatment plant is operating at the Stage 3 run-rate in 2027. Nine expatriate supervision roles have been allowed for in the estimate. All other roles will be filled by PNG nationals.
- The labour estimate included 16 operations personnel for the mobile crushing and concrete batching plants used at the existing Kainantu operation.
- Labour costs for the new treatment plant are inclusive of on-costs (70% for expatriates, 117% for PNG nationals has been allowed based on K92ML advice).
- Flight costs and camp accommodation costs have been allowed within the new treatment plant operating cost estimate.
- Plant mobile equipment maintenance and diesel costs have been included.
- ROM loader to the crusher included.
- Lease costs have been included for the new treatment plant (rock breaker, crane etc.).
- Laboratory and assay costs have been estimated for the new treatment plant by factoring operating costs provided by K92MLfor the existing site laboratory and include laboratory consumables and labour.
- The processing costs for the existing Kainantu process plant have been estimated based on 2023 and 2024 YTD operating cost data provided by K92ML.

The following items have been excluded from the operating cost estimate for the process plant however these have been included in other cost categories, or as deductions in the economic model:

- All sunk costs have been excluded.
- Government monitoring and compliance costs have been excluded.
- All K92ML head office costs and corporate overheads have been excluded.
- Withholding taxes and other taxes, such as GST or VAT have been excluded.
- Escalations from the date of estimate have been excluded.
- Financing costs have been excluded.
- Foreign exchange fluctuations have been excluded.
- Interest charges have been excluded.
- Land or crop compensations costs have been excluded.
- Subsidies to local communities have been excluded.
- Licence fees have been excluded.
- Royalties have been excluded (these are included separately in the economic model).
- No allowance for contingency has been made.
- No allowance for Insurances has been made.
- Security contract costs have been excluded.
- Labour union fees have been excluded.
- Amortization and depreciation charges have been excluded.
- All mining and exploration costs, including mining services have been excluded.
- Maintenance costs of all mine, haul and plant access roads have been excluded.
- No allowance has been made for offsite bullion transport and refining costs (these are included separately in the economic model).
- Concentrate transport, marketing, and insurance costs have been excluded (these are included separately in the economic model).
- Concentrate packaging for transport has been excluded.
- Concentrate smelting and refining costs have been excluded (these are included separately in the economic model).
- External assaying charges have been excluded.
- Tailings storage facility costs, including future lifts and rehabilitation have been excluded.

- Water supply costs have been excluded.
- Tailings filtration and Paste plant operating costs have been excluded.
- First fills costs have been excluded.
- External government required tailings monitoring and compliance costs have been excluded.
- Any environmental rehabilitation or closure costs have been excluded.

21.5.5 General and Administration Costs

The general and administration (G&A) costs have been compiled by K92 and is based on the following items:

- Site services maintenance.
- Finance and administration.
- External affairs and social development.
- Security.
- Health, safety, and environment.

The G&A cost over the life of mine equates to USD $25.39 /t mined tonne ($37.11 /ore tonne).

21.6 Paste Fill Plant Capital Costs

21.6.1 Introduction

This section provides capital cost estimates for the proposed paste backfill system to AACE Class 3 accuracy. Pricing provided includes provision for the Design and Procurement of all equipment, Civil works, Structural, Mechanical and Piping (SMP), Electrical and Instrumentation (E&I) installation activities. The battery limits and required inputs for each process are described in Section 21.6.4.

21.6.2 CAPEX Estimate

The estimated system capital cost is presented in Table 21-11 and is based on the following exchange rates.

- USD1.00 = AUD 1.42 (Australian Dollar).
- USD 1.00 = KINA 0.24 (PNG Dollar).
- USD1.00 = EURO 0.92 € (Euro).

Table 21-11 Estimated Kora Paste Backfill System Capital Cost Estimate (AACE Class 3 Estimate)

Cost Category	Project Area	Procure USD$M	Installation USD$M	Total (USD$M)
Direct Costs	Tailings Filtration Plant	$7.0	$8.9	$15.9
	Binder Mixing & Storage (Bus Stop)	$2.7	$3.6	$6.3
	Underground Paste Mixing & Pumping	$3.9	$6.9	$10.8
Indirect Costs	EPCM & Construction Indirects		$9.2	$9.2
	First Fills		$0.3	$0.3
	Total			**$42.5 M**

The cost estimates are based on contracting costs for the 1.2M tpa process plant EPC contract. The contingencies included account for any design growth and equipment cost variance. Productivity impacts have also been included for the underground construction works. Refer Section 21.4.2 for basis of estimating.

21.6.3 Battery Limit Summary

The assumed battery limits for all processes are as follows:

Incoming:

- Upper side of bulk earthworks.
- Power incomer into the paste plant MCC busbar at each plant location.
- Tailings available at the outlet flange of the tailings thickener underflow pump discharge.
- Potable water delivered to each of the paste plant locations, primarily for safety showers.
- Raw water Filtered raw of suitable quality for gland seals delivered to the bund of the paste plant at the various locations filter plant area.
- Fibre communications network connection between the filter plant and the main process plant included. There is no allowance for communications between the main plant and the Bus Stop or the underground paste mixing station.
- Ground Granulated Blast Furnace Slag delivered to the Bus Stop area.
- GP Cement delivered in 20t ISO containers suitable for transport underground.
- Tailings overflow discharged to the existing TSF as a single point discharge.
- Mine water delivered underground suitable for paste line flushing and washdown.
- All mine excavations and associated boreholes and drilling supplied by the mine.

Outgoing:

- Filtrate discharged at the process plant.
- Flush paste from the underground reticulation through the paste mixer and pumping system into cleanout sumps or stopes.

21.6.4 Exclusions

- No provision has been made for ground improvement works at the Filter building, Bus Stop or Underground.
- No Provision has been made for the supply of services outside of battery limits.
- Supply of all water sources to storage tanks at the paste plant sites is excluded.
- Supply of HV power and transformers as well as cabling and termination LV power onto the switchroom busbar is excluded.
- Excavation and support of underground excavations is excluded from this section of the report but included in the relevant mining section.
- Supply of mine compressed air to the underground receivers and instruments is excluded.
- Supply of power and communications to all nodes in the Project, including pipeline instrumentation and valves, is excluded.
- Permitting for construction and operation of the paste system, including acquisition of land for the Bus Stop transfer area and dump sumps is excluded.
- Supply of concrete materials is excluded (i.e. it is assumed that the client would free issue with strength of 32 MPa).
- Supply of fuel to site mobile equipment and gensets is excluded (i.e. fuel is assumed to be free issued).
- No provision has been made for design and management staff travel from Australia to site or messing and accommodation while onsite.

21.7 Paste Fill Plant Operating Costs

21.7.1 General

The paste backfill system operating costs have been compiled by GRES. The operating cost for the paste backfill system have been developed on the basis of a paste being placed at an average rate of 465,226 m^3/year at 4.0% binder.

The paste backfill system costs have been developed from the parameters specified in the WSP Kainantu paste backfill system design criteria and have been compiled from a variety of sources, including:

- Data supplied by K92 Mining Inc.

- Mine schedule generated as of Jan-2024.
- Labour costs applicable for PNG based mining and processing operations provided by K92 Mining Inc.
- Consumables prices from supplier quotations.
- Consultant reports.
- Laboratory test work results.
- GRES data based on similar operations.

The operating cost estimate is considered to have an accuracy of AACE Class 3 estimate. The costs are presented in United States Dollars and are based on prices in effect during the first quarter of 2024 (Q1 2024). The source data was collected in United States Dollars (USD), Papua New Guinean Kina (PGK) or Australian Dollars (AUD). Exchange rates used to develop the operating cost estimate.

Exchange rates used to develop the operating cost estimate were as follows:

- USD $1.00 = AUD $1.42.
- PGK 1.00 = USD $0.24.

21.7.2 Summary

A high level breakdown of the operating costs for the paste backfill system is presented by cost centre in Table 21-12. For the purpose of this analysis, the operating costs have been determined per cubic metre of paste placed.

Table 21-12 Paste Backfill Plant Operating Cost Summary by Cost Centre

Cost Centre	USD/m^3	Fixed	Variable
		USD'000/y	USD/m^3
Power	$2.00	-	$2.00
Operating Consumables	$11.49	-	$11.49
Process and Maintenance Labour	$3.66	$1,704	-
Maintenance Materials & Other	$1.92	$244	$1.40
Vehicles/Mobile Plant	$0.33	$154	-
Underground Reticulation	$3.71	-	$3.71
Tailings haulage from Bus stop to U/G backfill plant	$5.10	-	$5.10
Laboratory	$0.08	-	$0.08
Total	**$28.30**	**$2,102**	**$23.78**

Operating consumables represents the largest cost item for the paste backfill system at $11.49/m^3 of paste placed. The major operating consumable is binder at $10.66/m^3 of paste. The binder cost alone represents approximately 40% of the overall operating costs for the paste backfill system. Tailings filter cake haulage from the bus stop area to the underground paste backfill plant is the second largest cost item representing $5.10/m^3 of paste placed.

21.7.3 Qualifications and Exclusions

The following qualifications apply to the operating cost estimate:

- The paste backfill system operating costs are inclusive of tailings filtration, bus stop area and underground backfill plant.
- Underground reticulation costs have been included. Database rates for various specification pipe and valves and fittings were applied to the developed quantities.
- Underground containment bulkheads have been included.

- Costs to haul filtered tailings from bus stop area to underground backfill plant on a contract basis have been provided by K92 Mining Inc.
- The power cost estimate for the paste backfill system has been based on grid power, with an electricity cost of USD 0.156/kWh. The average power draw for each drive has been calculated from the installed power and the application of anticipated power draw factors.
- The plant maintenance spares and consumables cost for the paste backfill system has been estimated by factoring spares and consumables from the from the capital cost estimate for each section of the plant using factors from the GRES database.
- Binder costs have been based on a 90:10 blend of ground granulated blast furnace slag and GP cement.
- Ground Granulated Blast Furnace Slag costs are based on a price USD $204/t delivered to site.
- Cement costs are based on a price of USD $116/t delivered to site.
- Plant mobile equipment maintenance and diesel costs have been included.
- Diesel costs have been based on industry standard fuel burn rates and a diesel price of USD $0.67/L;
- Labour costs for the paste backfill system have been based on a manning estimate and salaries applicable for PNG based mining and processing operations provided by K92 Mining Inc. There will be a total of 9 operations personnel for the tailings filter plant, 18 operations personnel for the bus stop area and a total 26 underground paste backfill personnel including 5 supervisory or management personnel.
- Labour costs for the paste backfill system are inclusive of on-costs (117% for PNG nationals has been allowed based on K92 advice).
- Flight costs and camp accommodation costs have been allowed within the new treatment plant operating cost estimate.
- Quality control QA/QC costs have been included.

The following items have been excluded from the operating cost estimate:

- All sunk costs have been excluded.
- Government monitoring and compliance costs have been excluded.
- All K92 head office costs and corporate overheads have been excluded.
- Withholding taxes and other taxes, such as GST or VAT have been excluded.
- Escalations from the date of estimate have been excluded.
- Financing costs have been excluded.
- Foreign exchange fluctuations have been excluded.
- Interest charges have been excluded.
- Land or crop compensations costs have been excluded.
- Subsidies to local communities have been excluded.
- Licence fees have been excluded.
- No allowance for contingency has been made.
- No allowance for Insurances has been made.
- Security contract costs have been excluded.
- Labour union fees have been excluded.
- Amortization and depreciation charges for the mobile equipment have been excluded.
- No allowance for haulage of filtered tailings from tailings filter plant to bus stop area within the operating costs backload on mining fleet.
- No allowance for plasticiser has been made.
- No allowance has been made for offsite quality control or testing of paste samples.
- Maintenance costs of all mine, haul and plant access roads have been excluded.
- Water supply costs have been excluded.
- First fills costs have been excluded.
- External government required tailings monitoring and compliance costs have been excluded.
- Any environmental rehabilitation or closure costs have been excluded.

21.8 Tailings Storage Facility Capital Costs

The total estimated capital cost required to expand the existing Tailings Storage Facility (TSF) to accommodate the demands of the Definitive Feasibility Study (DFS) scenario for the Kora Expansion project is USD $7.9 million. This estimate includes a contingency allowance of 15% to account for potential variations in embankment construction costs.

21.9 Tailings Storage Facility Operating Cost

The estimated annual operating cost for managing the tailings facility is set at an average of USD $0.6 million. This estimate does not take into account the rate of discount and reflects the ongoing cost of operations. Significant components of these operating expenses include fees for consultant activities, which are crucial for ensuring compliance and optimizing tailings management strategies.

21.10 EIS Capital Cost Estimate

The EIS for the current mining lease has been completed and as such no costs are required for the DFS.

22. ECONOMIC ANALYSIS

22.1 Introduction

An economic analysis has been carried out for the project using a cash flow model. The model is constructed using annual cash flows by considering annual mined and processed tonnages and grades, process recoveries, metal prices, operating costs, refining charges, royalties, and capital expenditures (both initial and sustaining).

PNG tax regulations are applied to assess the tax liabilities. All amounts in this section are presented in USD.

The cash flow model commences with operation in 2024, with processing through the existing plant. Discounting has been applied monthly from the first year of operation.

The model reflects the base case and technical assumptions as described in the foregoing sections of this report.

22.2 Model Inputs and Assumptions

The key model inputs used in the economic analysis for the DFS are summarized in Table 22-1.

Table 22-1 Model Inputs

Model Inputs	Source	Unit / Value
Base Currency		USD
Base Date		1st Quarter 2024
Ore Processed over LOM	Entech	6.18 Mt
Metal Prices		
Gold	K92ML	$1,900 / oz (fixed)
Copper	K92ML	$4.50 / lb (fixed)
Silver	K92ML	$25 / oz (fixed)
Recoveries		
Gold (total)	MMS	92.6%
Gold (gravity to doré)	MMS	15.0%
Copper	MMS	94.0%
Silver	MMS	78.0%
Concentrate copper grade	MMS	18.7%
Processing Plant Capacity (dry tonnes of ore)		
Existing Plant	K92ML	600,000 tpa
New Plant	GRES	1,200,000 tpa
Royalties (deducted from net revenue)	K92ML	2.0%
Levies (deducted from gross revenue)	K92ML	0.5%
Tax Rate	K92ML	30%
Depreciation		Not considered
NPV Discount Rate	K92ML	5%

22.2.1 Ore Processed

Total ore processed from 2024 to 2030 is 6.18 Mt.

Average head grade over LOM is:

- Gold 6.7 g/t.
- Silver 19 g/t.
- Copper 1%.

22.2.2 Gross Revenue

Gross revenue from doré and concentrate (excluding treatment and refining charges and operating costs) is USD $2,863M over the LOM, or USD $464/t ore.

