Lycopodium

K92 MINING INC.

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Independent Technical Report, Kainantu Gold Mine Integrated Development Plan, Kainantu Project, Papua New Guinea

Definitive Feasibility Study

National Instrument 43-101 Technical Report

3268-GREP-001

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1.0 SUMMARY

1.1 Introduction

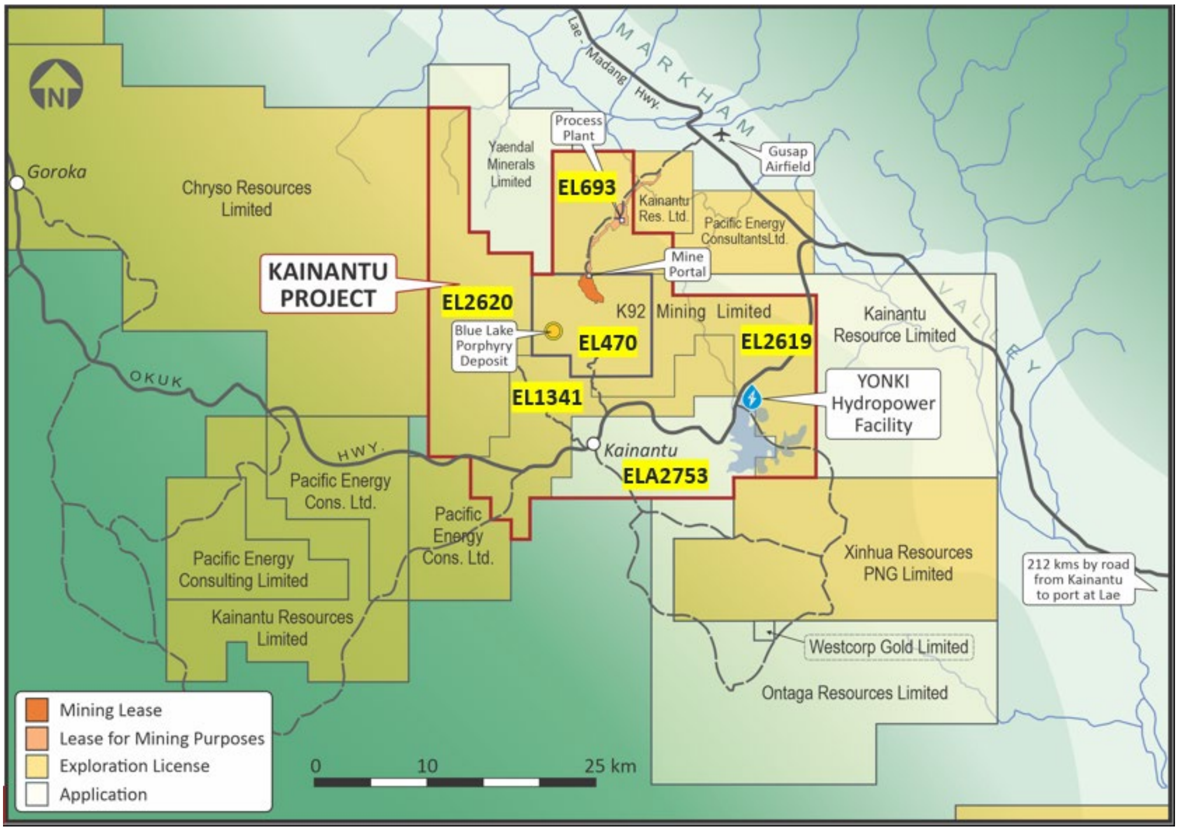
This report is an Independent Technical Report dated 1 January 2022 of the Integrated Development Plan (IDP) for K92Mining Inc.'s Kainantu Gold Mine Project (the 'Kainantu Project') in Papua New Guinea. The IDP includes the Kainantu Stage 3 Expansion Definitive Feasibility Study (DFS) Case and the alternative Kainantu Stage 4 Expansion Preliminary Economic Analysis (PEA) Case.

The IDP was independently prepared by Lycopodium Minerals Pty Ltd ('Lycopodium') of Brisbane, Australia; Entech Pty Ltd of Perth, Australia and Entech Mining Ltd of Toronto, Canada (collectively referred to as 'Entech'); ATC Williams Pty Ltd ('ATC Williams') of Brisbane, Australia; MineFill Services Pty Ltd. ('MineFill') of Newcastle, Australia; Metallurgical Management Services Pty Ltd ('MMS') of Perth, Australia, and; H & S Consultants Pty Ltd of Sydney, Australia.

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

The Property lies within an area of mostly rugged topography, with transecting rivers forming lower lying areas. Elevations range from 400m to 1900m above sea level. Vegetation is mostly primary rainforest with areas of shifting cultivation in valley floors.

Figure 1.1.1 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.

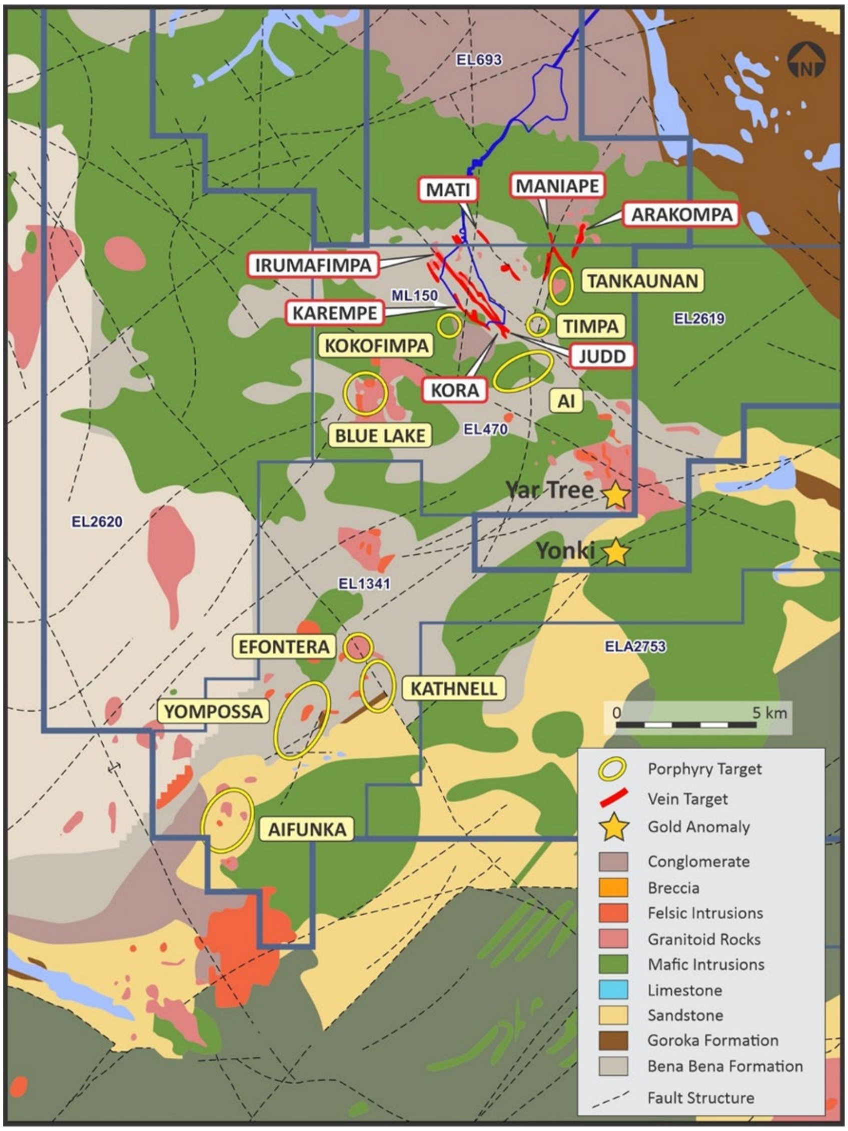
Dominant host rock of the Kora Consolidated-Irumafimpa vein systems is the highly sheared and deformed Bena Bena Formation, composed of low grade metamorphosed phyllites and amphibolites, intruded by the Elandora porphyry at the northwest end of the vein system.

Mineralization on the property includes gold, silver, and copper in epithermal Au-telluride veins (Irumafimpa) and Au-Cu-Ag sulphide veins of Intrusion Related Gold Copper ('IRGC') affinity (Kora and Judd) and also less explored porphyry Cu-Au systems (Blue Lake), and alluvial gold. The Kora Consolidated vein systems (including Kora, Eutompi and Kora North) has been demonstrated from K92ML's drilling and surface mapping results to be a continuous mineralized structure, over 1km in strike to date. This mineralized structure occurs in the centre of a large mineralization system approximately 5 km x 5 km in in which drilling has identified several individual zones of IRGC and porphyry style mineralization.

The current Mineral Resources for Kora Consolidated 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 Vein mineralized zone comprises a series of individual veins and stringer vein systems named from west to east, as K2, Kora Link and K1. The Judd vein system, which is located 90 to 150 m east of Kora Consolidated, comprises multiple veins starting with J1 as well as J2 to J4, although these latter veins are yet to be defined by drilling. The total width across the Judd vein system from J1 to J4 is between 60 to 80m. All the vein systems are composed of quartz sulphide veins that vary in width throughout the vein systems from <1 m pinch and swell structures at Irumafimpa to veins up to 10 m at Kora Consolidated. Strike continuity of the individual veins is variable.