22.2.3 Treatment and Refining Charges

Total treatment and refining charges, including penalty element deductions equates to USD $13 /t ore.

22.2.4 Operating Costs

Annual fixed and variable costs, as per Sections 21.5 and 21.6, are included in the cash flow and summarized in Table 22-2.

Table 22-2 Operating Costs

Area	Source	USD / Value
Mining (average over LOM)	Entech	$68 / t ore
Processing Plant - Existing - Fixed	GRES	$5.4M / year
Processing Plant - Existing - Variable	GRES	$12.3 / t ore
Processing Plant - New - Fixed	GRES	$12.5M / year
Processing Plant - New - Variable	GRES	$7.7 / t ore
Processing Plant - New - Total (average over LOM)	GRES	$19.4 / t ore
General & Admin - Total (average over LOM)	K92ML	$37.1 / t ore
Paste Plant - Fixed	GRES	$2.1 M / year
Paste Plant - Variable	GRES	$23.8 / m3 paste
Paste Plant - Total (average over LOM)	GRES	$29.6 / m3 paste
TSF (average over LOM)	ATC Williams	$0.64 / t ore
Transport & Insurance (average over LOM)	K92ML	$9.85 / t ore
Total (average over LOM)		**USD $145.4 / t ore**

22.2.5 Capital Costs

Pre-production capital expenditures for the DFS are defined in Table 22-3. Sustaining capital for the Plant, Mining and TSF expansion costs have been phased over the life of the project and detailed in Table 22-4. (This summary only include capital cost from 2024 onwards.)

Table 22-3 Pre-Production Capital Expenditure

Item	Source	Total (USD M)
Mine		Refer sustaining capital
Process Plant	GRES	$90.3
Power	GRES	$19.1
Camp Upgrade	K92ML	$4.2
Haul Road Bridges	K92ML	$14.4
Warehouse / Maintenance Facilities	K92ML	$3.8
Owner's Costs	K92ML	$20.1
Paste Plant	GRES	$42.5
Total		**$194.4**

Table 22-4 Sustaining Capital Expenditure

Item	Source	Total (USD M)
Mine	Entech	$265.3
TSF	ATC Williams	$7.9
TSF Closure Costs	ATC Williams	$9.0
Processing Plant	K92ML	$5.4
Surface Fleet	K92ML	$39.3
Site Services	K92ML	$19.3
Total		**$346.1**

22.2.6 Royalties

A royalty of 2.0% has been deducted from net revenue.

A levy of 0.5% has been deducted from gross revenue.

22.2.7 Depreciation

Depreciation has not been considered in the cash flow model.

22.2.8 Inflation

Inflation has not been included in the cash flow analysis. All costs are input at nominally Q1 2024 values.

22.3 Financial Model Results

The financial model indicates that the DFS project has a post-tax Net Present Value (NPV) of USD $680.5M at a discount rate of 5%.

Figure 22-1 shows the post-tax annual and cumulative cash flow for the project over the LOM.

Cash flow starts negative in 2024 and continues to 2025 due to the current expenditure on the stage 3 expansion project.

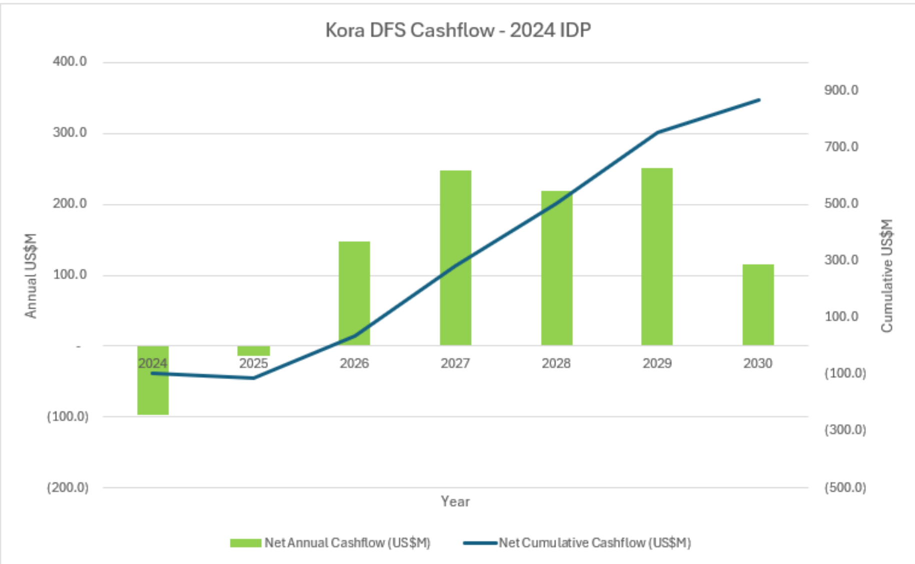


Figure 22-1 Cumulative Cash Flow

Table 22-5 shows a summary of the cash flow model for the project.

Table 22-5 Cash Flow Model Summary

	Source	Units	2024	2025	2026	2027	2028	2029	2030	TOTAL
Period Ending			31/12/2024	31/12/2025	31/12/2026	31/12/2027	31/12/2028	31/12/2029	31/12/2030	
Gross Revenue - After Treatment, Refining Costs & Penalties		**US$ 000's**	**211,735**	**310,242**	**405,349**	**575,357**	**506,877**	**547,468**	**225,166**	**2,782,193**
Total Operating Costs		**US$ 000's**	**90,752**	**122,035**	**149,508**	**165,164**	**160,143**	**159,120**	**51,235**	**897,957**
Total Royalties & Levies		**US$ 000's**	**5,128**	**7,514**	**9,842**	**13,939**	**12,133**	**13,274**	**5,470**	**67,299**
Total Capital Costs		**US$ 000's**	**189,210**	**158,678**	**46,515**	**55,854**	**37,881**	**30,343**	**21,980**	**540,461**
FINANCIAL SUMMARY										
Total Revenue		US$ 000's	211,735	310,242	405,349	575,357	506,877	547,468	225,166	2,782,193
Total Outgoings		US$ 000's	285,090	288,227	205,865	234,957	210,157	202,737	78,685	1,505,716
Earnings before Interest, Tax, Depreciation and Amortisation (EBITDA)		**US$ 000's**	**-73,355**	**22,015**	**199,484**	**340,401**	**296,720**	**344,731**	**146,481**	**1,276,477**
Taxable Profit / Loss		US$ 000's	(73,355)	22,015	199,484	340,401	296,720	344,731	146,481	1,276,477
Tax Payable		US$ 000's	24,499	35,962	52,065	92,953	77,750	92,997	30,828	407,056
Net Profit After Tax		US$ 000's	(97,854)	(13,947)	147,419	247,448	218,969	251,734	115,653	869,421
Cash Flow										
Cash Flow - After Tax		000's US$	(97,854)	(13,947)	147,419	247,448	218,969	251,734	115,653	869,421
Accumulated Cash Flow		000's US$	(97,854)	(111,801)	35,618	283,065	502,034	753,768	869,421	
Payback Period - Simple		years	1.00	1.00	0.76	-	-	-	-	2.76
Discounted Cash Flow - Monthly Discounting										
Discounted Cash Flow @ 5%	K92 Calc	US$ 000's	(95,460)	(13,740)	130,045	208,536	174,481	192,672	83,953	680,487
Discounted Cash Flow @ 8%	K92 Calc	US$ 000's	(94,111)	(13,600)	120,974	188,937	153,057	165,125	69,781	590,163
Discounted Cash Flow @ 10%	K92 Calc	US$ 000's	(93,245)	(13,501)	115,413	177,179	140,545	149,343	61,865	537,599
Discount Rate		%	**5.0%**	**8.0%**	**10.0%**					
Net Present Value (NPV) - After Tax		000's US$	**680,487**	**590,163**	**537,599**					

22.4 Sensitivities

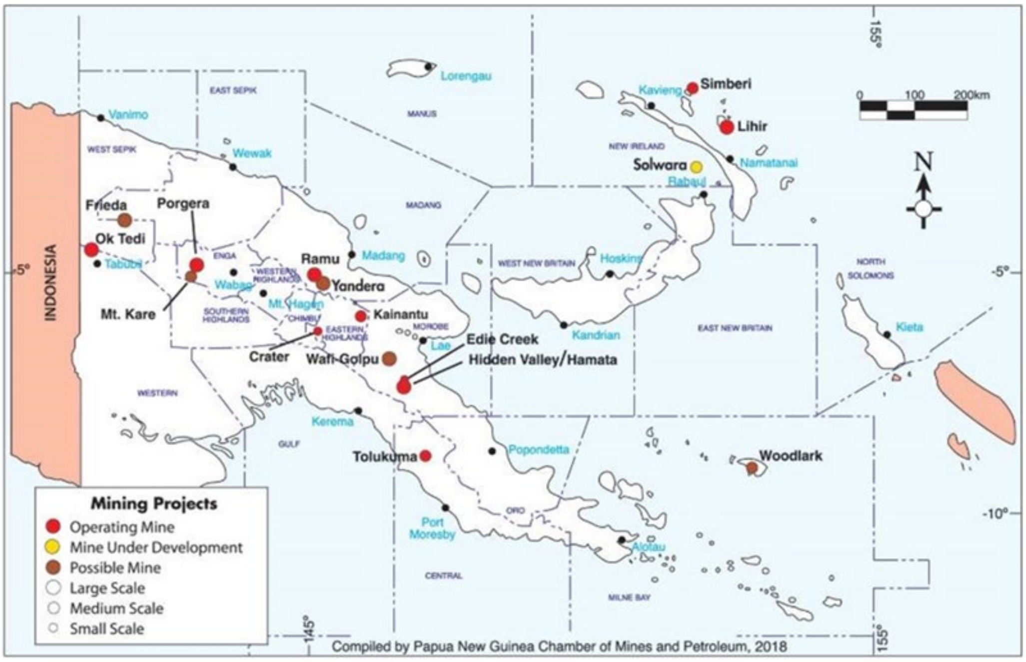
Table 22-6 shows the project NPV5% with different gold prices.

Table 22-6 NPV Sensitivity

Gold Price	After-Tax NPV5% (USD M)
$1,600	$475
$1,900	$680
$2,200	$886
$2,500	$1,091
$2,800	$1,296
$3,100	$1,501

23. ADJACENT PROPERTIES

Kainantu occurs within a well-endowed belt of epithermal and porphyry style mineralization that reportedly contains several major deposits (Figure 23-1). K92ML is unable to verify this information, and the information is not necessarily indicative of the mineralization on the property that is the subject of this technical report.



Source: PNG Chamber Mines and Petroleum (2018)

Figure 23-1 Location of Kainantu Project and Gold Deposits Within Major Mineralized Province

K92ML does not have any interest in any adjacent properties.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Kora 2024 PEA Introduction

The alternative PEA Case conceptualizes a multi-expansion plan to an ultimate plant run-rate of 1.8M tpa, representing a 200% increase from the upgraded Stage 2A Expansion run-rate of 600,000 tpa design throughput (previously 500,000 tpa). The PEA Case involves the construction of a standalone 1.2M tpa process plant adjacent to the 600,000 tpa Stage 2A Expansion process plant. In mid-2025, the Stage 2A Expansion process plant is idled as the 1.2M tpa Stage 3 process plant ramps up, with commissioning of the Stage 3 Expansion process plant commencing in late Q2 2025. Upon achieving the Stage 3 run-rate throughput in 2027, the Stage 2A Expansion plant is recommissioned in H2 2027, ramping up to run-rate throughput of 600,000 tpa by year end, for a combined processing run-rate of 1.8M tpa at the beginning of 2028.

To support the higher throughput rate, the underground mining fleet is significantly increased to support expanded mining operations opening up multiple mining fronts concurrently: Kora Upper, Lower and Central Zones within the Kora deposit, and the Judd deposit. Site infrastructure is also expanded, including power, camp facilities and the paste fill plant. Several capital items, such as the paste fill system, are configured during the construction of Stage 3 to be amenable to the larger ultimate Stage 4 Expansion run-rate.

The PEA uses the conclusions of the Company's Mineral Resource estimate for Kora and Judd (effective date of September 12, 2023) and does not incorporate post resource drilling results. The effective date of the PEA life of mine plan is January 1, 2024; therefore, Kora is net of post-resource mining depletion from September 12, 2023 to December 31, 2023 of 183,768 tonnes at 8.05 g/t Au, 0.92% Cu and 15.37 g/t Ag.

The PEA is preliminary in nature and 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, and there is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 24-1 PEA Highlights

USD unless otherwise stated	Life of Mine (starting January 2024)	Stage 4 Run-Rate[(1)] (2028 - 2034)
Production		
Mine life (years)	14 years	
Total mill feed (000s tonnes)	20,396	12,600
Average mill throughput (tonnes per annum)	1,457k tpa	1,800k tpa
Total Metal Production		
AuEq (000s ounces)	4,977	2,895
Gold (000s ounces)	3,620	2,011
Copper (Mlbs)	508	332
Silver (000s ounces)	11,799	7,399
Peak Annual Production		
Year	2034	
AuEq (000s ounces per annum)	485	
Average Annual Metal Production		
AuEq (000s ounces per annum)	356	414
Gold (000s ounces per annum)	259	287
Copper (Mlbs per annum)	36	47
Silver (000s ounces per annum)	843	1,057

USD unless otherwise stated	Life of Mine (starting January 2024)	Stage 4 Run-Rate[1] (2028 - 2034)
Average Grade		
AuEq grade (g/t)	8.2 g/t	
Gold grade (g/t)	6.0 g/t	
Copper grade (%)	1.2%	
Silver grade (g/t)	23 g/t	
Average Recovery		
Gold Recovery (%)	93%	
Copper Recovery (%)	94%	
Silver Recovery (%)	78%	
Costs		
Mining Cost ($/t ore mined)	$62.60	$58.58
Processing Cost ($/t processed)	$18.19	$17.65
General & Administrative Cost ($/t processed)	$25.45	$23.33
Paste Plant Cost ($/t processed)	$9.16	$9.89
TSF ($/t processed)	$0.48	$0.42
Transport and Insurance ($/t processed)	$10.24	$10.44
Total operating cost per tonne processed ($/t)	$126.12	$120.59
Royalties ($/t processed)	$10.75	$10.10
Sustaining capital per tonne of processed ($/t)	$44.15	$35.57
Total cost per tonne of processed ($/t)	$181.02	$166.06
Growth capital expenditure ($M)	$201	
Sustaining capital expenditure ($M)	$900	
Total capital expenditure with closure costs ($M)	$1,122	
Cash cost per ounce AuEq ($/oz)[2]	$633	$644
All-in sustaining cost per ounce AuEq ($/oz)[3]	$822	$805
Cash cost per ounce gold ($/oz)[2]	$174	$108
All-in sustaining cost per ounce gold ($/oz)[3]	$432	$338
Base Case Economic Analysis at USD 1,900/oz Gold, USD 4.50/lb Copper and USD 25.00/oz Silver		
After-tax NPV0%	$3.5 billion	
After-tax NPV5%[4]	$2.3 billion	
Economic Analysis at USD 2,500/oz Gold, USD 4.50/lb Copper and USD 25.00/oz Silver		
After-tax NPV0%	$5.0 billion	
After-tax NPV5%[4]	$3.3 billion	

1. *Run-rate calculated based on 2028-2034.*
2. *Cash costs are net of by-product credits and are inclusive of mining costs, processing costs, site G&A and refining charges and royalties.*
3. *AISC includes cash costs plus estimated corporate G&A, sustaining costs and accretion.*
4. *Net present value is calculated utilizing monthly discounting.*

24.2 Kora 2024 PEA Mining

24.2.1 Introduction

The mine plan considered in the PEA is designed as an incline access operation with a series of ore passes for efficient material movement between sublevels and the twin incline for material transport to surface. Initially, the mine plan employs a long hole open stoping mining method utilizing Avoca and modified Avoca, with waste rockfill. This mining method has been successfully employed at the mine since Q1 2020 at Kora and Q4 2021 at Judd. The mining method will then transition to long hole open stoping with paste fill in Q3 2025, upon the construction of the paste fill plant. Cemented paste fill provides improved geotechnical conditions for less dilution, higher mining recovery factors, and greater operating flexibility through the ability to mine above and below paste fill, plus a reduction in surface tailings.

Stopes were identified for the PEA mine plan based on the MSO program at a cut-off grade of 4.0 g/t AuEq. Stope shapes with uneconomic development access requirements were excluded from the assessment. Dilution was estimated based on a 0.5 m dilution skin for both the footwall and hanging wall using the MSO program for a minimum stope width of 3.0 m. An additional dilution factor of 5% for long hole stoping utilizing Avoca and modified Avoca and 2.5% for long hole open stoping with paste fill. Where a stope is within 5m proximity of the HW or FW of the fault gouge, an additional 1.0 m of dilution was added at a grade averaging 1.42 g/t AuEq. The overall dilution averaged 28.5%. Mining recovery factors of 90% and 95% were applied for long hole open stoping with Avoca and long hole open stoping with paste fill, respectively. The life of mine average head grade is 6.04 g/t Au, 1.21% Cu and 23 g/t Ag or 8.30 g/t AuEq.

The mine plan operates at the Stage 2A Expansion 600,000-tpa throughput until mid-2025, with the operation commencing ramp up to the Stage 3 Expansion run-rate of 1.2M tpa in late Q2 2025, sequentially ramping-up to the Stage 4 Expansion run-rate of 1.8M tpa at the beginning of 2028. To support the Stage 4 mining rate, multiple mining fronts are mined concurrently: Kora Central, Kora Upper and Kora Lower within the Kora deposit, and the Judd deposit.

24.2.2 Underground Operations

Mining Fleet

Assumptions for mine equipment requirements have been derived by K92ML based on actual site productivities and first principles' calculations. The listed equipment types have been reviewed by Entech and are considered appropriate for the mine design and schedule developed by Entech. Table 24-2 provides a summary of the peak number of units required for mine development and production.

This represents the equipment necessary to perform the following duties:

- Excavate the lateral and decline development in both ore and waste.
- Install all ground support including rockbolting and surface support.
- Maintain the underground road surfaces.
- Drill, charge, and bog (including remote loading) all stoping ore material.
- Drill slot rises for production stoping.
- Install all underground services for development and production.

Table 24-2 Peak Fleet Numbers

Equipment	Quantity
Primary	
Development Drills (DD421-60C & DD422i-60C)	6
Loaders (LH517i & LH621i)	9
Trucks (AD45B, TH545')	7
Trucks (Surface - A60H)	11
Trucks (Tailings - 730ADT)	5
Production Drills (DL421-7C/DL431-7C/DS 421C Bolter)	5
Slot Drill - 'Easer L'	1
Raisebore drill - Rhino400H	1
Production Charge-up (Getman)	2
Development Charge-up (Getman)	2
Ancillary	
Spraymec (Maxi-Jet MX3 & Maxi-Jet Sprayer)	3
Agitator (Transmixer X5 & Transmixer 5000)	3
IT CLG856H Development	4
IT CLG856H Production	4
12K Grader	1
Scissor Lift (Ultimec)	1
Light Vehicles	54

Personnel

The personnel requirements have been calculated based on the equipment requirements for underground production. The proposed personnel requirements are based on a mix of Papua New Guinean national employees and expatriate employees. The typical roster for the national employees is 20 days on site and 10 days off, with a roster of 4 weeks on and 4 weeks off for expatriate employees.

Peak underground personnel requirements (total employed personnel, not all on site simultaneously) by month are found in Figure 24-1.

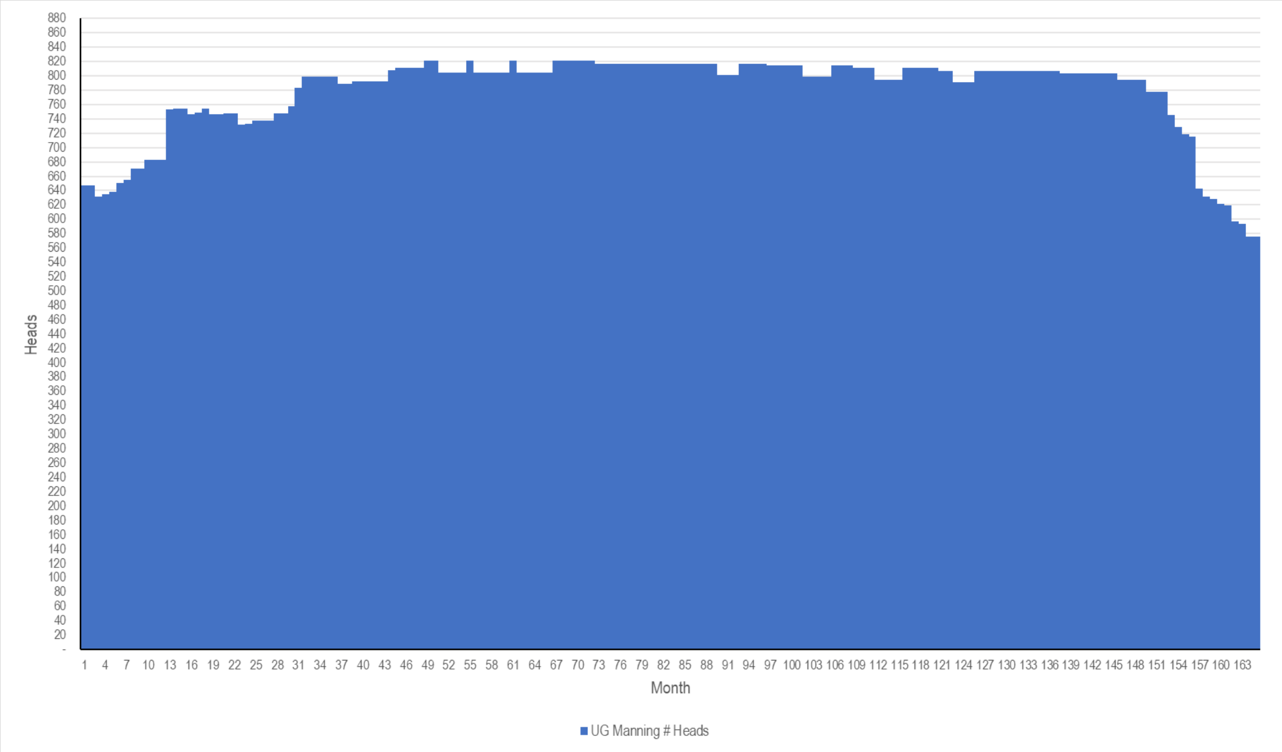


Figure 24-1 Total Underground Mining Personnel Per Month

Mining Capital and Operating Costs

Mine Costing Basis of Estimate

The mine costing estimations were built up using a fixed and variable cost format. The mining cost estimations assume K92 will be executing all mining activities to meet the PEA LOM schedule, except for vertical development and surface haulage which will be completed by mining contractors. Based on current site productivities, K92 utilized Entech's mine plan physicals to estimate the fleet and personnel requirements. Entech reviewed all K92 equipment and personnel estimates and found them appropriate for the mine plan.

Fleet and personnel costs comprise the fixed cost component of the estimate, with all mining and maintenance consumables captured in the variable component. The variable cost component was built up from current site actual costs with personnel and equipment costs built up from current site actual salaries and equipment purchase cost quotations. Other infrastructure such as primary ventilation fans and dewatering pumps utilized manufacturer quotations for pricing estimates. The vertical development schedule has been developed with the help of the onsite raiseboring contractor. The pricing estimate for these activities have been taken from the awarded contract rates.

Mining Capital Costs

The estimated capital costs for the PEA are summarized in Table 24-3.

Table 24-3 Mining Capital Cost Totals

Description	Unit	Value
Infrastructure	USD (M)	$125.2
Decline	USD (M)	$31.7
Cap Access	USD (M)	$16.7
Ventilation	USD (M)	$33.9
Escapeway	USD (M)	$4.3
Other Lateral Development	USD (M)	$141.3
Fleet	USD (M)	$155.4
Operators	USD (M)	$143.6
Capital Mine Services	USD (M)	$45.6
Capital Mine Overheads	USD (M)	$39.7
Total Capital	**USD (M)**	**$737.6**

A breakdown of the capital unit costs for the PEA is shown in Table 24-4.

Table 24-4 Mining Capital Unit Costs

Description	Unit	Value
Infrastructure	$ / t ore	$6.14
Decline	$ / t ore	$1.56
Cap Access	$ / t ore	$0.82
Ventilation	$ / t ore	$1.66
Escapeway	$ / t ore	$0.21
Other Lateral Development	$ / t ore	$6.93
Fleet	$ / t ore	$7.62
Operator	$ / t ore	$7.04
Capital Mine Services	$ / t ore	$2.24
Capital Mine Overheads	$ / t ore	$1.95
Total Capital Cost	**$ / t ore**	**$36.16**

Mining Operating Costs

The estimated operating costs for the PEA are summarized in Table 24-5.

Table 24-5 Mining Operating Cost Totals

Description	Unit	Value
Operating Access	USD (M)	$44.1
Ore Drive	USD (M)	$152.0
Stope	USD (M)	$346.2
Operators	USD (M)	$361.7
Operating Mine Services	USD (M)	$118.1
Operating Mine Overheads	USD (M)	$100.3
Surface Haulage	USD (M)	$115.4
Grade Control	USD (M)	$39.0
Total Operating	USD (M)	$1,276.8

A breakdown of the operating unit costs for the PEA is shown in Table 24-6.

Table 24-6 Mining Operating Unit Costs

Description	Unit	Value
Operating Access	$/t ore	$2.16
Ore Drive	$/t ore	$7.45
Stope	$/t ore	$16.97
Operators	$/t ore	$17.73
Operating Mine Services	$/t ore	$5.79
Operating Mine Overheads	$/t ore	$4.92
Surface Haulage	$/t ore	$5.66
Grade Control	$/t ore	$1.91
Total Operating	**$/t ore**	**$62.60**

24.3 Kora 2024 PEA Processing

The existing Kainantu process plant would be operated in parallel with new 1.2M tpa treatment plant to achieve a combined throughput of 1.8M tpa. The main change to the processing is that the flotation tailings from both plants would be combined for water recovery in the tailings thickener. The combined tailings would be thickened to approximately 59% solids and then pumped either to the TSF for disposal by a two-stage system with duty/standby pumping trains or to the tailings filtration plant.

The PEA conceptualizes constructing a new standalone processing plant for the Stage 3 Expansion adjacent to the existing Stage 2A Expansion process plant. The existing Stage 2A Expansion process plant operates until Q3 2025, where it is idled as the Stage 3 Expansion starts commissiong and ramp-up (starting in late Q2 2025). The process plants are fed with material that is trucked ~6km from the 800 Portal to the Kainantu process plant, where it is either stockpiled or direct tipped.

The Stage 3 Expansion process plant design flowsheet incorporates a conventional comminution circuit, utilizing a single stage of jaw crushing (200 tph) reporting to a crushed ore overflow surge bin, followed by a SAB milling circuit (150 tph) with an open circuit SAG mill and close circuit ball mill. The ball mill product reports to cyclone classification, where the cyclone overflow reports to the flotation circuit and the cyclone underflow stream being split between the gravity circuit, flash flotation circuit and ball mill for grinding. The gravity circuit involves one batch centrifugal concentrator followed by two stages of gravity separation using shaking tables. The upgraded gravity concentrate is then calcined and smelted to produce gold doré. The flotation circuit consists of flash flotation and conventional sulphide flotation, followed by thickening and filtering to produce a gold-copper-silver concentrate.

Processing performance estimates were based on a combination of test work and operating results. Over the period, January 2018 to August 2024, six stages of processing plant upgrades allowed increased throughput and improved recovery performance. The plant has operated in its current configuration (Stage 2A expansion) since June 2023. Metals recovery factors for the PEA were based on an analysis of processing results from June 2023 until August 2024, plus a consideration of economic factors associated with metals offtake agreements and concentrate transport costs.

24.3.1 Kora 2024 PEA Process Plant Operating Costs

The operating costs for the PEA case have been developed on the basis that the existing Kainantu process plant would be operated in parallel with new 1.2M tpa treatment plant to achieve a combined throughput of 1.8M tpa and include all direct processing costs associated with producing doré and saleable concentrates.

The operating cost estimate for the PEA is deemed to be of a level of accuracy consistent with an AACE Class 5 estimate. The costs are presented in United States Dollars (USD) and are based on prices in effect during the first half of 2024 (1H24).

The high-level breakdown of the processing cost estimate for the PEA case on a combined annual basis is summarised in Table 24-7 including total weighted average processing costs for the PEA case by cost centre.

Table 24-7 Operating Cost Summary for PEA Case by Cost Centre

Cost Centre	PEA 1.8M tpa			Total
	USD/t	Fixed USD'000/y	Variable USD/t	USD/t
Power	$5.27	$2,384	$3.95	$5.32
Operating Consumables	$3.66	-	$3.66	$3.63
Process & Maintenance Labour	$4.99	$8,986	-	$5.50
Maintenance Materials & Other	$3.08	$2,762	$1.55	$3.08
Laboratory	$0.65	$1,015	$0.08	$0.67
Total	**$17.65**	**$15,147**	**$9.24**	**$18.19**

Power is the largest cost item for the PEA representing $5.27 /t of material processed for a full year. Process and maintenance labour is the second largest cost item representing $5.17 /t of material processed. Labour estimates for the PEA case are based on senior management and key technical roles being shared for new treatment plant and existing process plant operating in parallel.