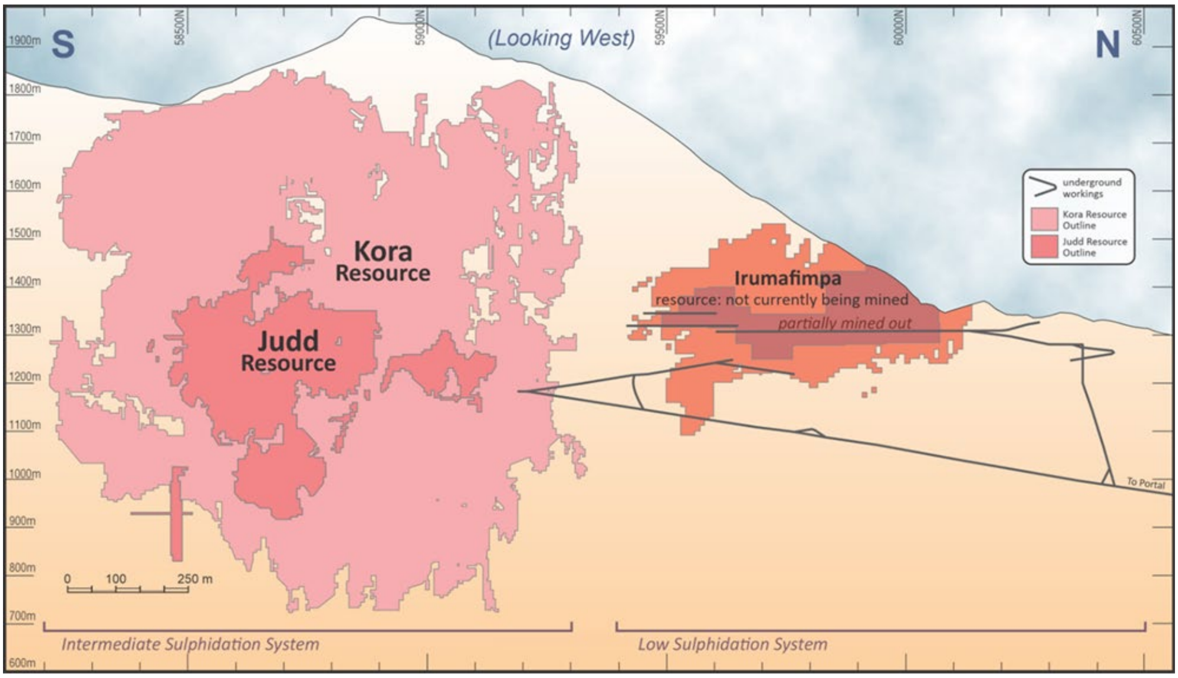
Figure 1.2.1 below shows the main vein systems and porphyry targets identified to date at the Kainantu project.

Figure 1.2.1 Kainantu Property Geology and Known Vein and Porphyry Deposits and Prospects



At Kora Consolidated and to the north along strike at Irumafimpa two stages of mineralization have been recognized. There is an early sulphide-rich copper-dominant stage overprinted by a later quartz-rich mineralization stage with high grade gold associated with tellurides. At Kora Consolidated both the sulphide-rich copper-dominant and quartz-rich Au-dominant mineralization occur along the same NW trending sub-vertical structure, tellurides are sometimes present but are insignificant and copper mineralization is in economic concentrations and generates revenue for the mine. The Kora Consolidated deposit currently comprises two parallel, steeply west dipping, N-S 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 1.2.2 Kora Judd and Irumafimpa Longitudinal Section



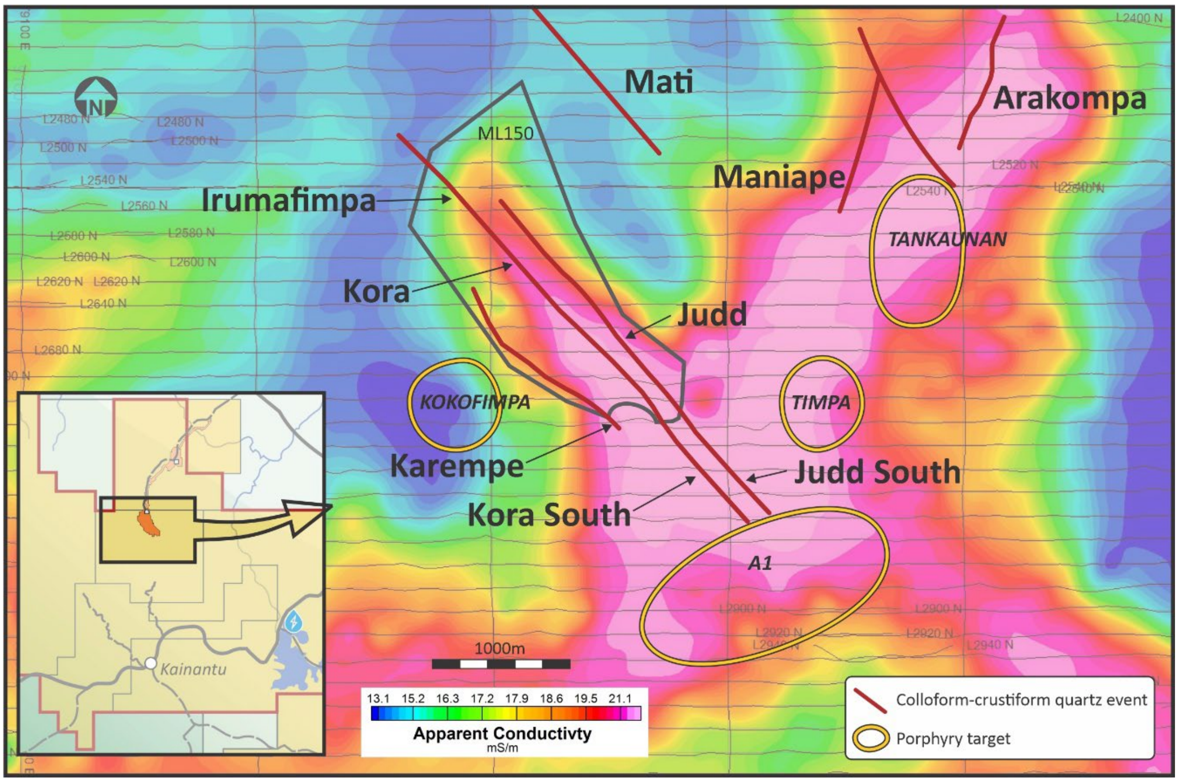
Source: K92ML (2021)

1.3 Exploration

The Kainantu Project is recognized 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. K92 Mining Ltd (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. Numerous conductive targets were identified, and, where previously drill tested, conform closely with known deposits and prospects, both vein and porphyry occurrences.

Figure 1.3.1 Judd, Kora and Irumafimpa Vein Systems with Mobile MT Contours



Source: K92ML (2021)

1.4 Minerals Processing and Metallurgical Testing

1.4.1 Process Plant

A series of tests were conducted on samples from K92 current plant operations, mining and development activities, and exploration drilling. The sequence of testwork was as follows:

- Sample receiving, preparation and preliminary compositing:
  - stream samples (final concentrate, cleaner tailings, and final tailings) from plant operations whilst processing individual ores
  - belt cuts of mined ore, K1, K2 and Judd
  - 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
  - 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 K92 plant.
- 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 the three individual ore type and the master composite samples:
  - rougher rate tests at grind size of P_{80} 0.075 mm using the same reagent suite as the operating plant to investigate required dosage rates
  - rougher rate tests at grind size of $P_{80's}$ from 0.053 to 0.150 mm to investigate primary grind requirements
  - a rougher cleaner, re-cleaner test incorporating bulk sulphide roughing followed by selective cleaning
  - rougher rate tests at a selected grind size P_{80} using four alternative reagent suites
  - rougher, cleaner re-cleaner tests using the selected reagents and with three levels of concentrate regrind.
- Locked cycle rougher, cleaner 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.

In parallel with the laboratory testwork, there was continued process development work at the operating 400 ktpa K92 concentrator. Plant optimization work was supervized by K92 management.

Results from the testwork were used in the design of the new concentrator. Recovery predictions for gold are based with consideration of existing plant operations for Quarter 4 2021, and Quarter 1 2022 in the context of testwork results. Recovery predictions for copper are based on the locked cycle testwork results in the context of plant operating data.

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 included the following:

- Characterization testing, including:
  - tailings mineralogy, particle size distribution, and specific gravity
  - process water chemical analysis
  - binder prism testing.
- Thickener testing, including:
  - dynamic testing (in a 99 mm bench scale rig) for high-rate thickener design
  - a limited amount of larger scale (190 mm test rig) for representation of a high compression thickener.
- Filtration testing, which included cloth disc filtration testing by two OEM's (Outotec and Bokela) with both yielding similar results.
- Rheology testing included bench scale vane rheometer testing of both cemented and uncemented paste material. Testing also included an assessment of changes in rheology with time, when including cement and/or slag.
- Cemented strength testing included only unconfined compressive strength testing. Mixes included in this work considered the influence of binder type, binder quantity and hydration time.

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 component.

1.5 Mineral Resource Estimates

The updated Mineral Resources (MRE) for Kora Consolidated and Judd reported in March 2022 was based on diamond drill hole samples from both surface and underground drilling along with face sampling of underground development drives. Total drilling at the time of the MRE was 504 diamond drill holes for 93,480.6m and face samples totalled 991 for 4,630.4m. The MRE was used in the DFS and PEA.

The updated Global Mineral Resource estimate (using a 1.75 g/t gold cut-off grade) for the Kora Consolidated deposit effective 11 November 2021 is presented in Table 1.5.1.

Table 1.5.1 2022 Kora Consolidated Resource Estimate

Kora									
Category	**Mt**	**Au g/t**	**Au Moz**	**Ag g/t**	**Ag Moz**	**Cu %**	**Cu Kt**	**Au_Eq g/t**	**Au_Eq Moz**
Measured	2.8	9.07	0.8	15.7	1.4	0.85	24.1	10.51	1.0
Indicated	4.4	6.68	0.9	20.2	2.8	0.97	42.4	8.35	1.2
Total M & I	**7.2**	**7.62**	**1.8**	**18.4**	**4.3**	**0.92**	**66.4**	**9.20**	**2.1**
Inferred	8.1	7.12	1.8	27.3	7.1	1.38	111.1	9.48	2.5

The Global Mineral Resource estimate (using a 1.75g/t gold cut-off grade) for the Judd deposit effective 20 January 2022 is presented in Table 1.5.2.

Table 1.5.2 2022 Judd Resource Estimate

Judd									
Category	**Mt**	**Au g/t**	**Au Moz**	**Ag g/t**	**Ag Moz**	**Cu %**	**Cu Kt**	**Au_Eq g/t**	**Au_Eq Moz**
Measured	0.22	11.26	0.08	19.9	0.14	0.72	1.59	12.56	0.09
Indicated	0.15	7.46	0.04	13.9	0.07	0.77	1.20	8.76	0.04
Total M & I	**0.38**	**9.70**	**0.12**	**17.5**	**0.21**	**0.74**	**2.79**	**11.00**	**0.13**
Inferred	1.01	4.24	0.14	11.0	0.36	0.87	8.82	5.66	0.18

Table 1.5.3 2022 Combined Kora Consolidated and Judd Resource

Kora and Judd									
Category	**Mt**	**Au g/t**	**Au Moz**	**Ag g/t**	**Ag Moz**	**Cu %**	**Cu Kt**	**Au_Eq g/t**	**Au_Eq Moz**
Measured	3.1	9.23	0.9	16.0	1.6	0.84	25.7	10.66	1.0
Indicated	4.5	6.70	1.0	20.0	2.9	0.97	43.6	8.36	1.2
Total M & I	**7.6**	**7.72**	**1.9**	**18.3**	**4.5**	**0.91**	**69.2**	**9.29**	**2.3**
Inferred	9.1	6.80	2.0	25.5	7.4	1.32	119.9	9.05	2.6

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Resources were compiled at 1.75, 2.5, 3, 4, 5, 6, 7, 8, 9 and 10 g/t gold cut-off grades for Kora and 1.75, 2.5, 3, 4, 5 for Judd.
- Density (t/m^3) is on a per zone basis, K1, K2: 2.84 t/m^3; Kora Link: 2.74 t/m^3; Judd: 2.71 t/m^3; waste: 2.67 t/m^3.
- Minimum mining width for wireframes: Kora: 5.2 m; Judd: 5.2 m.
- Reported tonnage and grade figures are rounded from raw estimates to reflect the order of accuracy of the estimate.
- Minor variations may occur during the addition of rounded numbers.
- Estimations used metric units (metres, tonnes, and g/t).
- Gold equivalents are calculated as AuEq = Au g/t + Cu%*1.607*92.8% + Ag g/t*0.0125*89%. Gold price USD1,600/oz; Silver USD20/oz; Copper USD3.75/lb. Metal payabilities and recoveries are incorporated into the AuEq formula. Recoveries of 92.8% for copper and 89% for silver.

The key to the confidence of the resource estimates is the apparent good reconciliation of the block model with the mill production 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.

1.6 Mineral Reserve Estimates

The total Mineral Reserve for the Kainantu Project is shown in Table 1.6.1. The Mineral Reserve estimate is based on the Judd Mineral Resource estimate and the Kora Mineral Resource estimate net of post-resource mining depletion from 1 November 2021 to 31 December 2021 of 36,765 tonnes at 12.94 g/t Au, 0.59 % Cu and 9.29 g/t Ag.

Table 1.6.1 Kainantu Mineral Reserve Statement (Effective Date 1 January 2022)

	Tonnes	Gold		Silver		Copper		AuEq	
	Mt	g/t	moz	g/t	moz	%	kt	g/t	moz
Kora									
Proven	2.26	7.58	0.55	14.96	1.09	0.82	18.52	9.17	0.67
Probable	3.55	5.88	0.67	19.46	2.22	0.95	33.90	7.76	0.89
Total P&P	**5.81**	**6.54**	**1.22**	**17.71**	**3.31**	**0.90**	**52.41**	**8.31**	**1.55**
Judd									
Proven	0.21	9.99	0.07	16.88	0.11	0.57	1.17	11.18	0.07
Probable	0.14	6.50	0.03	10.65	0.05	0.59	0.81	7.65	0.03
Total P&P	**0.34**	**8.60**	**0.09**	**14.40**	**0.16**	**0.58**	**1.98**	**9.77**	**0.11**
Kora and Judd									
Proven	2.46	7.78	0.62	15.12	1.20	0.80	19.69	9.34	0.74
Probable	3.69	5.90	0.70	19.13	2.27	0.94	34.70	7.75	0.92
Total P&P	**6.15**	**6.65**	**1.32**	**17.53**	**3.47**	**0.88**	**54.39**	**8.39**	**1.66**

- 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 is US$1,600/oz gold, US$4.00/lb Copper, US$20/oz Silver.
- Gold equivalents are calculated as AuEq = Au g/t + Cu % *1.7143 + Ag g/t*0.0125. 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 at waste grade. This results in a total average dilution of 20%.
- 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 3.0 g/t AuEq was used to define stoping blocks. Stope shapes above cut-off that were found to be uneconomic were excluded. The cut-off grade considers site operating costs, G&A costs, sustaining capital costs and relevant processing and revenue inputs.
- The Mineral Reserves estimates have been prepared using industry accepted methodologies and the classification of Proven and Probable Reserves conform to CIM (2014) definitions.
- Measured Mineral Resources were used to report Proven Mineral Reserves.

- Indicated 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

Entech were engaged by K92 Mining Inc. (K92) to complete a Feasibility Study (FS) 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 K92 since October 2016 and has a current processing throughput of 400,000 tpa. The FS targets an expansion to a peak processing rate of 1,200,000 tpa with a 7-year mine life. The FS assumes a start date of the life of mine schedule of 1 January 2022.

Currently Avoca and modified Avoca stoping methods, which utilize longhole stoping (LHS) with unconsolidated waste backfill, are used on site. Underground stoping and development excavations are undertaken using mechanized mining methods, with similar mechanized methods also proposed for the FS.

1.7.2 Geotechnical

A total of 3,662 m of drill core was geotechnically logged from core photographs in detail for the Kora underground mining assessment during Q1 and Q3 2021. The drillholes were selected from existing exploration holes. From this total, 1946 m, was logged as part of an initial campaign, with a second campaign of 1,716 m completed in October 2021.

In addition to the 3662 m, 472.3 m of core was validated and logged associated with the Judd orebody.

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 K92 staff and provided to Entech.

Rock property testing specific to the orebodies was unable to be sampled and sent for testing again due to covid restrictions, resourcing availability in country, and available drill holes for sampling. 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 remain a significant information gap for the project and should be addressed as soon as practical.

The drill core logging data was analysed by Entech and forms the basis of this study. This information has allowed for characterization of the rock mass, assessment of stable stoping span predictions and estimates of dilution factors

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 utilized for visualization 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 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.

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 pastefill plant, and open stoping with cemented pastefill with both bottom-up and 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 and 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, both the Kora and Judd orebodies can be classed as 'good ground'. Table 1.7.1 outlines the recommended stope spans and dilution estimates for the Kora and Judd orebody.

Table 1.7.1 Summary of Assessed Stoping Parameters for Kora and Judd

Orebody	Parameters	Hangingwall	Footwall
K1	Allowable Strike Length	16 m	20 m
	Dilution	0 - 0.5 m	0 - 0.5 m
K2	Allowable Strike Length	31 m	19 m
	Dilution	0 - 0.5 m	0 - 0.5 m
Judd	Allowable Strike Length	35 m	35 m
	Dilution	0 - 0.5 m	0 - 0.5 m

The presence of the FGZ and pervasive structure throughout the rock units will dominate stope wall behaviour with the possibility of slabbing, sliding and unravelling failure types occurring 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 will be the key to minimising stoping performance issues.

1.7.3 Mining Method Selection

A mining method selection process was completed by generating a long list of potential mining methods, which were shortlisted. The short listing comprised scenario modelling and evaluation of qualitative factors, such as minimization of tailings storage on surface. Scenario modelling was used to estimate inventories, costs, cashflows and net present values for each mining method and enable final method selection. Trade-off studies were completed on mining methods, materials handling options, production rate analysis, and ventilation strategies.

The mining method selection process resulted in Avoca and modified Avoca being selected for mining prior to commissioning of the paste fill plant in Q2 2024, and longhole stoping with pastefill for the remainder of the mine life.