24.4 Kora 2024 PEA Paste System

The current paste system is designed to be readily expandable to the meet the backfilling requirements at 1.8 million tonne per annum of ore throughput. The underground paste plant itself is decoupled from the filter plant operations. The filter plant can operate independently of the paste plant and can continue to produce and stockpile filter cake at the portal area until storage capacity is reached. Likewise, the paste plant can continue to operate until all the filter cake is consumed in the storage area by the portal. This concept allowed for a streamlined approach to expanding the paste plant capacity to service the mine as ore production ramps up from 1.2 to 1.8 million tonne annum.

The paste plant at the onset is designed to produce 148 m^3/h of paste to meet the increased mine throughput. For the 1.2 million tonne per annum mining rate, the paste plant simply reduces the hours of operation for backfilling. At the filter plant, to meet the increased throughput, additional plates are added to the pressure filters, in other words, the pressure filter frame and auxiliary systems are all sized to meet the higher throughput upon initial installation.

24.4.1 Kora 2024 PEA Paste Fill Plant Operating Cost

The operating costs for the paste backfill system for the PEA case have been compiled by GRES. The operating cost for the paste backfill system for the PEA case have been developed on the basis of a paste being placed at an average rate of 636,370 m^3/year at 4.0% binder.

The paste backfill system operating cost estimate for the PEA case is deemed to be of a level of accuracy consistent with a AACE Class 5 estimate. The costs are presented in United States Dollars (USD) and are based on prices in effect during the first half of 2024 (1H24).

A high level breakdown of the paste backfill system operating costs for the PEA case is presented by cost centre in Table 24-8.

Table 24-8 Paste Backfill System PEA Operating Cost Summary by Cost Centre

Cost Centre	PEA Case		
	TOTAL	Fixed	Variable
	USD / m^3	USD 000's / pa	USD / m^3
Power	$1.82		$1.82
Operating Consumables	$11.49		$11.49
Labour	$2.68	$1,704	
Maintenance Materials & Other	$1.84	$244	$1.46
Vehicles / Mobile Plant	$0.25	$154	
Underground Reticulation	$3.24		$3.24
Tails Haulage - Bus Stop to UG Plant	$5.11		$5.11
Laboratory	$0.08		$0.08
Total	**$26.51**	**$2,102**	**$23.2**

24.5 Kora 2024 PEA Infrastructure

Power

For the PEA, a further expansion of the back-up power is required.

The new back-up power station will be expanded to 14 x 1.63 MW units for the PEA. The back-up power station will act as back up for the entire Kainantu operation once it becomes operational including for paste backfill system and both the existing Kainantu process plant and new 1.2M tpa treatment plant operating in parallel.

Table 24-9 Electric Power Consumption for the PEA Case

Area	Installed Power (kW)	Average Demand (kW)	Peak Power (kW)
Camp	1,650	1,100	1,200
0.6M tpa Plant	5,000	1,938	2,500
1.2M tpa Plant	8,096	5,443	6,329
Paste Backfill System			
Tailings Filtration Plant	750	510	610
Bus Stop Area	370	290	340
Underground Paste-Backfill Plant and Booster Station	3,872	1,668	1,965
Mine	9,002	6,217	8,299
Total	**28,740**	**17,166**	**21,243**

The PEA incorporates tailings management upgrades, with underground paste fill commencing in Q3 2025 on the completion of construction of the paste fill plant. The delivery of paste fill involves pumping thickened tailings from the process plant, then dewatered in filter presses and trucked to the underground paste fill plant. This is followed by pumping the final paste fill product to void stopes. Remaining tailings are pumped to the tailings impoundment as thickened tailings. The significant increase to processing capacity, mining rates and underground infrastructure requires additions to the standby diesel generating capacity on site. The PEA also involves an upgrade to the local electrical grid to facilitate improved availability and distribution of clean hydroelectricity to the mine. This upgrade is expected to materially reduce overall Scope 1 and Scope 2 greenhouse gas emission intensity per ounce produced.

Accommodation Camp

For the PEA case, the Kumian Camp undergoes further expansion. This includes an additional two 64 bed blocks and an additional 10 two-person ensuite blocks to what is described under Section 18.7 in this report.

24.6 Kora 2024 PEA Tailings Storage

The Preliminary Economic Assessment (PEA) for the Kora Expansion project has suggested that it is feasible to increase the existing processing plant's capacity from 600,000 tpa to 1,800,000 tpa. However, this expansion will consequently increase the total volume of tailings generated. Preliminary assessments have indicated that the existing TSF lacks sufficient capacity to store the additional tailings anticipated.

Over the 2024 - 2037 period, the tailings generated from the Kora Expansion PEA scenario are expected to total 19.4 million tonnes, as outlined in Table 24-10. Once the Paste Backfill plant commences operations by end-2025, approximately 70% of the tailings generated annually will be processed through this facility, with the remaining portion stored in the TSF and a fraction stockpiled to be used as dry backfill into the remaining underground voids at the end of the LOM. The distribution of tailings processed through the paste backfill plant and those stored in the TSF is detailed in Table 24-11.

Table 24-10 Kora Expansion - PEA Production Forecast

Year	Mill Processed Tonnage (t)	Tailings Tonnage (t) (*)	Cumulative Tailings Tonnage (t)
2024	468,999	447,023	447,023
2025	802,460	770,447	1,217,470
2026	1,059,419	1,018,561	2,236,031
2027	1,440,000	1,380,550	3,616,581
2028	1,800,000	1,709,816	5,326,396
2029	1,800,000	1,708,226	7,034,622
2030	1,800,000	1,692,916	8,727,538
2031	1,800,000	1,701,026	10,428,564
2032	1,800,000	1,717,026	12,145,590
2033	1,800,000	1,715,233	13,860,824
2034	1,800,000	1,717,007	15,577,830
2035	1,741,685	1,652,686	17,230,516
2036	1,565,696	1,470,851	18,701,367
2037	717,753	669,219	19,370,587
Total	**20,396,012**	**19,370,587**	

Table 24-11 PEA Forecast - Tailings Production

Year	Tailings Generated (t)	Tailings to TSF (t)	Tailings to Paste Backfill (t)
2024	447,023	447,023	0
2025	770,447	674,017	96,430
2026	1,018,561	415,882	602,679
2027	1,380,550	476,160	904,390
2028	1,709,816	515,283	1,194,533
2029	1,708,226	494,773	1,213,453
2030	1,692,916	444,616	1,248,300
2031	1,701,026	452,726	1,248,300
2032	1,717,026	533,934	1,183,092
2033	1,715,233	491,485	1,223,749
2034	1,717,007	468,707	1,248,300

Year	Tailings Generated (t)	Tailings to TSF (t)	Tailings to Paste Backfill (t)
2035	1,652,686	284,512	1,237,484
2036	1,470,851	33,343	1,235,008
2037	669,219	331,719	0
Total	**19,370,587**	**6,064,178**	**12,635,719**

Since the inception of K92 operations, the TSF has accommodated approximately 1.15 million m^3 of tailings. The PEA forecasts that from 2024 to 2037, around 19.4 million tonnes of tailings will be produced, with approximately 12.6 million tonnes processed by the paste backfill and the remaining 6.1 million tonnes stored in the TSF. Given the assumed settled density of 1.35 t/m^3 for future tailings, a volume capacity of approximately 4.5 million m^3 will be required to store the tailings generated from the Kora Expansion PEA scenario. Given that the existing Tailings Storage Facility (TSF) has a little more than 2 million cubic meters (m^3) of available storage capacity, a high-level desktop study has been conducted to explore options for additional storage capacity.

To accommodate the increased tailings volume anticipated from the PEA Kora Expansion project, several alternatives for expanding the Tailings Storage Facility (TSF) capacity were considered. These include raising the dam embankment of the existing TSF or constructing a new facility.

Raising the embankment crest level to RL 530 m (Stage 3) would extend the facility's lifespan until mid-2029, providing sufficient time to develop the necessary studies, design, and implement a new facility. However, an increase beyond RL 530 m to accommodate the entire PEA tailings production is not considered feasible; therefore, no further analysis of this option was conducted.

The evaluation of alternatives for a new TSF required for the PEA scenario must consider site limitations, client recommendations, and applicable technologies. The technologies selected are not exclusive but are chosen based on their adaptability to site conditions and their ability to mitigate risks.

The main criteria considered for adopting solutions for a new tailings facility should include the following aspects:

- Minimize the overall footprint.
- Increase the in-situ density of the tailings.
- Be compatible with complex site-specific geological and geotechnical conditions.
- Be adequate for the high seismicity of the area.
- Minimize the requirements for borrow material.
- Reduce the amount of water released from the tailings impoundment.
- Keep free water away from the embankments where possible.
- Promote unsaturated conditions in the tailings.
- Reduce free water within the impoundment.
- Ensure long-term sustainability and minimize overall risks.
- Provide suitable conditions for progressive reclamation.

A combined approach has been recommended to maximize the use of the existing TSF and delay the operational need for a new TSF. The preferred option for the new TSF, based on a high-level desktop study, is to construct a conventional TSF with a rockfill dam embankment and subaerial discharge of tailings. This design includes a dam with an 18 m height, which provides sufficient volume capacity to accommodate the tailings forecasted for the PEA production scenario. K92 prefers rockfill construction due to the availability of suitable materials once the mine shifts to a new material handling strategy. This strategy involves the use of paste backfill and the stoping method to fill underground voids.

The need to improve the understanding of the foundation conditions at the potential new TSF site has been highlighted as an important consideration, especially given the geological and geotechnical complexities typical of PNG. Over recent years, a significant increase in knowledge and understanding about the characteristics of the tailings, however, there are possibilities for variations from the Kora Expansion in the coming years. Ongoing testing and data collation of tailings characteristics will be beneficial to achieve desired settled densities and validate the suitability of mechanical dewatering solutions.

Figure 24-2 illustrates the TSF embankment raises for the existing TSF and the embankment construction sequence for the proposed new TSF. Figure 24-3 shows the location of the new TSF preferred option alongside the corresponding existing infrastructure.

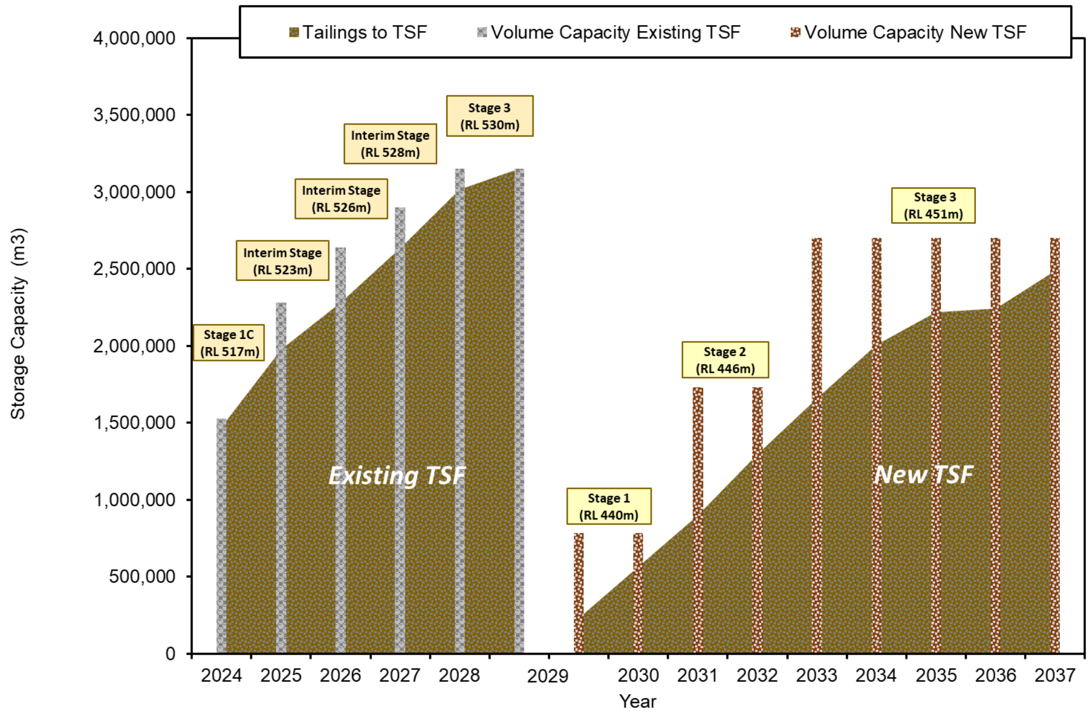


Figure 24-2 TSF Embankment Raises for PEA Production

Figure 24-3New TSF Preferred Option

The estimated total cost to manage tailings for the Kora Expansion PEA scenario, including the expansion of the existing TSF and the construction and operation of the new TSF with a rockfill dam is approximately **USD**

$74.5 million. It is advisable to conduct a comprehensive options study to explore alternative solutions. These alternatives could potentially reduce the capital cost for the implementation of a new TSF, decrease the total volume required, facilitate progressive reclamation, and explore the possibility of repurposing tailings as raw materials for products such as concrete, glass, construction materials, and tiles. Moreover, engaging local communities and businesses in these processes can enhance social perception, which is highly valued by regulators and shareholders.

24.7 Kora 2024 PEA Economic Analysis

24.7.1 Introduction

An economic analysis for the PEA has been carried out for the expanded 1.8M tpa project using a cash flow model, similar to that carried out for the 1.2M tpa project.

24.7.2 Model Inputs and Assumptions

The key model inputs used in the economic analysis for the PEA are summarized in Table 24-12.

Table 24-12 Model Inputs

Model Inputs	Source	Unit / Value
Base Currency		USD
Base Date		1st Quarter 2024
Ore Processed over LOM	Entech	20.4 Mt
Metal Prices		
Gold	K92ML	USD1,900 / oz (fixed)
Copper	K92ML	USD4.50 / lb (fixed)
Silver	K92ML	USD25 / oz (fixed)
Recoveries		
Gold (total)	MMS	92.6%
Gold (gravity to doré)	MMS	15.0%
Copper	MMS	94.2%
Silver	MMS	78.0%
Concentrate copper grade	MMS	22.5%
Processing Plant Capacity (dry tonnes of ore)		
Existing Plant	K92ML	600,000 tpa
New Plant	GRES	1,200,000 tpa
Royalties (deducted from net revenue)	K92ML	2.0%
Levies (deducted from gross revenue)	K92ML	0.5%
Tax Rate	K92ML	30%
Depreciation		Not considered
NPV Discount Rate	K92ML	5%

Ore Processed

Total ore processed from 2024 to 2037 is 20.4 Mt.