1.7.4 Mine Design and Physicals

On selection of the mining method, Entech completed stope optimizations on the Mineral Resource. The FS only includes Measured and Indicated Mineral Resources.

K92 selected a 3.0 g/t AuEq stope cut-off grade to carry forward for the final mine design and FS 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.

Dilution of 0.5 m on both the footwall and hanging wall of the stope shapes (1.0 m total), was applied during the optimization phase at contained Mineral Resource grade. An additional 0.5 m of dilution at contained waste grade was applied to stope shapes that were within 2 m from the fault gouge zone. These dilution parameters were based on Entech's geotechnical study.

An additional 2.5% dilution at waste grade was applied to paste filled stoping from overmining of paste fill. An additional 5% waste rock dilution at waste grade was applied for Avoca stoping methods from overmining of waste rock fill. Paste fill and Avoca dilution are based on benchmark and site dilution data. The LOM average stope dilution is ~20%.

To account for the ore 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 subeconomic to mine.

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 development design can be found in Figure 1.7.1 below, followed by Figure 1.7.2 detailing the stope extraction methods.

Figure 1.7.1 Isometric View of the Kainantu Underground Development Layout

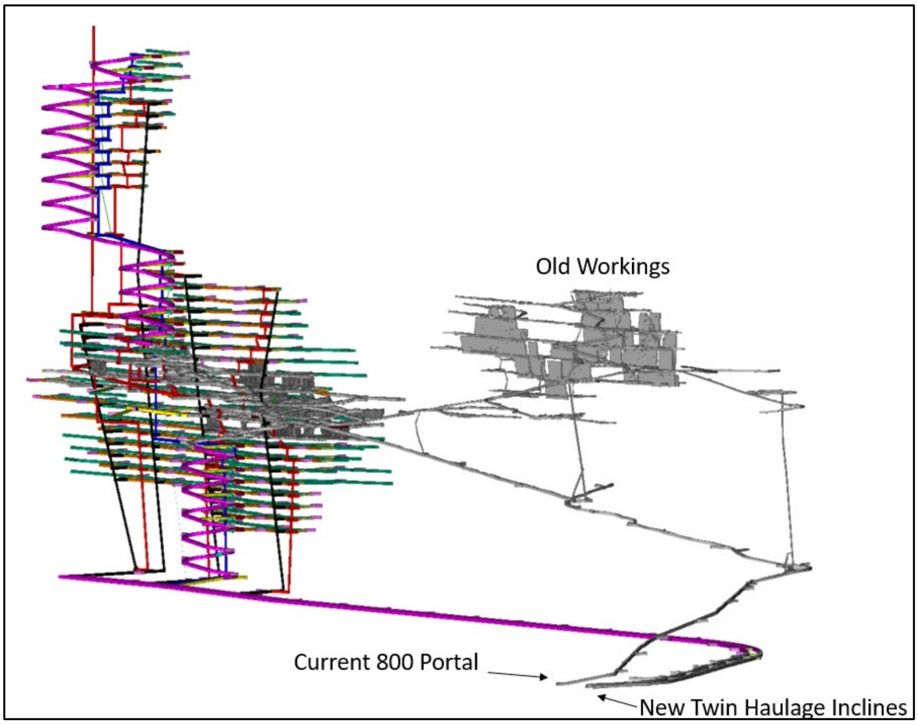
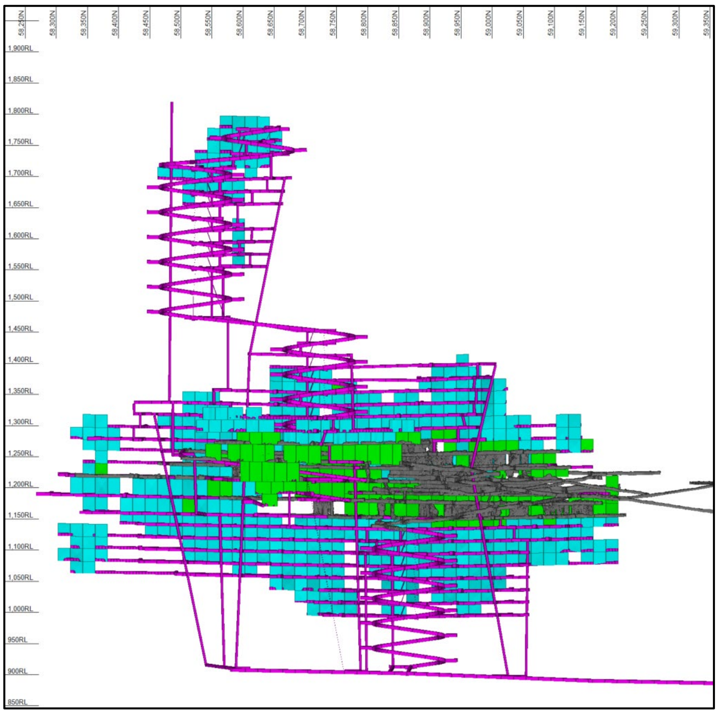
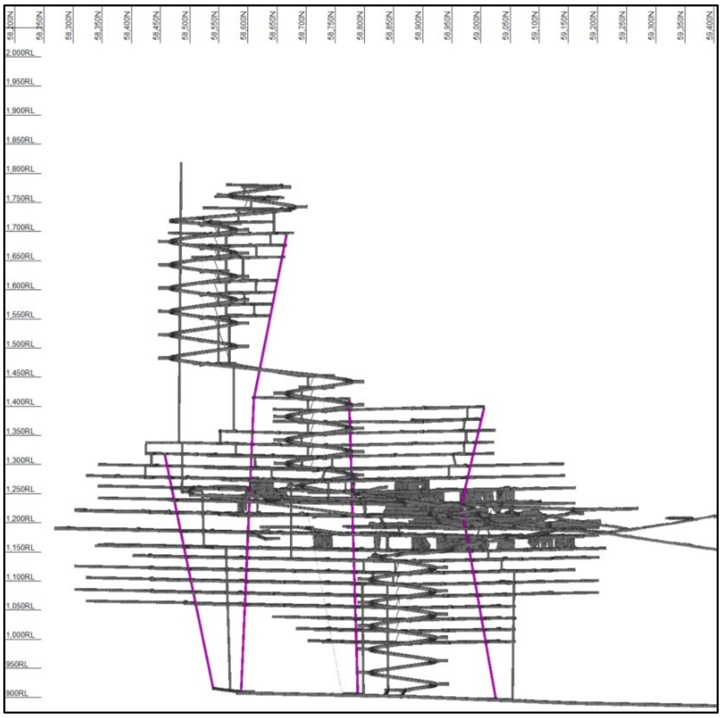


Figure 1.7.2 Looking West – LHS with Fill (Blue), Avoca (Green), As-builts (Grey)



The trade-off study quantitative and qualitative analysis resulted in the selection of the FS utilising an ore pass network to increase production rate and efficiency, while future proofing the design. The ore pass network is shown in Figure 1.7.3 below.

Figure 1.7.3 Looking West - Development Design (Ore Passes in Pink)



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 peak rate of 1,200,000 t per annum (1.2 Mtpa). The mine life is 7 years with a 3-year production ramp up (inclusive of 2022), with sustained production of 1.2 Mtpa for 3 years, and final production year of 0.7 Mtpa. A summary of the key physicals per annum is shown in Table 1.7.2.