Average head grade over LOM is:

- Gold 6.0 g/t.
- Silver 23.1 g/t.
- Copper 1.2%.

Gross Revenue

Gross revenue from doré and concentrate (excluding treatment and refining charges and operating costs) is USD $9,168M over the LOM, or USD $449.5/t ore.

Treatment and Refining Charges

Total treatment and refining charges, including penalty element deductions equates to USD $11.6 /t ore.

Operating Costs

Annual fixed and variable costs for the PEA are included in the cash flow and summarized in Table 24-13.

Table 24-13 Operating Costs

Area	Source	USD / Value
Mining (average over LOM)	Entech	$62.6 / t ore
Processing Plant - Existing - Total (average over LOM)	GRES	$19.8 / t ore
Processing Plant - New - Fixed	GRES	$9.9M /year
Processing Plant - New - Variable	GRES	$5.36 / t ore
Processing Plant - New - Total (average over LOM)	GRES	$17.5 / t ore
General & Admin - Total (average over LOM)	K92ML	$25.5 / t ore
Paste Plant - Fixed	GRES	$2.1M /year
Past Plant - Variable	GRES	$23.2 / m^3 paste
Paste Plant - Total (average over LOM)	GRES	$28.09 / m^3 paste
TSF (average over LOM)	ATC Williams	$0.48 / t ore
Transport & Insurance (average over LOM)	K92ML	$10.24 / t ore
Total (average over LOM)		**$126.18 / t ore**

Capital Costs

Pre-production capital expenditures for the PEA are defined in Table 24-14. Sustaining capital for the Plant, Mining and TSF expansion costs have been phased over the life of the project and detailed in Table 24-15. (This summary only include capital cost from 2024 onwards.)

Table 24-14 Pre-Production Capital Expenditure

Item	Source	Total (USD $M)
Mine		Refer sustaining capital
Process Plant	GRES	$90.3
Power	GRES	$19.2
Camp Upgrade	K92ML	$6.2
Haul Road Bridges	K92ML	$14.4
Warehouse/Maintenance Facilities	K92ML	$5.8
Owner's Costs	K92ML	$20.1
Paste Plant	GRES	$45.1
Total		**$200.9**

Table 24-15 Sustaining Capital Expenditure

Item	Source	Total (USD $M)
Mine	Entech	$727.1
TSF	ATC Williams	$57.6
TSF Closure Costs	ATC Williams	$21.0
Processing Plant	K92ML	$5.4
Surface Fleet	K92ML	$48.6
Site Services	K92ML	$61.9
Total		**$900.5**

Royalties

A royalty of 2.0% has been deducted from gross revenue.

A levy of 0.5% has been deducted from gross revenue.

Depreciation

Depreciation has not been considered in the cash flow model.

Inflation

Inflation has not been included in the cash flow analysis. All costs are input at nominally Q1 2024 values.

24.7.3 Financial Model Results

The financial model indicates that the PEA project has a post-tax Net Present Value (NPV) of USD $2,295M at a discount rate of 5%.

The PEA is preliminary in nature and 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, and there is no certainty that the PEA will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Figure 24-5 shows the post-tax annual and cumulative cash flow for the project over the LOM.

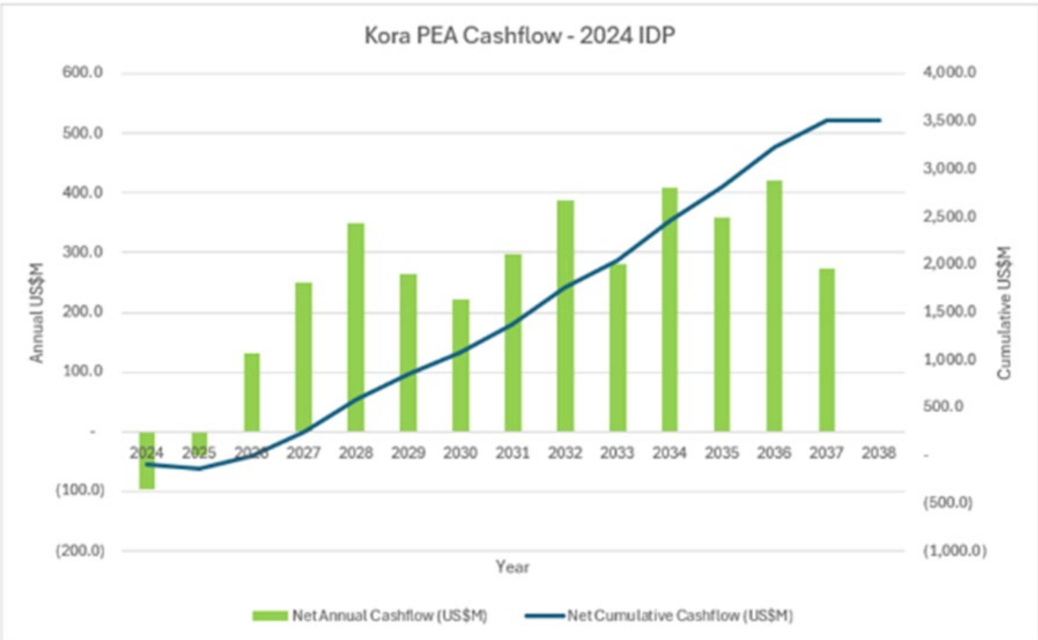


Figure 24-4 PEA Cumulative Cash Flow.

Table 24-16 PEA Cash Flow Model Summary

	Source	Units	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2037	TOTAL
Period Ending			31/12/2024	31/12/2025	31/12/2026	31/12/2027	31/12/2028	31/12/2029	31/12/2030	31/12/2031	31/12/2032	31/12/2033	31/12/2034	31/12/2035	31/12/2036	31/12/2037	
Gross Revenue - After Treatment, Refining Costs & Penalties		**US$ 000's**	**214,069**	**317,148**	**417,183**	**627,524**	**788,597**	**675,314**	**609,649**	**711,767**	**835,616**	**673,701**	**877,279**	**800,497**	**868,362**	**490,319**	**8,907,027**
Total Operating Costs		**US$ 000's**	**94,767**	**123,343**	**158,057**	**182,252**	**215,215**	**216,449**	**218,692**	**216,942**	**216,192**	**219,856**	**216,129**	**211,147**	**203,926**	**79,324**	**2,572,290**
Total Royalties & Levies		**US$ 000's**	**5,253**	**7,781**	**10,303**	**15,497**	**19,421**	**16,580**	**14,895**	**17,469**	**20,616**	**16,562**	**21,665**	**19,713**	**21,397**	**12,104**	**219,254**
Total Capital Costs		**US$ 000's**	**185,608**	**190,271**	**69,059**	**83,124**	**68,152**	**75,125**	**69,939**	**63,919**	**58,226**	**50,344**	**59,934**	**58,791**	**53,000**	**36,867**	**1,122,358**
FINANCIAL SUMMARY																	
Total Revenue		US$ 000's	214,069	317,148	417,183	627,524	788,597	675,314	609,649	711,767	835,616	673,701	877,279	800,497	868,362	490,319	8,907,027
Total Outgoings		US$ 000's	285,627	321,396	237,419	280,872	302,789	308,154	303,525	298,329	295,033	286,762	297,727	289,652	278,323	128,294	3,913,903
Earnings before Interest, Tax, Depreciation and Amortisation (EBITDA)		**US$ 000's**	**-71,558**	**-4,247**	**179,765**	**346,652**	**485,808**	**367,160**	**306,124**	**413,438**	**540,583**	**386,938**	**579,552**	**510,845**	**590,040**	**362,025**	**4,993,124**
Taxable Profit / Loss		US$ 000's	(71,558)	(4,247)	179,765	346,652	485,808	367,160	306,124	413,438	540,583	386,938	579,552	510,845	590,040	362,025	4,993,124
Tax Payable		US$ 000's	24,318	35,754	49,415	97,466	135,325	103,600	83,668	115,496	152,677	106,077	170,771	151,474	169,912	89,383	1,485,337
Net Profit After Tax		US$ 000's	(95,876)	(40,002)	130,350	249,185	350,483	263,560	222,457	297,941	387,907	280,861	408,781	359,371	420,128	272,642	3,507,787
Cash Flow																	
Cash Flow - After Tax		000's US$	(95,876)	(40,002)	130,350	249,185	350,483	263,560	222,457	297,941	387,907	280,861	408,781	359,371	420,128	272,642	3,507,787
Accumulated Cash Flow		000's US$	(95,876)	(135,878)	(5,528)	243,657	594,140	857,700	1,080,156	1,378,098	1,766,004	2,046,865	2,455,646	2,815,017	3,235,145	3,507,787	
Payback Period - Simple		years	1.00	1.00	1.00	0.02	-	-	-	-	-	-	-	-	-	-	3.02
Discounted Cash Flow - Monthly Discounting																	
Discounted Cash Flow @ 5%	K92 Calc	US$ 000's	(93,600)	(38,200)	114,400	209,100	281,200	201,500	161,800	205,200	256,800	175,900	244,200	205,000	228,700	142,700	2,294,700
Discounted Cash Flow @ 8%	K92 Calc	US$ 000's	(92,400)	(37,200)	106,100	189,000	247,700	172,600	134,600	165,500	202,500	134,300	181,400	148,300	161,000	98,200	1,811,600
Discounted Cash Flow @ 10%	K92 Calc	US$ 000's	(91,600)	(36,600)	101,100	177,000	228,000	156,000	119,400	143,800	173,400	112,600	149,500	120,100	128,100	77,000	1,557,800
Discount Rate		%	**5.0%**	**8.0%**	**10.0%**												
Net Present Value (NPV) - After Tax		000's US$	**2,294,700**	**1,811,600**	**1,557,800**												

25. INTERPRETATION AND CONCLUSIONS

25.1 Mineral Resource Estimate

The effective date of the MRE for the Kora lodes is the 12th of September 2023, which was the date that the latest database was received by H&SC.

The MRE for the Kora and Judd deposits were prepared using Ordinary Kriging (OK) in the H&SC in-house GS3 modelling software package. H&SC considers OK to be an appropriate estimation technique for the type of mineralization, its extent and the nature of the available data. The resource estimation includes some internal low-grade material. The drillhole data and resulting GS3 models were loaded into the commercially available Surpac mining software for geological interpretation, composite selection, block model creation and validation, resource estimate reporting and to facilitate any transition to future mining studies.

The GS3 modelling software was developed by Neil Schofield (ex-Stanford University) and has capacity for both Multiple Indicator Kriging (MIK) and OK modelling techniques.

The approach to resource estimation for the Kora and Judd deposits is relatively straightforward. A 3D geological interpretation of mineral domains as wireframes for the K1, K2, Kora Link and Judd lodes was completed using the Surpac software. These wireframes were then used to select 1 m 'best fit' data composites for gold, copper and silver from samples in the drillhole database which were then subject to data analysis including aspects of spatial distribution (variography). OK modelling was used with up to four search domains for each lode, based on subtle variations in dip and strike of the lodes with the resulting 3D models loaded into a Surpac block model. The same search parameters were used for all the lodes. Post-modelling processing, including block model validation and reconciliation, was undertaken in Surpac. The newly generated MRE were classified taking into account a number of factors including sample spacing and distribution, variography, geological understanding, QAQC procedures and outcomes, density data, core recovery and reconciliation with mill feed.

The recent drilling has confirmed the geological interpretation of mineral domains for Kora with only very minor modifications to the previous 2021 domain designs. The main changes are extensions to K1 and K2 in the Kora South area. The Kora Link lode has been incrementally expanded around its margins.

The Judd lode system has been expanded to comprise four lodes, J1, J2, J3 and J1W. J1 has been substantially expanded without too many surprises, J2 sits in the J1 footwall but is markedly thinner and of lower grade. J3 sits in the footwall of J2 and is also relatively thin and of low grade. J1W lies in the hanging wall to J1 and is fairly limited in spatial extent, mainly due to a lack of drilling but has average grades similar to J1 that suggest some common link. The multiple lenses are partly a function of the previously reported "break down" in the apparent geological continuity at the southern end of J1, which has partly been resolved by the new drilling.

The MRE are reported for a 3 g/t gold equivalent cut off, a change from the 1.75 g/t Au cut off used in 2021/2, which is based on mining studies completed by K92ML. The amount of Measured and Indicated Resource for Kora has slightly decreased but has a slightly higher gold grade indicating the same amount of gold ounces. Gold equivalent ounces have gone slightly up mainly due to an increase in the copper and silver grades which are a function of the increased amounts of the more copper-rich K2 lode. The recent drilling has essentially replaced mining depletion. However, the Kora Inferred Resource has significantly increased in size albeit with lower gold grades that are partially offset by higher copper grades from the more copper-rich Kora South area to result in only a slightly lower average gold equivalent grade but significantly more gold equivalent ounces. The Judd Measured and Indicated Resource has been trebled in size, in spite of mining depletion, but with a 20% drop in gold (and silver) grade leading to a still substantial doubling of gold equivalent ounces. The Judd Inferred Resource has almost doubled in size with a 17% drop in gold grade resulting in a 73% increase in gold equivalent ounces.

The global grade tonnage curves indicate a consistent shape to the curves with some improvement in smoothness of the gold grade curves. The MRE are in line with H&SC's expectations based on a combination of K92ML drilling strategy and the generally lower grades associated with the recent infill drilling over the past two years. Further consideration on the use of appropriate top cuts is required, with a possible revision for K2 needed.

There is an issue with the density measurements from the Kora South upper zone that needs addressing.

The MRE reported in this section have been classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves.

25.2 Mining and Minerals Reserve Estimate

The total Mineral Reserve for the Kainantu Project is shown in Table 25-1. The Mineral Reserve estimate is based on the Global Kora and Judd Mineral Resource estimate (September 12, 2023 effective date - refer to Table 25-1 below), net of post-resource mining depletion from September 12, 2023 to December 31, 2023, of 183,768 tonnes at 8.05 g/t Au, 0.92% Cu and 15.37 g/t Ag

Table 25-1 Total Mineral Reserve for the Kainantu Project

Kora and Judd Deposit Reserve Summary (January/2024)										
	Tonnes	Gold		Silver		Copper		Gold Equivalent		
	mt	g/t	moz	g/t	moz	%	kt	g/t	moz	
Kora Deposit										
Proven	2.95	7.4	0.70	19	1.9	1.1	31	9.4	0.89	
Probable	2.52	5.7	0.46	19	1.6	1.0	26	7.6	0.61	
Proven & Probable	**5.47**	**6.6**	**1.16**	**19**	**3.4**	**1.1**	**57**	**8.6**	**1.50**	
Judd Deposit										
Proven	0.24	8.3	0.06	17	0.1	0.6	1	9.4	0.07	
Probable	0.47	6.5	0.10	13	0.2	0.5	2	7.5	0.11	
Proven & Probable	**0.71**	**7.1**	**0.16**	**14**	**0.3**	**0.5**	**4**	**8.1**	**0.18**	
Consolidated										
Total Proven	3.19	7.5	0.77	19	2.0	1.0	33	9.4	0.96	
Total Probable	2.99	5.8	0.56	18	1.8	1.0	28	7.6	0.73	
Total Proven & Probable	**6.18**	**6.7**	**1.32**	**19**	**3.7**	**1.0**	**61**	**8.5**	**1.69**	

25.3 New 1.2M tpa Treatment Plant

The new 1.2M tpa processing plant design flowsheet incorporates a conventional single stage jaw crushing (200 tphr) reporting to a crushed ore overflow surge bin and dead stockpile providing 12 hours of stockpile capacity. The primary crushed ore supplies a SAB milling circuit (150 tphr) that includes an open circuit SAG mill and closed circuit ball mill. The ball mill product reports to hydrocyclones, with cyclone overflow reporting to the flotation circuit and the cyclone underflow stream being split between the gravity circuit, flash flotation circuit and ball mill for grinding.