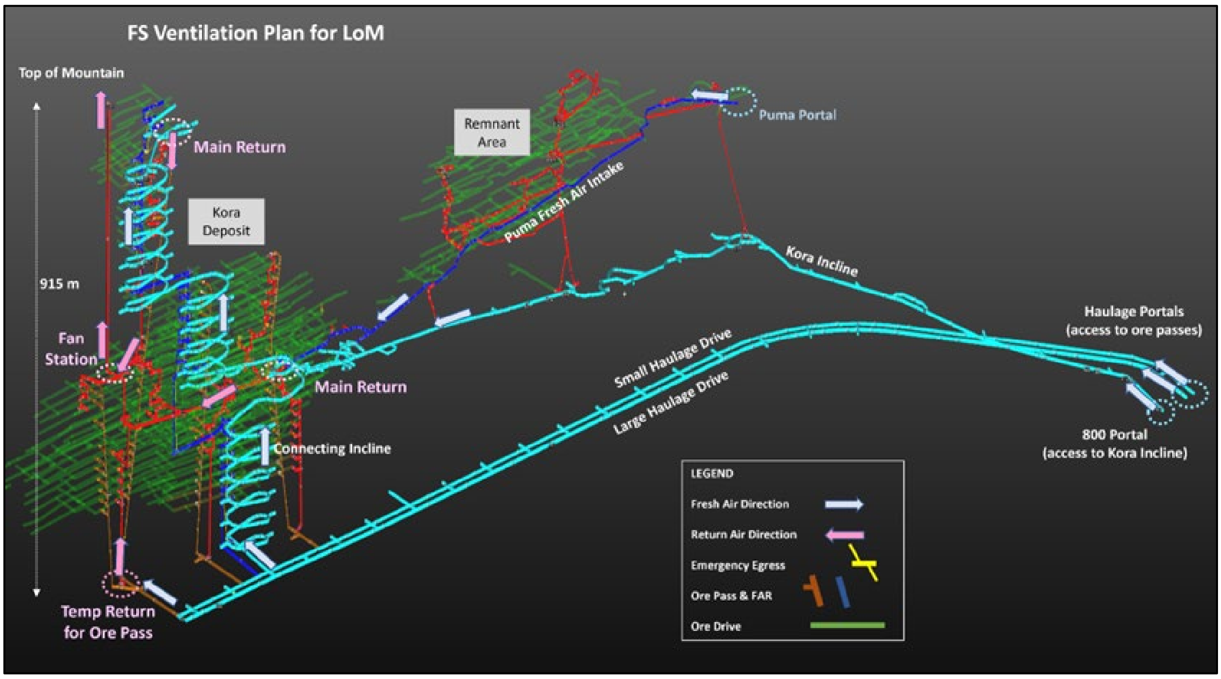
Table 1.7.2 Physicals Per Annum

		Total/Av	2022	2023	2024	2025	2026	2027	2028
Lateral Development Total	**m**	**54,427**	**9,029**	**12,028**	**10,556**	**9,237**	**9,093**	**4,134**	**350**
Vertical Development Total	**m**	**16,357**	**846**	**2,234**	**3,733**	**3,094**	**2,531**	**2,582**	**1,339**
Ore Profile									
Ore Tonnes	t	6,152,812	435,507	519,757	901,593	1,198,330	1,194,751	1,192,199	710,674
Au Grade	g/t	6.65	9.07	8.55	7.08	6.33	6.77	5.49	5.53
Au Ounces	oz	1,316,007	126,929	142,862	205,318	244,031	259,967	210,457	126,442
Cu Grade	%	0.88	0.56	0.65	0.75	0.92	1.02	0.88	1.13
Cu Tonnes	t	54,391	2,423	3,354	6,785	11,044	12,211	10,525	8,050
Ag Grade	g/t	17.53	11.47	12.94	14.40	17.99	20.03	17.84	23.04
Ag Ounces	oz	3,466,855	160,559	216,167	417,297	692,991	769,545	683,874	526,422

1.7.5 Ventilation

The FS plan to ventilate the Kainantu underground mine sees fresh air entering through four portals in the side of the mountain hosting the Kora deposit, with air exhausting out a single exhaust raise daylighting at the top of the mountain. The Puma portal will be repurposed as an intake, which is currently the main exhaust for the existing workings. Figure 1.7.4 illustrates the plan.

Figure 1.7.4 LOM Ventilation Plan Profile Facing Northwest



Fresh air enters all portals and converges at the mid-point, where production is currently occurring. From here an incline continues to the top and bottom of the deposit. Exhausting air links all production zones to a single underground fan station, located at the base of a 480 m raise to surface.

Table 1.7.3 contains a list of the mobile equipment intended for use at K92 (or equivalent unit) and shows the peak number of vehicles anticipated with the subsequent total flowrate.

Table 1.7.3 Peak Mobile Equipment Flowrate Calculation

Diesel Unit	Assumed Model	Engine Power Rating (kW)	Flowrate Requirement (m^3/s) *	Count	Total Flowrate (m^3/s)
Truck	Sandvik TH545	515	26	7	180
loader	Sandvik LH517	310	16	5	78
Charge-up	Getman	120	6	3	18
Development Drill **	Sandvik DD421-60C	110	6	6	0
Production Drill **	Sandvik DL431-7C	119	6	2	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	Normet Spraymec SF 050 D	90	5	2	9
Agitator	Normet LF 700 transmixer	170	9	2	17
IT	Volvo L120H	203	10	5	51
LV	Toyota Landcruiser	151	8	9	68
Total Flowrate for Diesel (m^3/s)					**433**
Leakage @ 15%					65
Total Flowrate Including Leakage (m^3/s)					**498**
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.**

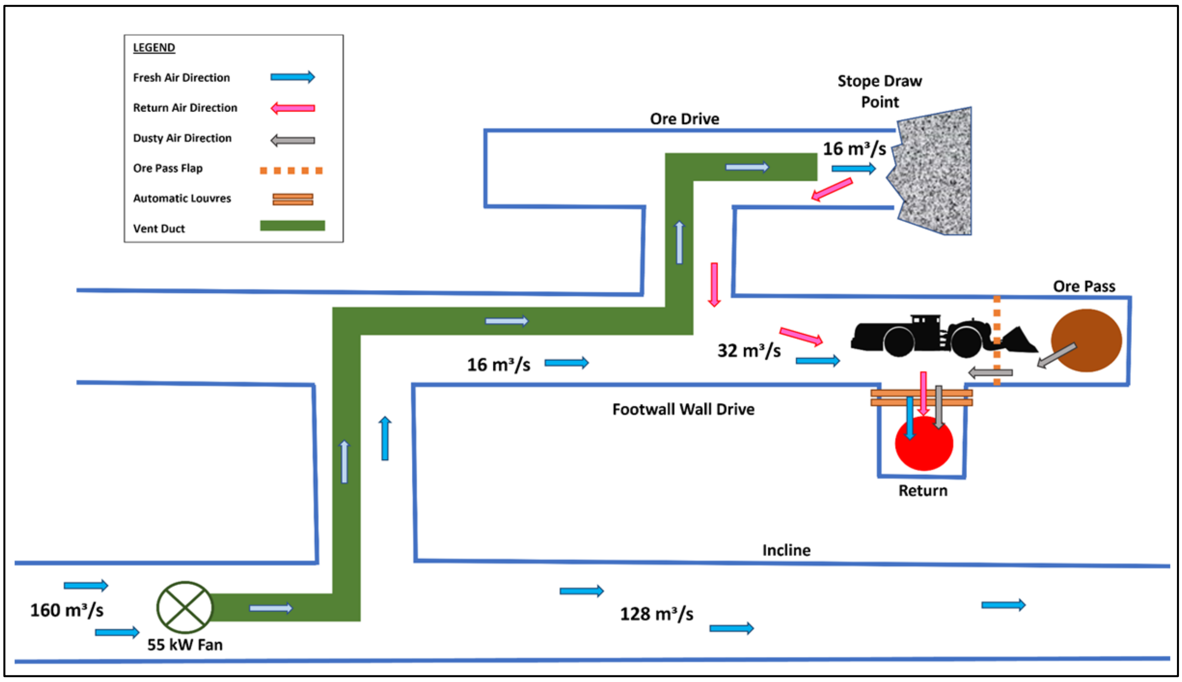
*****Minimum flowrate for activities relying on secondary air. The combination of two fans refers to activities in parallel.**

Production level design will combine secondary air with primary air depending on mining activity.

Development headings will be forced vented with 55 kW fans 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 on 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 at the end of the drive where ore will be tipped into ore pass. Any dust generated from the ore pass will directly report to the return air rise without polluting the incline. See Figure 1.7.5 for a typical ventilation plan.

Figure 1.7.5 Example of Production Level Vent Plan



1.7.6 Mining Fleet and Personnel

The Kainantu mine is an owner miner model whereby K92 runs all aspects of the mining operation, barring a small proportion of activities which utilize mining contractors. As such K92 employ the site personnel, and directly purchase infrastructure, equipment, and consumables as required.

Based on current site productivities, K92 utilized Entech's mine plan physicals to estimate the fleet and personnel requirements. The labour force is a mixture of expatriate employees and Papua New Guinean nationals. Entech reviewed all K92 equipment and personnel estimates and found them appropriate for the mine plan.

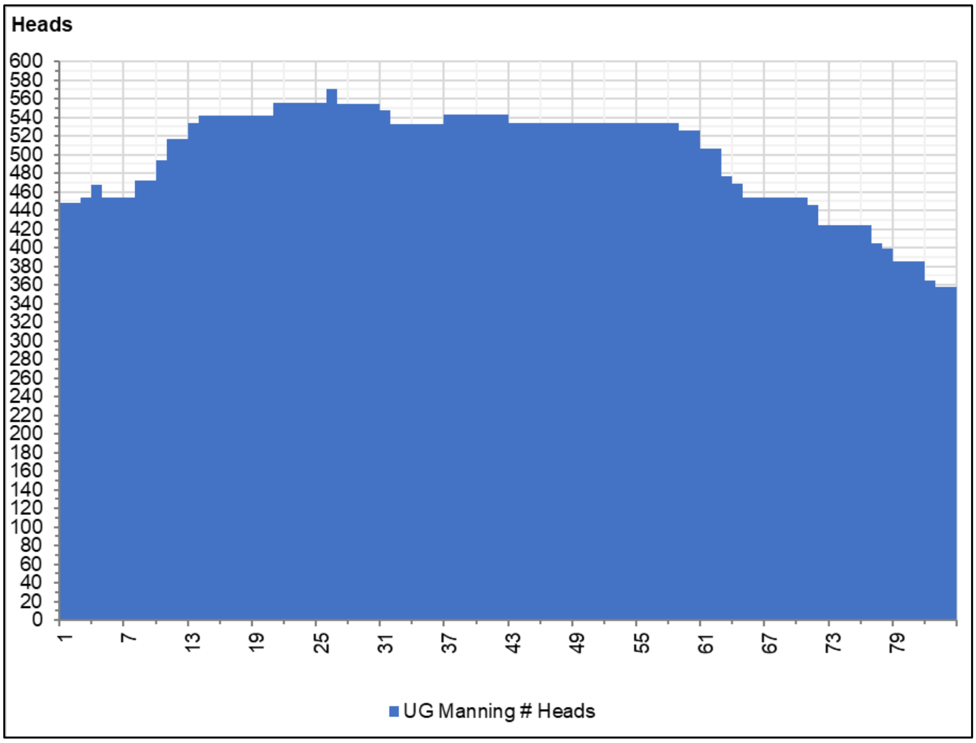
The peak fleet requirements are shown in Table 1.7.4 below.

Table 1.7.4 Peak Equipment Requirements

Equipment	Quantity
Primary	
Development Drills (DD421-60C)	6
Loaders (LH517)	5
Trucks (TH545)	7
Production Drills (DL421-7C/DL431-7C/DS 421C Bolter)	3
Slot Drill - Rhino 100 (or equivalent)	1
Production Charge-up (Getman)	1
Development Charge-up (Getman)	2
Ancillary	
Spraymec (6050WP)	2
Agitator	2
IT 856H Development	3
IT 856H Production	2
12K Grader	1
Scissor Lift (Ultimec)	1
Light Vehicles	9
Raisebore	1

The peak total underground personnel requirements are shown in Figure 1.7.6 below.

Figure 1.7.6 Total Underground Mining Personnel per Month



1.8 Recovery Methods

The process plant design is based on a robust metallurgical flowsheet designed for optimal metal recovery. The flowsheet chosen is based upon unit operations that are well proven in industry.

The key criteria for equipment selection are suitability for duty, reliability, and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements, whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the plant are:

- Throughput of 1.2 Mtpa of ore with a grind size of 80% passing (P_{80}) 106 µm.
- Crushing plant utilization of 68.5% (6,000 h/y).
- Grinding and flotation plant utilization of 91.3% (8,000 h/y) supported by crushed ore storage, stand-by equipment in critical areas and emergency power, if required, for controlled shutdown, emergency lighting and continuous operation of critical equipment.
- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control as and when required.

The treatment plant design incorporates the following unit process operations:

- Primary crushing with a jaw crusher.
- A crushed ore overflow surge bin and a dead stockpile to provide a buffer between the crushing and grinding circuit.
- The grinding circuit is a SAB type, which consists of an open circuit SAG mill and a closed-circuit ball mill with hydrocyclones, to produce a cyclone overflow product size of P_{80} 106 µm.
- A gravity gold recovery circuit for removal of coarse gold from the grinding circuit contains one batch centrifugal concentrator followed by two stages of gravity separation using shaking tables. The upgraded gravity concentrate is calcined and smelted in the gold room to produce gold doré bars.
- A flash flotation circuit for removal of coarse gold and copper particles from the grinding circuit contains one flash flotation cell. Flash flotation concentrate will flow by gravity to the concentrate thickener for addition to the final saleable concentrate product.
- The flotation circuit consists of rougher, scavenger, cleaner, cleaner scavenger, and recleaner stages to produce a saleable high-grade gold-copper concentrate.

- The flotation concentrate will be pumped to the concentrate thickener to dewater and increase the slurry density prior to concentrate filtration. The thickened concentrate will be filtered to achieve a discharge cake moisture of 10% before being loaded into 20 ft sea-containers for shipment overseas.

- Flotation tailings thickener will increase the slurry density for water recovery prior to tailings discharge to either the paste plant for backfill in the underground mine workings or the tailings storage facility.

1.9 Project Infrastructure

1.9.1 Power

The supply of electric power to the mine is via a grid connection to the PNG Power Ltd (PPL) network, 22kV overhead line (OHL).

A diesel back-up power station forms part of the study. Generators that are located on the existing power system at the underground and camp will operate independently as satellite systems.

1.9.2 Water

Raw water for the process plant will be sourced from the mine dewatering system from the 800 Portal. The water from the 800 Portal will be gravity fed to the plant site via an existing HDPE pipe and reports to the sediment settling ponds. Raw water is then pumped from sediment pond two into the raw water tank and used as required.

1.9.3 Paste Fill Plant Project Infrastructure

To improve underground geotechnical stability, facilitate complete ore extraction and dispose an estimated 46% of the Kainantu tailings underground, cemented paste backfill manufactured from fresh concentrator tailings is the preferred mine backfill method. The recommended system is designed to utilize 100% of the instantaneously available tailings, with a design production capacity of 123 m^3/hr. After accounting for an expected volumetric consolidation of around 7%, the proposed system is expected to satisfy the peak Kora backfill demand (of 45,000 m^3/mth), with a utilization of 53%. This utilization is at the lower end for quality backfill systems and, even after accounting for the complexity of the Kainantu delivery system, is expected to be comfortably achievable with the recommended paste system.

Stope bulkheads and exclusion zones (in front of bulkheads) have been sized to allow stopes to be filled in a single pass, which will eliminate downtime associated with fill cure periods and accelerate stope turn-around times. This is a proven methodology adopted at other sites with similar height 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 500 kPa) are typically required. A paste cure period of 14 days is incorporated into the mining schedule and this, combined with the relatively low strength requirements, means that an average binder content less than 3.5% is required. Optimization studies show the optimal binder product to be predominantly ground granulated blast furnace slag (slag) with GP cement addition. Being a waste product, the predominant use of slag (in the binding agent) presents a favourable solution from an Environmental perspective.

The main paste plant infrastructure is located at three different locations across the Kainantu site, with connection using a single high-pressure tailings transport pipeline. These infrastructure locations include Process Plant, 800L (Portal) and 1205 L (underground), with the relative location of each shown schematically in Figure 1.9.1. Also presented in this figure is the vertical extend of the Kora orebody.

Figure 1.9.1 Paste System Layout

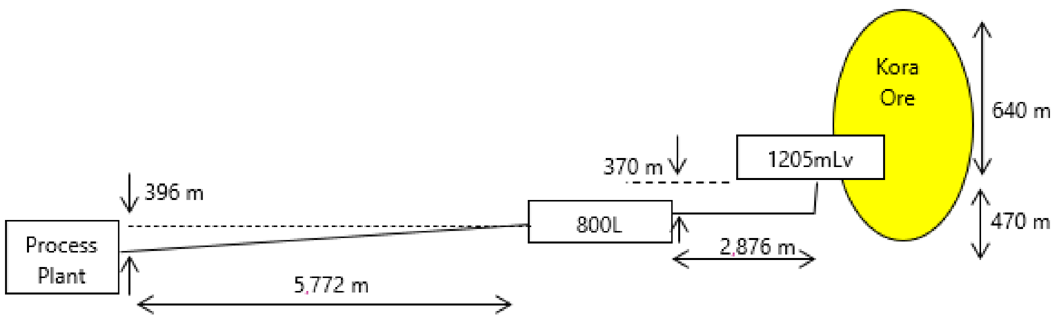


Figure 1.9.1 shows that, from the process plant, tailings/paste must be transported very significant distances both laterally and vertically to the Kora orebody.

Two unique aspects of the paste fill that are important for the system design include:

- Rheology testing of Kainantu tailings paste shows that addition of slag creates no noticeable change to the paste rheological behaviour, however addition of even small portions of GP cement creates a significant increase in yield stress. The magnitude of rheology change is such that cement addition changes the paste yield stress from approximately 50 Pa, which is ideal for hydraulic transport, to a value of approximately 300 Pa, which is considered ideal for paste deposited into Kora stopes.

- Paste strengths are largely controlled by the quantity of slag addition, with a constant addition (of circa. 1.0% GP) cement required to liberate the slag reaction.

Taking advantage of this behaviour, the recommended design includes:

- Tailings dewatering (including thickening and filtration) is to occur at the process plant to generate a relatively low yield stress paste tailings (circa. 50 Pa yield stress). Infrastructure is included to allow a slightly higher yield stress paste to be generated, which is then regulated back to 50 Pa (through water addition) to ensure paste consistency.

- Low yield stress paste is transported hydraulically, using a piston diaphragm pump, from the Process plant to the 800 Portal site.

- The 800 Portal site includes paste and waste storage (adequate for flushing and surge capacity) and infrastructure for the storage, metering and mixing of slag into the paste. As slag forms most of the binding agent, addition at the 800 Portal prevents the need to haul the slag component of the binder underground.

- Low yield stress paste (containing slag) is transported hydraulically, using a piston diaphragm pump, from the 800 Portal to the 1205/1170 underground chamber site.

- At the 1205 level paste enters a mixer where the material is combined with nominally 1% (w/w) GP cement. Paste mixing takes place in a 10,000 L ploughshare mixer. While GP cement addition is relatively small, the large mixer is considered appropriate to ensure paste with a consistent rheological behaviour and ensure an even distribution of binder collected at the mixer discharge, where quality control sampling is to be carried out.

- Mixed paste at a yield stress of nominally 300 Pa flows down a 35 m borehole to the 1170 level, where it is fed directly into a hydraulic piston pump. This delivery arrangement ensures a high pump net positive suction head, which is critical for minimising pump maintenance and downtime.

- From the hydraulic piston pump, paste is transported throughout the Kora mine. The 1205/1170 chamber is positioned to ensure that the hydraulic piston pump can distribute paste to all levels in the Kora orebody, without the need for any further pump stages.

- The paste reticulation system is equipped with valving and instrumentation to provide operators with visibility over the system operation and clear blockages as required, as well as a backup high pressure water flush pump.