The gravity circuit involves one batch centrifugal concentrator followed by two stages of gravity separation using shaking tables to upgrade the gravity concentrate, which is then calcined and smelted to produce gold doré. The flotation circuit includes flash flotation and conventional sulphide flotation, followed by thickening and filtering to produce a gold-copper-silver concentrate. The circuit is based on simple conventional technology with the flowsheet largely similar but optimized from the existing Kainantu processing plant. The key difference between the existing plant and the proposed Stage 3 Expansion process plant is the implementation of a one stage crushing circuit (vs two stage crushing) and two stages of milling with a SAG and ball milling (vs one stage of ball milling).

25.4 Paste Fill Plant

Backfill systems have been designed to satisfy both the DFS and PEA requirements. The design criteria for the backfill system will satisfy paste backfill requirements of 108 to 143 m^3/hr. The system will produce pressure filtered mill tailings, transport them using surface truck haulage of the cake to the underground plant, and underground mixing of backfill to service mine operations.

The backfill system design will require significant haulage considerations and special consideration for the movement and storage of binder to the underground plant location. The bus stop area design has been prepared to address those issues. Therefore, these challenges have been solved and the system is ready to be advanced to the detailed engineering stage.

Underground distribution system flow modelling has confirmed that the underground backfill production system will be ideally situated within the centre of the orebody. This location will minimize the amount of binder needed to produce the backfill and limit pumping requirements to distribute the backfill throughout the ore body.

25.5 Tailings Storage Facility

Tailings generated from the production expansion will be stored in the existing TSF and the underground mine stopes.

A tailings consultant has been engaged to review the current TSF conditions and to provide the necessary assistance. The adopted tailings management solutions will follow local and international regulations and guidelines, including:

- Australian National Committee on Large Dams (ANCOLD, 2019) - the recommended body for Australian Tailings Management Standards for Planning, Design, Construction, Operational and Closure.
- Global Industry Standards on Tailings Management (GISTM, 2020) - aims to strengthen the mining industry's current practices by integrating social, environmental, and technical considerations.

26. RECOMMENDATIONS

26.1 Mineral Resource & Exploration

- Continue infill drilling to upgrade both the Kora and Judd Mineral Resources.
- Drilling to the south of both Judd and Kora, following up significant isolated drill intercepts of mineralization, with a view to expanding the current Mineral Resources.
- Continue reconnaissance drilling of other potential mineral lodes/systems identified from surface mapping and sampling within the general Kora Consolidated and Judd lode areas.
- A review is required for the density data currently in the drill hole database.
- Revisit the Kora Fault interpretation with an attempt to upgrade the quality of interpretation. Look to assess areas of geological complexity; e.g. the possible 'doubled up oblique fault section' in the middle of the deposit.
- Review grade control procedures to continue to improve model reconciliation.
- Consideration of a more sophisticated modelling technique to help with the grade control e.g. E-type models from Multiple Indicator Kriging.
- Near mine targets are considered to be the most important initial targets, as they are the most likely to be able to generate additional resources that can become mill feed in the short and longer term. The Kora - Judd - A1 corridor and adjacent veins should be the highest priority, in this regard.
- Maniape and Arakompa, are the next priority, both have historic resources, in proximity to plant and camp infrastructure.
- Surface work should include detailed assessment of lithocaps and vein expressions, as well as geophysical anomalies, assisted by the utilization of spectral data.
- For regional reconnaissance, blanket geochemistry is recommended, followed by trenching where structural information is sought, especially in vein corridors. All in conjunction with detailed mapping of lithological contacts, alteration and structure.
- K92ML has enjoyed consistent success from its exploration drill programs, both underground and at surface. Therefore, drilling should continue to be the ultimate goal for all of its project evaluations, with an objective to impressively expand the resource inventory, especially in the near mine environment. At the same time, it is important to maintain a project pipeline, such that there are active projects at all major levels of execution (i.e. grassroots through to brown fields and resource definition), so that options are always available for drill targeting.
- In summary, the exploration strategy should involve a prioritized approach focusing on extending known trends, exploring high-potential areas with historic resources, tailoring work programs to the specific characteristics of each tenement, and maintaining a strong emphasis on drilling as the primary evaluation method.

26.2 Mining

- Optimization studies to increase NPV through scheduling and design improvements.
- Optimization of ore pass locations, and potential use of truck chutes at their termination points as opposed to rehandle with a loader.
- Continue to develop the mine with setting up required infrastructure i.e. air, water, power and ventilation.
- Prioritize decline and incline developments to link upper part (access from Portal 1) to lower part (access from Portal 2/3) of the mine together so multiple accesses to entire mine can be established.
- A dedicated geotechnical drilling program designed to cover critical infrastructure locations and the Kora orebody.
- Ongoing campaign geotechnical logging of resource definition drill holes, with focus on any holes intercepting mineralised zones Judd, K1 and K2 and the FGZ.
- Additional geotechnical testwork is required for all lithologies to confirm DFS assumptions for operational levels of robustness.
- A stress measurement program utilising the either WASM AE (Acoustic Emission) method or Hollow Inclusion Cell (HI-Cell) method is advised to be commissioned from the Kora mining area to assess the in-situ major principal stress magnitudes and directions at the Kora deposit.

- Detailed geotechnical mapping of ore drives should be routinely undertaken for input into stope design and determining a more accurate characterization of the rock mass.
- Numerical modelling of the stope extraction sequence needs to be refined once final stope designs are determined and re-run using the stress measurement test results to assess mining-induced stress re-distribution in terms of magnitudes and directions.
- A full review of the ground support systems should be undertaken as mining progresses to include updated geotechnical data to improve and justify the standards recommended.

26.3 1.2M tpa Treatment Plant

- Complete the construction of the new 1.2M tpa treatment plant.

26.4 Tailings Storage Facility

The recommendations to manage the tailings produced due to the Kora Expansion DFS scenario include:

- The deposition of thickened tailings into the existing TSF with an embankment raised to 525mRL and managing the facility by adopting Best Available Practices.
- Validate suitability of foundations.
- Implement actions to maximize the settled density of tailings.
- Consider the provision of additional drainage to promote consolidation of tailings.
- Provide sustained discharge of water from tailings, to minimize stored water within the TSF.
- Management of water pond size and location, such that beaches are maintained against the embankment.

26.5 Paste Plant

The filter plant has the potential to be a bottleneck for backfill operations. Pressure filters are known to require significant maintenance with lower availability compared to other filtration methods. K92ML would ensure that sufficient operational spares for the filters are always stocked on site to deal with all maintenance eventualities and the vendors should be tasked with providing a list of best practices for operations.

K92ML should implement a geotechnical program at the proposed "bus stop" area to determine final foundation requirements for the support of the tailings stockpile, wash bay and binder silo installations. In addition, enhanced fleet maintenance planning will be required to allow for the additional trucking requirements for hauling binder to site and servicing the extra wear that the haul fleet will experience due to the back haul of filter cake tailings from the mill to the bus stop. Also, the extra vehicles required to haul cake from the bus stop to the paste plant will introduce additional maintenance capacity requirements which will need to be accounted for in shop design planning.

The back haul of filter cake tailings using the ore haul fleet has the potential to introduce over-sized material into the backfill production stream. It will be necessary for K92ML to ensure that all truck operators are well trained on the use of the wash bay facilities. Proper procedure and training programs will help to prevent excessive fouling of the backfill stream and protect the piston diaphragm pumps from potential damage caused by oversized material entering the system.

Slag and cement supply streams will form a significant portion of the operational costs of running the backfill system. Volatility in the price of these commodities will therefore have a significant impact on the cost per tonne of backfill for K92ML. Kainantu staff should remain vigilant in looking for opportunities to find alternative sources for these inputs in case there is ever a disruption, or price increase from currently suppliers.

Locating the paste plant underground will involve different maintenance and operational challenges. K92ML must be prepared to service the maintenance needs of a remote plant. The paste plant will require a level of care and maintenance similar to the mill. Additionally, the operational challenges presented by pumping backfill up into the mine and the handling and stockpiling of filter cake will require proper training and management of operational procedures for upset and abnormal conditions. Operators should be trained on the intricacies of this installation configuration and be tested regularly for upset condition response preparedness.

26.6 Environmental Studies, Permitting and Social or Community Impact

- Geochemical characterisation of the ore body and waste rock should be reviewed, and an ore/waste acid mine drainage potential block model developed. With an extensive exploration program underway, drill core sampling is a cost-effective way to sample, and feed into a predictive ore/waste model. Potential acid generating wastes can be managed at the underground source rather than retrospectively on surface.
- Timely resolution of the 2020 MOA renegotiation with the Provincial Government and Landowners, with signing of the MOA still awaiting approval from the National Executive Council of the PNG Government pending the results of the PNG Land Titles Commission appeals process. This will result in flow of royalties (held in escrow) to the landowners and provide the basis for future negotiations as the mine develops.
- The Environmental Management System (EMS) and statutory Environmental Management and Monitoring Plan (EMMP) should continue to be updated to include the project expansion aspects and impacts. Environmental program specific management plans and SOPs should include new permit requirements.
- The mine closure plan will need to be reviewed and updated to reflect material changes following the expansion.

27. REFERENCES

ALS Job number A22313, Mineralogy Report MIN5046 Part II, dated June 2021 (Judd).

ALS Job number A22313, Mineralogy Report MIN5046 Part III, dated June 2021 (K2).

ALS Rheology Report: Metallurgical Testwork conducted upon Milled Concentrate Samples for K92 Mining Limited, Report No. A23132, dated April 2022 Unpublished.

AMMTEC Report A8602, Metallurgical Testwork Conducted Upon Samples of Ore from the Kainantu Gold Project for Highlands Pacific / Ausenco Pty Ltd, Interim Draft dated September 2003.

Barrick Australia Pacific, memo dated 7 April 2009, 'Kainantu - Processing Testwork on Composites 1 and 2', and associated Excel spreadsheet results. Unpublished.

Blenkinsop, T., 2005. Structural Geology Presentation - Irumafimpa-Kora. Unpublished Consultants Presentation.

Bond, R., Dobe, J., & Fallon, M., 2009. Kora Geology & Estimate of Mineralized Inventory, Draft, April 2009. Barrick internal report.

Codes Analytical Laboratories, Job Number P1063. Gold And Tellurium Contents In Pyrite and Chalcopyrite in Three Samples for ALS, authors Paul Olin, Karsten Goemann and Leonid Danyushevsky, dated August 2021 (To determine Au and Te contents in pyrite and chalcopyrite, and F contents in mica and chlorite in three samples: K1 Ore, Judd Ore and K2 Ore (ALS MIN5046)) Unpublished.

Coote A., 2019. Petrologic Studies of Drill Core from KTDD001, KTDD006, KTDD007 & KTDD008, Blue Lake Project, Eastern Highlands, Papua New Guinea. APSAR Internal unpublished report. 60p.

Corbett, G., 2009. Comments on Au-Cu exploration project at the Oro project and environs, Papua New Guinea.

Corbett, G., 2019. Comments on the exploration potential of the Blue Lake Prospect, Bilimoia, Papua New Guinea. Internal unpublished report.

Corbett, G.J., 2005, Geology and Mineral Potential of Papua New Guinea: Ed. A Williamson & G Hancock, Papua New Guinea Department of Mining, 152p.

Corbett, G.J., and Leach, T.M., 1998, Southwest Pacific gold-copper systems: Structure, alteration and mineralization: Special Publication 6, Society of Economic Geologists, 238p.

Core Metallurgy Report No. 1243A-001 'K92 Mining Incorporated.

Microscopy of sized fractions from final concentrate and tailings generated by testwork conducted on Kainantu gold ore', author John Knights, dated May 2020.

Dobmeier, C.J., Poke, B., and Wagner, B., 2012. 1:100 000 Geological map publication series of Papua New Guinea, Sheet 7886 Minj: Port Moresby, Mineral Resources Authority.

Dow, D.B. & Plane, M.D, 1965. The geology of the Kainantu goldfields. Bur. Miner. Res. Aust. Rep., 79.

EMM Report E211081 RP1. Kora Hydrology Assessment. Dec 2021.

Entech Report 771. K92 Mining Project - Mining Section for Feasibility Study. Oct 2022.

Entech Report 842. K92 Mining Project - Kora Underground Mine Geotechnical Feasibility Study. Oct 2022.

Entech Report 924. K92 Mining Project - Mining Section for Preliminary Economic Study. Oct 2022.

Espi, J.O., Kajiwara, Y., Hawkins, M.A., and Bainbridge T., 2002. Hydrothermal alteration and Cu-Au mineralization at the Nena high sulfidation-type deposit, Frieda River, Papua New Guinea. Resource Geology 52(4), 301-313.

Expert Geophysics, 2021. Mobile MT Survey for K92 Mining. November, 2021.

Geotechnica, 2024. Analysis of Paste Fill Exposure Stability. Geotechnica-Report-24008. Prepared for Entech and K92 Mining August 2024.

Gmelig, M. 2018. K92 Noise Report. Prepared for K92 Mining Limtied, Kainantu, Eastern Highlands Province.

Gmelig, M. 2019. K92 Airborne Contaminant Report. K92 Mine. Prepared by SGC Safety for K92 Mining Limited, Kainantu, Eastern Highlands Province.

Highlands Pacific ASX Release dated 25 September 2003, announcing a successful Definitive Feasibility Study.

Fragomeni, D.,2006., Maxine Hoffman, Andrew Kelly, Simon Yu and Norman O. , 2006. Flotation Mini Pilot Plant Experience at Falconbridge Limited MPMSC Conference, Sudbury, June 6-7, 2006.