1.9.4 Tailings Storage Facility

An increase to the processing throughput from 400,000 tpa to 1,200,000 tpa from the Kora deposit will result in an increased amount of tailings generated from 2022 to 2028. The tailings generated over the period will be stored by enlarging the capacity of the existing tailings storage facility (TSF) and implementing a Paste Backfill system to allow the use of tailings for stopes underground. During the initial period, 100% of the tailings generated will be sent to the TSF before the Paste Backfill operation commences. At mid-2024, once the Paste Backfill plant commences operation, approximately 50% of tailings will be sent to the voids of the underground mine, with the remaining portion of tailings sent to the TSF.

The existing TSF is approximately 21 m high cross-valley earth-fill dam with a crest level at RL 509 m, and currently under construction to raise the embankment to a crest level of RL 515 m (Stage 1A & 1B). The initial TSF was designed by Golder Associates Pty Ltd (Golder), and construction was completed by the end of 2005. A detailed design is already developed for an embankment crest level up to RL 520 m (Stage 2); however, raising the TSF to Stage 2 does not fulfil the capacity requirements due to the Kora Expansion. Therefore, to meet the increased production needs, the TSF requires an embankment raise to crest level up to RL 530 m (Stage 3).

The site has a tropical rainforest climate characterized by high annual rainfall with precipitation levels ranging between 1,750 mm and 2,000 mm. Based on the seismic activity of PNG, and probabilistic seismic hazard analysis, the likelihood of a seismic event at the TSF is considered very high.

To assess the consequence category of the TSF, a dam break analysis was carried out to estimate runout distances and the population at risk in the dam failure scenario. The results from the study determined that the consequence category for an environmental spill event is 'Very Low'; meanwhile, the consequence category for the dam break scenario was determined as 'Extreme'. Estimation of the population at risk showed that the Kumian mine camp is the most vulnerable facility with a large number of populations at risk. Hence, the recommendation to install a flood protection levee of at least 7 m high on the camp's eastern, southern, and western to protect the camp and minimize the impacts on the population from a dam break scenario.

The geotechnical investigation was undertaken to understand the foundation conditions beneath the TSF raise footprints identified further uncertainty relating to the ground conditions, concerning the presence of potentially liquefiable soils. Thus, there is a recommendation to conduct additional site investigation to address the uncertainties of the soil foundation, given the very high seismicity risk at the site.

To date, the preliminary assessment for expanding the existing TSF has confirmed the feasibility of raising the embankment to Stage 3 (RL 530 m). Further, preliminary information suggests that the existing TSF can accommodate additional dam construction, providing that the embankment slopes match the foundation strengths if foundation uncertainties are cleared once the site investigation is conducted.

Geochemical and physical testing of tailings found that the material is non-plastic, contains pyrite, and is potentially acid forming. However, testing of waste rock used for the embankment construction found that it has a low sulphur content and is non-acid forming.

An alternative to the existing TSF is the construction of a new TSF, which can provide the option to implement a range of solutions for tailings management; however, a significant amount of time is required to conduct an appropriate site investigation, develop the design, and undertake construction. Furthermore, the increase in the disturbed area will significantly increase the closure works required at the completion of the mining. Therefore, under the DFS Kora Expansion production plan scenario, the option of a new TSF is not considered a feasible alternative.

Expanding the existing TSF coupled with the Paste Backfill system is therefore confirmed as the preferred option due to the simplicity of operation, knowledge of the site conditions, capital cost, and operating cost.

1.9.5 Other 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.

Three bridges over water courses will be upgraded as part of the project scope. These bridges are on the service road from the process plant up to the 800 Portal, located within the mine lease.

The Kumian Camp will be upgraded as part of the study which includes three additional 64 bed accommodation units, an additional 20 two-person ensuites, a new community affairs building, addition of a training and induction centre, an OHS department building, extension to the mess, expanding the water, power and sewage systems and an upgrade to the recreation facility.

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 Conservation and Environment Protection Authority (CEPA) have granted various approvals, permits and licenses required by K92 to operate the Kainantu Gold Mine. The primary legislation for these approvals are the Mining Act 1992 and the Environment Act 2000 and they are subject to review and likely amendment for the Kora Expansion Project.

K92's existing mining lease, (ML150) contains approximately 90% 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 K92's exploration license, (EL 470). K92 will need to extend ML150 or apply for another mining license to include the extension of Kora to the south prior to mining.

K92 holds Environment Permit EP-L3(34) issued under the Environment Act 2000 for existing activities at the Kainantu Gold Mine. Expansion of the mine for the project will be subject to approval under the Environment Act 2000. The proposed increase in throughput to 1.2 Mtpa and the raising of the TSF embankment are substantial changes to what was originally approved in the 2002 Environment Plan and is likely to be subject to a major amendment of the existing environment permit for which an environment impact statement (EIS) will be required.

K92 is progressing a series of minor amendments to support current operations. K92 will require CEPA to issue the major amendment to the existing environment permit before the ramp-up of milling throughput to 1.2 Mtpa can commence. There is provision in the Environment Act 2000 for an environmental bond to be levied on a proponent before construction can commence.

1.10.2 Environment and community setting

The existing operation is in an environment that has been modified through existing mining activities.

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 K92 and previous operators. Results have been reported in annual reports from 2004 to 2020 and this monitoring is ongoing.

The existing project area is part of the upper Ramu River catchment. The surroundings have dendritic drainage system 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. A hydrogeological model is being developed as part of this FS and findings will be used as part of the technical assessments to inform the EIS.

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 Bilimoia Baseline Study (K92 Mine Limited, 2018) classified the population of the communities in the mine impacted area as uplanders in the Eastern Highlands Province and lowlanders in Morobe and Madang Provinces. The total population of the uplanders is around 7,580. The affected communities in the project area includes Bilimoia, Unantu / Puanono and Pomasi villages. The villages in the lowland area in the Madang and Morobe Province are Waterais villages and Marawasa / Musuwan villages with a total approximate population of 2,460. The main source of income for the area is through the Kainantu Gold Mine and associated support contracts (K92 Mine Limited, 2019).

K92 has a range of initiatives to support community and social development including the long-term supply of services to the mine, prioritization of local hiring, water and infrastructure development and medical and educational support.

In 2003, Highlands Pacific Limited (previous operator of Kainantu Gold Mine) signed a Lands and Environment Compensation Agreement with identified impact communities. Reviews of the agreement was scheduled to occur every three years however there has been no reviews to date. This is due to delays with the Land Titles Commission (LTC) completing an investigation into landholding within the existing operations leased boundaries. Landownership will remain under dispute until the LTC declaration of 2009 is resolved.

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 will be the focus of investigations during the EIS.

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 community through provision of livelihood trainings, educational assistance for children, 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 ML approximately 100 metres it is anticipated that there would be no change to the recognized Landowners. It is noted that the outstanding Land Titles Commission (TLC) determination of landholdings within the existing project will also apply to the extension of the ML. At this point some compensation payments, including distribution of a portion of royalties, are being accrued pending the determination from the TLC. A meeting was held in Goroka and the MRA has agreed to provide funding for the LTC.

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.

Greenhouse gas (GHG) emissions associated with the Stage 3 mine expansion are expected to reduce significantly based on a GHG/tonne milled. K92 is committed to transitioning the haulage fleet to electric powered. Further initiatives including: the improved materials handling design (reduction in TKM’s via orepasses), upgrading of the OHPL, and increased reliance on power supply from PNG Power Limited will contribute to K92’s objective of reducing the site’s carbon footprint.

1.10.4 Management plan

The existing operations run under the CEPA-approved 2020-2022 environment management plan (EMP). The purpose of the EMP is to satisfy conditions under the existing Environment Permit EP-L3(34); and to provide a policy framework, K92 management commitments, and monitoring and improvement actions necessary to prevent, mitigate or manage environmentally degrading conditions resulting from the existing operations. Routine environmental monitoring as part of the EMP will continue to provide relevant baseline data to assess existing operational impacts, provided it is conducted with a high degree of quality assurance and quality control. The K92 environment team will be involved in reviewing environmental management plans developed by contractors for the Kora Expansion Project activities to check they are consistent with the EMP 2020-2022 (or updates thereof) and to identify the need for further measures to be included in the contractor EMPs to minimize the environmental impact of the expansion project.

K92’s community relations team will prepare a Stakeholder Engagement Plan and Communications Pack for the Kora Expansion Project to inform community engagement.

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:

- Managing acid and metalliferous drainage issues post closure.
- Maintaining TSF integrity.
- Stability of underground stopes to reduce cave in risk.

1.10.6 Forward works plan

The forward works plan for environment and community studies is focussed on supporting the permitting of the expansion project, i.e., the preparation, submission, and assessment of the EIS. This includes a range of short-and long-term technical environmental and community studies to be undertaken by external consultants and supported by K92.

The indicative cost estimate for the forward works plan, i.e., the preparation and submission of the EIS, is AUD$495,000. This assumes the socio-economic impact assessment is part of the EIS preparation. It is advisable to allow for approximately AUD$200,000 for CEPA’s peer review of the EIS.

The timeframe for preparing the EIS is approximately 9 to 12 months from commencement, with an additional 6 to 9 months expected for the CEPA assessment, approval, and permit amendment process.

1.11 Capital and Operating Costs

1.11.1 Capital Cost

The capital estimate is summarized in Table 1.11.1 and Table 1.11.2. The initial project capital cost is estimated at US$177.3 M.