Huang Liming, and Sunil Koppalkar, 2007. Evolution and Optimization of the Gravity Recoverable Gold Test, 39th Annual Meeting of the Canadian Mineral Processors, January 23 to 25, 2007 Ottawa, Ontario, Canada.

Hydrobiology Pty Ltd. 2005. Kainantu Gold Mine Six Monthly Environment Report. Prepared for Highlands Kainantu Limited, Kainantu, Eastern Highlands Province.

IFC. 2007. Environmental, Health and Safety Guidelines of Mining.

IUCN. 2021. The IUCN Red List of Threatened Species. Version 2021-2. A WWW publication accessed on 12 October 2021 at https://www.iucnredlist.org. IUCN UK Office, Cambridge.

K92 Mining. 2018. Bilimoia Baseline Study. Compiled by K92 Mining Limited, Kainantu, Eastern Highlands Province.

K92 Mining. 2020. Environmental Management Plan. Compiled by K92 Mining Limited, Kainantu, Eastern Highlands Province.

K92 Mining. Undated. Conceptual Mine Closure Plan. Compiled by K92 Mining Limited, Kainantu, Eastern Highlands Province.

Kavalieris, I. and Bat-Erdene, K., 2019. SWIR Spectrometer Alteration of the Blue Lake Project. Plus K92 Mining internal unpublished report. 26p.

Kohler, A., Tear, S., and Woodward, A., 2022. Independent Technical Report, Mineral Resource Estimate Update, Kora and Judd Gold Deposits, Kainantu project, Papua New Guinea', effective date 20 January 2022.

Loffler, E. 1974. Explanatory Notes to the Geomorphological Map of Papua New Guinea. CSIRO Melbourne.

Lycopodium, 2022. Kainantu Project Definitive Feasibility Study. NI 43-101 Technical Report.3268-GREP-001.October 2022.

Metso Outotec Report 3305582 K92, Kora Expansion Project Thickening Report - Part B Rev ., dated 28 January 2022.

Metso Outotec Report 3305582 'K92, Kora Expansion Project Thickening Report - Part A Rev 0., dated 28 January 2022.

Metso Outotec Report 3305583 K92, Kora expansion project-MO Filtration Test Report External Rev 0., dated 2 February 2022.

Microsoft PowerPoint presentation 'Mineralogical analysis of K92 Gold Ore: Aspects Promoting Fluorine Accumulation', author Dr Jim Lem (Senior Lecturer in Mineral Processing Engineering, PNG University of Technology), dated 12 February 2020, Unpublished.

Mining Associates, 2006 . Highlands Pacific Limited Kainantu Project Resource Update October 2006 - Mining Associates Pty Ltd (MA611). Unpublished Consultants Report.

NSR. 2001. Kainantu Gold Project - Environmental Plan. Prepared by NSR Environmental Consultants Pty Ltd for Highlands Kainantu Limited, Kainantu, Eastern Highlands Province.

Page, R.E., 1976, Geochronology of igneous and metamorphic rocks in the New Guinea Highlands: Bureau Mineral Resources of Australia Bulletin 162.

Page, R.W., and McDougall, I., 1972. Ages of mineralization of gold and porphyry copper deposits in New Guinea Highlands: Economic Geology, 67, p. 1034-1048.

Panda Geoscience, 2022. Petrology of 12 samples from Papua New Guinea for K92 Mining Inc. 87p.

Quantitative Automated Mineralogical Analysis conducted on Three Float Products from K1 Ore (Final Con, Cleaner Tails and Final Tails) from the Kainantu Mine Project, Papua New Guinea for Nagrom / K92 Mining Limited. ALS Job number A22313, Mineralogy Report MIN5046 Part I, dated June 2021.

Rogerson, R., and Williamson, A., 1986, Age, petrology and mineralization associated with two Neogene intrusive types in the eastern highlands of Papua New Guinea, in The, G.H., and Paramanthan, S., eds., Fifth regional congress on geology, mineral and energy resources of Southeast Asia, 9-13 April 1984, proceedings: Bulletin of the Geological Society of Malaysia, v. 20, p. 487-502.

Rogerson, R., Hilyard, D., Francis, G. and Finlayson, E., 1987. The foreland thrust belt of Papua New Guinea, in Proceedings of the Pacific Rim Congress 1987, Gold Coast, p. 579-583. The Australasian Institute of Mining and Metallurgy, Melbourne.

Sheppard, S., and Cranfield, L.C., 2012. Geological framework and mineralization of Papua New Guinea — an update: Mineral Resources Authority, Papua New Guinea, 65p.

Small, G., A. Michelmore, and S. Grano, 2003. Size dependent gold deportment in the products of copper flotation and methods to increase gold recovery, The Journal of The South African Institute of Mining and Metallurgy, November 2003.

SMC TEST® REPORT Nagrom Tested by: ALS Metallurgy WA Perth, Western Australia, Prepared by: Matt Weier, JKTech Job No: 21001/P60, Testing Date: May 2021, Unpublished.

SRK, 2006. Kainantu Gold Mine Review. Report Prepared for Highlands Pacific Limited. SRK Project Number HPL301. December 2006. Unpublished Consultants Report.

Tear, S., and Woodward A.J., 2022. Independent Technical Report. Mineral Resource Estimate. Blue Lake Porphyry Deposit, Kainantu, Papua New Guinea. 01 August 2022.

Tingey, R.J., and Grainger, D.J., 1976. 1:250,000 Geological Map and Explanatory Notes Sheet SB/55-10. Papua New Guinea Geological Survey Explanatory Notes.

Turia, R., Kaidong, E., Malan, P., Antiko, J., Rome, G., Kadowaki, D., Ochi, A. 2019. Papua New Guinea Forest Base-Map and Atlas. Compiled by PNG Forest Authority and Japanese International Cooperation Agency, Port Moresby.

Wardell-Johnson G., A Bax, W P Staunton, J McGrath and J Eksteen, 2013. A Decade of Gravity Gold Recovery, World Gold Conference / Brisbane, Qld, 26 - 29 September 2013.

Weather Atlas. 2021. Weather Atlas. A WWW publication accessed on 11 November 2021 at http://weather-atlas.com/en/papua-new-guinea/kainantu-climate.

Williamson, A., and Hancock, G., (editors), 2005. The geology and mineral potential of Papua New Guinea: Papua New Guinea Department of Mining, 152p.

Woodward, A.J., 2015. Independent Technical Report and Resource Estimate, Kainantu Project, Papua New Guinea. 06 March 2015.

Woodward, A.J., 2015. Independent Technical Report, Resource Estimate and Summary of Mining Facilities, Kainantu Project, Papua New Guinea. 01 May 2015.

Woodward, A.J., 2016. Independent Technical Report, Resource Estimate and Summary of Mining Facilities, Kainantu Project, Papua New Guinea. 15 April 2016.

Woodward, A.J, Desoe C., and Park L.J., 2017. Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment of Irumafimpa and Kora Gold Deposits, Kainantu Project, Papua New Guinea, 02 March 2017.

Woodward, A.J, Tear S., Desoe C., and Park L.J., 2018. Independent Technical Report, Mineral Resource Estimate Update and Preliminary Economic Assessment of Kora North and Kora Gold Deposits, Kainantu Project, Papua New Guinea, 07 January 2019.

Woodward, A., Tear, S., Desoe, C., & Park, L. J. 2020. Independent Technical Report - Mineral resource estimate update and preliminary economic assessment for expansion of the Kainantu Mine to treat 1M tpa from the Kora Gold Deposit, Kainantu Project, Papua New Guinea, 02 April 2020.

28. CERTIFICATES OF QUALIFIED PERSONS

I Andrew Guy Kohler hereby state:

1. I am a full-time employee of K92 Mining Limited, employed as Chief Geologist and reside at 259 Canning Highway Perth, Western Australia. (Telephone +61 415 842510)
2. In 1986 I graduated from Curtin University of Technology Western Australia with a bachelors degree in Applied Science (Geology) and in 2004 from Edith Cowan University with a Postgraduate Certificate in Geostatistics.
3. I have over 30 years of experience in the minerals industry as a Geologist in the fields of mineral exploration, mine geology and mineral resource estimation. I have had senior exploration and mining roles with K92 Mining Limited, India Resources Limited, Ramu NiCo Management (MCC) Limited, Avocet Gold Limited, Panaustralian Resources Limited, Portman Iron Ore Ltd, Resolute Tanzania Ltd and Sons of Gwalia Limited. I have worked and conducted advanced exploration projects in Australia, Papua New Guinea, India, Malaysia, Laos and Tanzania. I am currently employed as the Chief Geologist and previously employed as the Mine Geology and Mine Exploration Manager at Kainatu Operations, PNG, for K92 Mining Limited since 2016 and prior to that as Technical Services Manager for India Resources Limited from 2014 to 2016, at the Surda Copper Mine and on Exploration Licenses in India. Also, I was employed as Chief Geologist for Ramu Nico Management (MCC) Limited 2011 to 2014 at Kurambukari In PNG. Resource Development Manager Avocet Gold Limited 2008 to 2009 in Malaysia. Mine Geology Superintendent Panaustralian Resources Limited from 2005 to 2008, at Phu Kham Copper-Gold Operation Laos. In these roles I have been responsible for mine and exploration geology, drilling, surveying, mine planning, environment (acid mine drainage), assay Laboratory QAQC and mineral resources.
4. Applicable to the Kainantu Project is my extensive experience in mineral deposits in volcanic terrains, specifically the Penjom gold mine in Malaysia and Panaustralian copper-gold mine in Laos. I have also worked on epithermal/ hydrothermal and porphyry-style mineralisation in similar environments in Papua New Guinea, Laos, Malaysia and Australia.
5. I am a member of the Australian Institute of Geoscientists (member number 6795)
6. For the purposes of the Independent Technical Report titled 'Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis. National Instrument 43-101 Technical Report, effective date 1 January 2024, of which I am a part author and responsible person, I am a Qualified Person as defined in National Instrument 43-101 (the Rule).
7. I am responsible either wholly or partly for the preparation of report Sections: 1 to 12, 23, and 25 to 27.
8. I am full-time employee of K92ML. As Chief Geologist I visited the Kainantu site in February, April, June and August of 2023 and prior to that was regularly at the mine site as the Underground Mine Geology & Mine Exploration Manager, on a roster

basis that was completed in March 2021. I regularly inspected the diamond drilling sites and core produced, core handling, sampling and the assaying processes. Also, numerous visits were made underground to all the ore drives for Kora and for Judd. During these visits detailed verification of the processes took place.

9. I have read the Rule and this technical report is prepared in compliance with it's provisions. I have read the definition of 'qualified person' set out in the Rule and certify that by reason of my education, affiliation with a professional association (as defined in the Rule) and past relevant work experience, I fulfil the requirements to be a 'qualified person' for the purposes of the Rule.
10. To the best of my knowledge, information and belief the technical report contains all scientific and technical information that is required to be disclosed in order to make this report not misleading.
11. I do have a full-time position with K92 Mining Limited and I am a small shareholder of K92 stock and hold options directly in K92 Mining Inc, which are all part of the K92 Mining Limited's employee share scheme. However, I have no direct or indirect interest in the properties which are the subject of this report and I have had no involvement with the property prior to 2016, the year I started work for K92 Mining. I have no direct or indirect interest in K92 Holdings, K92 or other companies with interests in the exploration assets thereof. Although I am an employee of K92ML, I am acting in accordance with and as independence is described by Section 1.5 of NI 43-101.
12. I will receive only normal salary, paid monthly for the preparation of this report.

Dated 16th day of October 2024.

____

Andrew Guy Kohler, BAppSc (Geol)

I Simon James Tear hereby state:

1. I am a Director and Consultant Geologist of H&S Consultants Pty Ltd, with a business address of Level 4, 46 Edward Street, Brisbane, QLD 4000, Australia.
2. I graduated from the Royal School of Mines, Imperial College, London, UK in 1983 with a BSc (Hons) degree in Mining Geology.
3. I am registered as a Professional Geologist with the Institute of Geologists of Ireland (Registration Number 17) and as a European Geologist with the European Federation of Geologists (Registration Number 26). I have worked as a geologist in the mining industry for over 40 years. I have extensive experience with a variety of different types of mineral deposits and commodities in Europe, Africa, South America, Asia and Australia. I have over 25 years' experience with the resource estimation process including 3.5 years minesite experience (open pit and underground) and have worked on feasibility studies. I have completed over 150 resource estimations on a variety of deposit types including narrow vein gold, structural gold, nickel laterite, stratabound base metal including Iron Ore and industrial minerals. I have completed over 40 reports that are in accordance with the JORC Code and Guidelines and/or NI43-101 rules.
4. My relevant experience for the purpose of this Technical Report is:

- Involvement from high level review to geological interpretation and resource estimation for over 50 gold projects worldwide including narrow vein epithermal and mesothermal gold deposits.
- Completion of geological modelling and/or resource estimates for the following narrow gold vein deposits: Cavanacaw (N.Ireland), Nbanga (Burkino Faso), Kestanalik (Turkey), Savoyardy (Kyrgyzstan), Woolgar, Barambah, Glen Eva and Koala (all Queensland).
- Due diligence / property assessment for the following narrow gold vein deposits/mines: Curraghinalt (N.Ireland), Tolukuma (PNG), Lorena, Pajingo (both Queensland), Bronzewing, Marda (both Western Australia).
- Completion of a geological interpretation and resource estimates for the Kora Vein system in 2018, 2020 and 2021/2.

5. I have visited the project's mining lease and operations on two occasions dated 21 to 23 October 2018 for 3 days and 23 to 28 June 2023 for 4 days.
6. I have read the definition of 'qualified person' set out in Section 1.1 of the National Instrument 43-101 (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 fulfill the requirements to be a qualified person for the purposes of this

Technical Report Report titled ‘Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis. National Instrument 43-101 Technical Report, effective date 1 January 2024,

7. I am responsible, either wholly or partly, for Sections 1, 14, 25 and 26 of the Technical Report.
8. I do not hold, directly or indirectly, any shares in K92ML, K92 Holdings, K92 or other companies with interests in the exploration assets thereof. I am independent of K92ML, K92 Holdings, K92, and, the Property, as independence is described by Section 1.5 of NI 43-101.
9. Prior to 2018, I had no involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and this Technical Report has been prepared in compliance with the version of NI43-101 that came into effect on 30 June 2011.
11. To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this __16th__ day of October 2024.

Signed "Simon Tear"

Signature of Qualified Person

Simon Tear

____

Print name of Qualified Person

Simon Tear PGeo, EurGeol

Director and Consulting Geologist

H&S Consultants Pty Ltd

I Evan Kirby hereby state:

1. I am a Director and Consulting Metallurgist of Metallurgical Management Services Pty Ltd, with a business address of 17 Constellation Drive, Ocean Reef, WA 6027, Australia.
2. I graduated from the University of Newcastle upon Tyne, UK in 1972 with a BSc (Hons) degree in Metallurgy, and in 1978 with a PhD.
3. I am registered as a Fellow of the Southern African Institute of Mining and Metallurgy. I have worked as a metallurgist in the mining industry for over 48 years. I have extensive experience with a variety of different types of minerals processing and extractive metallurgical plants covering a wide range of minerals and metals in Africa, South America, North America, Asia and Australia. For over 16 years I worked as a metallurgist, employed by South African mining companies. Then for 10 years I worked for Australian based engineering companies associated with the mining industry. During this time, I worked in both management and technical roles associated with operations management, feasibility studies and development projects. In 2002, I established the Australian consulting company, Metallurgical Management Services Pty Ltd and have had ongoing involvement in a diverse range of projects, working with owner's teams as a consultant, and in some cases, as a director of the companies. During my career, I have supervised laboratory and plant testwork campaigns that supported feasibility studies, provided engineering design data, and investigated processing issues. I have also taken leading roles in the design, commissioning and optimization of ore processing plants.
4. Following is my relevant experience for the purpose of this Technical Report.
5. As consulting metallurgist for Metallurgical Management Services (from 2002 to present), had technical and management responsibility in studies and projects for the following companies: Dwyka Diamonds (RSA), Aquarius Platinum, (RSA), Stillwater Mining Company (USA), Millennium Minerals Limited, Wedgetail Gold Project (Australia), Platinum Australia Limited Smokey Hills, Kalplats and Rooderand (RSA), Sylvania Resources Limited (PGM recovery from Chrome Mine Tailings)(RSA), European Metals Holdings Cinovec Lithium Project (Czech Republic), Walkabout Resources Lindi Jumbo Graphite Project (Tanzania), and Calidus Resources Warrawoona Gold Project (Australia).
6. As Mining & Metals Technology Manager for Bechtel Corporation (from 1997 to 2002), had technical and review involvement in studies, projects and plant expansions. Had leading roles in technical work associated with the following projects: Newcrest Mining Cadia Copper Gold Project (Australia), West Angelas Iron Ore Project (Australia), Pasminco Century Zinc Project (Australia), Randgold Resources Syama Gold Project (Mali), Altonorte Acid Plant Number 3 (Chile), Stillwater Mining Company Concentrator and Smelter Operations (USA), Renison Tin (Australia), Windimurra Vanadium (Australia), Aquarius Platinum (RSA).
7. I have visited the project's mining lease and operations on three occasions dated 31 March to 05 April 2017, 09 June to 12 June 2022, and 29 May 2024 to 02 June 2024

8. I have read the definition of ‘qualified person’ set out in Section 1.1 of the national Instrument 43-101 (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 fulfill the requirements to be a qualified person for the purposes of this Technical Report titled ‘Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis. National Instrument 43-101 Technical Report, effective date 1 January 2024.
9. I am responsible, either wholly or partly, for Sections 1, 13, 25 and 26 of the Technical Report.
10. I do not hold, directly or indirectly, any shares in K92ML, K92 Holdings, K92 or other companies with interests in the exploration assets thereof. I am independent of K92ML, K92 Holdings, K92, and, the Property, as independence is described by Section 1.5 of NI 43-101.
11. Prior to 2017, I had no involvement with the property that is the subject of the Technical Report.
12. I have read NI 43-101 and this Technical Report has been prepared in compliance with the version of NI43-101 that came into effect on 30 June 2011.
13. To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th day of October 2024.

Signed “Evan Kirby”

Evan Kirby FSAIMM

entech.
MINING CONSULTANTS

CERTIFICATE OF QUALIFIED PERSON

Daniel Donald
Entech Pty Ltd
8 Cook, West Perth WA 6005, Australia

To accompany the Technical Report entitled: "Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis" dated January 1, 2024.

I, Daniel Donald, B.Eng Hons (Mining) MBA FAusIMM MSME, do hereby certify that:

1. I am Principal Consultant of Entech Pty Ltd, an independent mining consultant, with an office at 8 Cook St, West Perth, Western Australia, Australia.
2. I am a graduate from the Western Australian School of Mines, Curtin University Australia in 1995 with a B.Eng Hons (Mining) and subsequent MBA in 2010. I have practised my profession continuously since 1996. My relevant experience for the purpose of the Technical Report is: Over 25 years of gold and base metals industry experience in feasibility studies, operational mine start-up, mine costing and steady state production.
3. I am a Fellow of the Australasian Institute of Mining and Metallurgy.
4. I have read the definition of "Qualified Person" set out in National Instrument 43- 101 (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 completed a site visit to the project mining lease and operation from 29 May 2024 - 31 May 2024. I was previously involved in the 2022 Feasibility Study and Preliminary Economic Assessment in a consulting role. Prior to 2021, I had no involvement with the property that is the subject of the Technical Report.
6. I am responsible, either wholly or partly, for sections 1, 15, 16, 21, 24, 25 and 26 of the Technical Report.
7. I am an independent "qualified person" within the meaning of section 1.5 of National Instrument 43-101 - Standards of Disclosure for Mineral Projects of the Canadian Securities Administrators.
8. I have read NI 43-101 and Form 43-101F1 and have prepared and read the report entitled " Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis " dated January 1, 2024, for K92 Mining Inc, in compliance with NI 43-101 and Form 43-101F1.
9. I do not hold, directly or indirectly, any shares in K92ML, K92 Holdings, K92 or other companies with interests in the exploration assets thereof. I am independent of K92ML, K92 Holdings, K92, and, the Property, as independence is described by Section 1.5 of NI 43-101.
10. That, at the effective date of this technical report January 1, 2024, to the best of my knowledge, information, and belief it contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

This 16thth day of October 2024 'Original Signed and Sealed'

Signed “Daniel Donald”

Daniel Donald Principal Consultant Entech Pty Ltd

CERTIFICATE OF QUALIFIED PERSON

I, Brendan Mulvihill, MAusIMM (CP Met), RPEQ, of Queensland, Australia, do hereby certify that:

I am currently employed as Senior Process Engineer at GR Engineering Services Limited, Building 3, Level 3, Kings Row Office Park 42 McDougall Street, Milton, Queensland, 4064, Australia.

This certificate applies to the technical report titled "Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis" and dated effective January 1, 2024 (the "Technical Report") by K92 Mining Inc. (the "Company").

I am a Chartered Professional Member of the Australasian Institute of Mining and Metallurgy (#309808) and Registered Professional Engineer of Queensland under the discipline of Metallurgy (#15189). I graduated from the La Trobe University Bendigo, Australia (B.App.Sc. Metallurgy (Hons.), in 1995. I have practiced my profession for over 25 years in the minerals industry and have experience in preliminary and feasibility studies, process optimization, process engineering design, and operation of mineral processing plants. I have been directly involved in feasibility studies and process engineering design and commissioning of base metal and precious metal extraction plants in Australian and International projects.

I have read the definition of "qualified person" set out in National Instrument 43-101 (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 fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

I have personally inspected the subject project on two occasions between August 8, 2023 and August 23, 2023;

I am responsible for Section 17 and parts of 1, 21, and 24.

I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

I have had no prior involvement with the property that is the subject of the Technical Report.

I have read NI 43-101 and the section of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101.

As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the section of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 01 January 2024

Signing Date: 16 October 2024

Signed "Brendan Mulvihill"

Brendan Mulvihill, MAusIMM (CP Met), RPEQ

Senior Process Engineer

GR Engineering Services

CERTIFICATE OF QUALIFIED PERSON ISAAC AHMED

I, Isaac Ahmed, state that:

(a) I am a Senior Process Engineer at:

WSP Canada Inc.
6925 Century Avenue
Mississauga, Ontario Canada L5N 7K2

(b) This certificate applies to the technical report titled: Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis" with an effective date of: 01 January 2024 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of University of Toronto with a Bachelor and a Master of Applied Science Degrees in Chemical Engineering and Applied Chemistry. My relevant experience after graduation and over 16 years for the purpose of the Technical Report includes the application of paste backfill in underground mining operations. Specifically, I lead the Process and Mine Infrastructure Design group in Canada that specializes in paste backfill plant design, from laboratory investigations to conceptual studies, detailed design, commissioning and operational support.

(d) I have not visited the property described in the Technical Report.

(e) I am responsible either wholly or partly for the preparation of report Sections 1, 13, 18, 24, 25 and 26 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the part of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the part of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Richmond Hill, Ontario Canada this 16th of October 2024.

Signed "Isaac Ahmed"

<u>Signature of Qualified Person with Professional Seal or Stamp</u>

Isaac Ahmed, BASc, MASc, P.Eng (ON)

I Ralph Holding hereby state that:

1. I am a Senior Principal Engineer of ATC Williams Pty Ltd, with a business address of 16-20 Edmondstone Street, Newmarket, QLD 4051, Australia.
2. I graduated from the University of Southern Queensland, Australia in 1992 with a Bachelor of Engineering degree in Civil Engineering.
3. I am registered as a Fellow of Engineers Australia, CPEng, and a Member of IEPNG (Reg Eng).
4. I have over 30 years' experience in a wide range of civil and geotechnical engineering projects servicing mining, industrial and local government clients. Speciality areas include mine tailings and mine water storages, mine waste rock facilities and mine post-closure design, and landfill design. My involvement has included from initial planning and approvals, through to detailed design and construction, as well as operational and post-closure aspects. Engineering skills include geotechnical investigations, data analysis and design, hydrology, site water balances, hydraulic structures designs, hydrogeology, dam embankment design, civil earthworks design, capping and liner system design and hydraulic performance evaluation, risk and safety in design, and project management.
5. Following is my relevant experience for the purpose of this Technical Report. Mt Rawdon Gold Project (Qld) - Tailings Storage, Kainantu Gold Project (PNG) - Tailings Management, Garden Well Project (WA) - Tailings Storage, Moolart Well Project (WA) - Tailings Storage , Rocklands Copper Project (QLD) - Tailings Storage, McPhillimays Gold Project (NSW) - Tailings Storage, Wolfram Camp (Qld) - Tailings Storage, McArthur River Mine - Tailings Management and Storage, Kagara (Qld) - Tailings Storages, Thalanga Base Metals Project (Qld) - Tailings Storage and Mine Waste Management, Bonikro Mine (Cote dIvoire) - Tailings Storage, Goondicum Project (Qld) - Tailings Storage, Pipeline Design and Water Management, Maldon Gold Project (Vic) - Tailings Storage, Okvau Gold Project (Cambodia) - Tailings Storage, Platina Resources (NSW) - Filtered Residue Facility, Tarong Power Station (Qld) - Ash Dam Management, Rishton Gold Project (Qld) - Tailings Dam, Nolans Gold Project (Qld) - Tailings Dam, Territory Iron (NT) Tailings storage options assessment, Eloise Copper Mine (Qld) - Tailings management and storage design, Mt Freda Project (Qld) - Tailings and water management study, Taronga Tin Project (NSW) - Tailings and water management concept and detailed design, Mt Carlton Project (Qld) - TSF design and water management, Black Jack Gold Mine (Qld) - Tailings management, Mt Dromedary (Qld) - Mine Waste and Water Management, Lake Cowal Gold Mine (NSW) - Mine Waste, Mine Closure, Wafi Golpu (PNG) - Mine Waste Management, Horn Island Gold Mine, (Torres Strait Qld) - Mine Rehabilitation, Croydon Gold Mine (Qld) - Mine Rehabilitation, Century Zinc Project (Qld) - Weir reconstruction, flood levee design, Palm Valley Gas Field (NT) - Brine salts disposal study, Koniambo Nickel Ash Dam (New Caledonia) - Ash management concept study, Tara Evaporation Pond (Qld) - Lined evaporation pond design, Ranger Uranium (NT) - In-pit tailings and freshwater liner separation system concept, Wilton Fairhill Project (Qld) - Flood protection bund concept.
6. I have previously visited the project's mining lease and operations during the initial TSF construction on two occasions in 2004.
7. I have read the definition of 'qualified person' set out in Section 1.1 of the national Instrument 43-101 (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 fulfill the requirements to be a qualified person for the purposes of this Technical Report titled 'Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis. National Instrument 43-101 Technical Report, effective date 1 January 2024.
8. I am responsible, either wholly or partly, for Sections 1, 18, 21, 24, 25 and 26, of the Technical Report.
9. I do not hold, directly or indirectly, any shares in K92ML, K92 Holdings, K92 or other companies with interests in the exploration assets thereof. I am independent of K92ML, K92 Holdings, K92, and, the Property, as independence is described by Section 1.5 of NI 43-101.
10. Prior to 2004, I have had no involvement with the property that is the subject of the Technical Report.

11. I have read NI 43-101 and this Technical Report has been prepared in compliance with the version of NI 43-101 that came into effect on 30 June 2011.
12. To the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 16th day of October 2024.

Signed "Ralph Holding"

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Ralph Holding, CPEng, FIEAust
Senior Principal Engineer
ATC Williams Pty Ltd

I Nicholas Currey hereby state:

1. My name is Nicholas Currey, I work with EMM Consulting Pty Limited at Level 1/87 Wickham Terrace, Spring Hill QLD 4000, Australia as an Associate Director, Strategic Advisor, ESG & Approvals (Mining)

2. This certificate applies to the Environment and Community section of the technical report titled "Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis" and effectively dated 01 January 2024.

3. I hold a BSc (Applied Science) from Swinburne University of Technology, Australia. I have 40 years' experience in environmental and social management and advisory, much of this has been in the resource sector. I have worked on more than 40 mining projects across a range of geographies. I am an Associate Member of the AusIMM, and a qualified person for the purpose of the NI-43-101 (Standards of Disclosure for Mining Studies) (the Instrument).

4. I have visited the K92 Mining Inc. project area.

5. I am a responsible person for the Environment and Community section of the technical report titled "Kainantu Gold Mine Updated Integrated Development Plan, Kainantu Project, Papua New Guinea, Definitive Feasibility Study and Preliminary Economic Analysis" and dated 01 January 2024 (the 'Technical Report').

6. I am independent of the K92 Mining Ltd project pursuant to Section 1.5 of the Instrument, and I do not hold, directly or indirectly, any shares or interests in K92 Mining Ltd, or associated companies or the property.

7. I have not had any prior involvement with the K92 Mining Inc project area or property.

8. I have read the National Instrument and Form 43-101 F1 (the Form) and the section of the study for which I am responsible for. This section has been prepared in compliance with the Instrument.

9. I am responsible, either wholly or partly, for Sections 1, 20, 25 and 26 of the Technical Report.

10. To the best of my knowledge, information and belief as at 01 January 2024, the Environment and Community section of the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 16th October 2024.

Signed "Nicholas Currey"

Nicholas Currey