Table 1.11.1 Capital Estimate Summary (USD, 2Q22, +15% / -5%)

Main Area	US$M
Mine	0.0
Process Plant	80.1
Paste Plant	39.9
Power	23.4
Camp	5.5
Bridges	3.7
EPCM	20.4
Owners Costs	4.2
Subtotal	**177.3**

The duration of the detailed design and construction phase of the project has been estimated to be 23 months. The total LOM capital cost is estimated at US$401.7M, including sustaining capital costs of US$223.9M, as shown in Table 1.11.2.

Table 1.11.2 Sustaining Capital Estimate Summary (USD, 1Q21, +15% / -5%)

Main Area	US$M
Mining	199.6
TSF	18.8
Closure	5.5
Grand Total	**223.9**

1.11.2 Operating Cost

The mining operating cost estimate has been compiled by Entech. For the new process plant, the operating cost estimate has been compiled from a variety of sources, including K92 advice, metallurgical testwork, Orway Mineral Consultants (OMC) comminution modelling, first principle calculations, vendor quotations and the Lycopodium database. In terms of the infrastructure, the paste backfill system operating cost estimate has been compiled by MineFill Services. The TSF operating cost estimate has been compiled by ATC Williams. The general and administration (G&A) costs have been compiled by K92.

The operating cost estimates are summarized in Table 1.11.3.

Table 1.11.3 Operating Costs Summary (USD, 1Q22, +15% / -10%)

Area	Source	Unit / Value
Mining (average over LOM)	Entech	USD58 / t ore
Processing Plant – New - Fixed	Lycopodium	USD8.8M / year
Processing Plant – New - Variable	Lycopodium	USD7.61 / t ore
Processing Plant – New - Total (average over LOM)		USD15 / t ore
General & Admin - Total (average over LOM)	K92	USD29 / t ore
Paste Plant - Fixed	MineFill	USD1.1M / year
Past Plant - Variable	MineFill	USD22.87 / m^3 paste
Paste Plant - Total (average over LOM)		USD25 / m^3 paste
TSF (average over LOM)	ATC Williams	USD0.90 / t tails
Transport & Insurance (average over LOM)	K92	USD3.07 / t ore
Total (average over LOM)		**USD116 / 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).

1.12.1 Model Inputs and Assumptions

The key model inputs used in the economic analysis are summarized in Table 1.12.1.

Table 1.12.1 Key Model Inputs

Model Inputs	Source	Unit / Value
Base Currency		USD
Base Date		3rd Quarter 2022
Ore Processed over LOM	Entech	6.15 Mt
Metal Prices		
Gold	K92	USD1,600 / oz (fixed)
Copper	K92	USD4.00 / lb (fixed)
Silver	K92	USD20 / oz (fixed)
Recoveries		
Gold (total)	MMS	93.0%
Gold (gravity to doré)	MMS	15.0%
Copper	MMS	95.2%
Silver	MMS	80.0%
Concentrate copper grade	MMS	15.5%
Processing Plant Capacity (dry tonnes of ore)		
Existing Plant	K92	500,000 tpa
New Plant	Lycopodium	1,200,000 tpa
Royalties (deducted from gross revenue)	K92	2.0%
Levies (deducted from gross revenue)	K92	0.5%
Tax Rate	K92	30%
Depreciation		Not considered
NPV Discount Rate	K92	5%

1.12.2 Financial Model Results

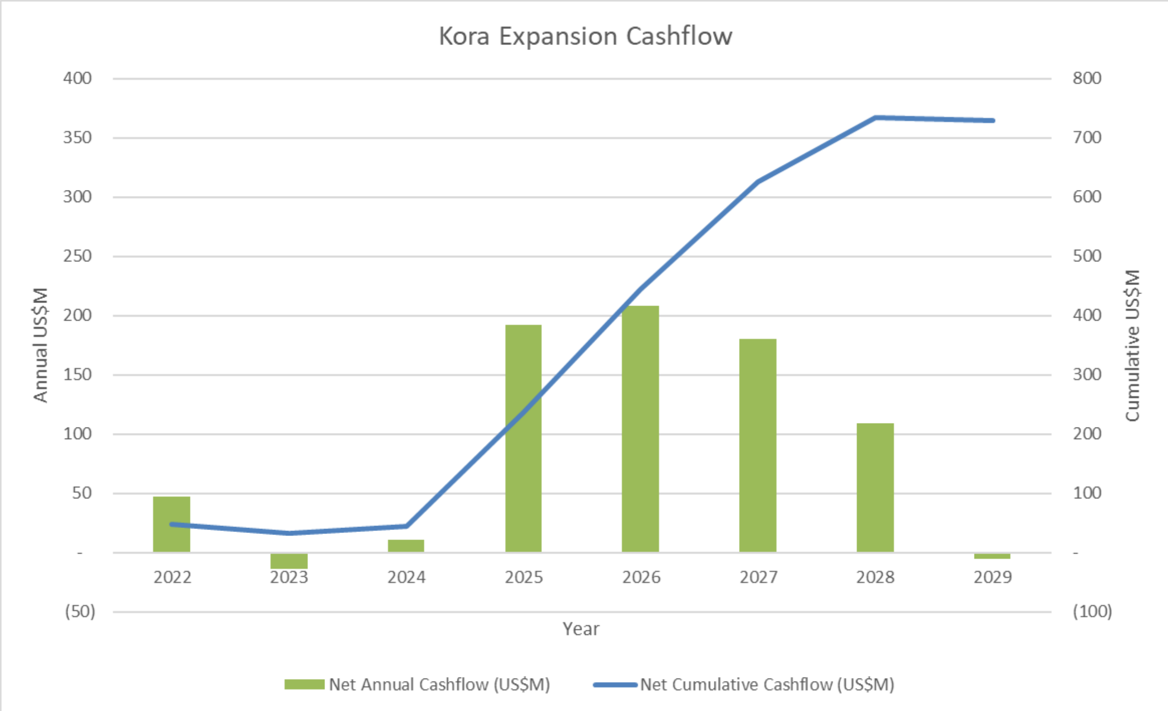
The financial model indicates that the project has a post-tax Net Present Value (NPV) of USD586M at a discount rate of 5%.

Figure 1.12.1 shows the post–tax annual and cumulative cash flow for the project over the LOM.

As the model starts with a positive cash flow in 2022 due to revenue from the existing plant operation, the cumulative cash flow position is never negative.

Due to this initial positive cash flow, an Internal Rate of Return (IRR) is not published.

Figure 1.12.1 Cumulative Cash Flow



1.13 Kora 2022 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 1.7 Mtpa, representing a 240% increase from the Stage 2A run-rate of 500,000 tpa. The PEA Case involves the construction of a standalone 1.2 Mtpa process plant adjacent to the 500,000 tpa Stage 2A process plant. At the end of 2024, the Stage 2A process plant is idled as the 1.2 Mtpa Stage 3 process plant ramps up, with commissioning of the Stage 3 process plant commencing in Q3 2024. Upon achieving the Stage 3 run-rate throughput in 2025, the Stage 2A plant is recommissioned in mid-2026, ramping up to run-rate throughput of 500,000 tpa by year end, for a combined processing run-rate of 1.7 Mtpa at the beginning of 2027.

To support the higher throughput rate, the underground mining fleet is significantly increased to support the expanded mining operations and opening 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 run-rate.

The PEA uses the conclusions of the Company's mineral resource estimate for Kora (effective date of October 31, 2021) and Judd (effective date of December 31, 2021) and does not incorporate post resource drilling results. The effective date of the PEA life of mine plan is 1 January 2022; therefore, Kora is net of post-resource mining depletion from 1 November 2021 to 31 December 2021 which totals 36,765 tonnes at 12.94 g/t Au, 0.59% Cu and 9.29 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 1.13.1 PEA Highlights

US Dollars unless otherwise stated	Life of Mine (Starting January 2022)	Stage 4 Expansion (3Q 2026 onwards)
Production		
Mine life (years)	11 years	
Total mill feed (000s tonnes)	12,156	
Average mill throughput (tonnes per annum)	1,105 ktpa	1.7 Mtpa (run-rate)[1]
Total Metal Production		
AuEq (000s ounces)	**3,398**	**2,134**[2]
Gold (000s ounces)	2,555	1,579[2]
Copper (mlbs)	302	198[2]
Silver (000s ounces)	7,040	4,726[2]
Peak Annual Production		
Year		2027
AuEq (000s ounces per annum)		500
Average Annual Metal Production		
AuEq (000s ounces per annum)	**309**	**406 (run-rate)**[1]
Gold (000s ounces per annum)	**232**	297 (run-rate)[1]
Copper (mlbs per annum)	27	39 (run-rate)[1]
Silver (000s ounces per annum)	640	928 (run-rate)[1]
Average Grade		
AuEq grade (g/t)	**9.3 g/t**	
Gold grade (g/t)	7.0 g/t	
Copper grade (%)	1.2%	
Silver grade (g/t)	23 g/t	
Average Recovery		
Gold Recovery (%)	93%	
Copper Recovery (%)	95%	
Silver Recovery (%)	80%	
Costs		
Mining cost per tonne (US$/t)	$72.05	$65.43 (run-rate)[1]
Processing cost per tonne (US$/t)	$18.02	$17.65 (run-rate)[1]
G&A cost per tonne (US$/t)	$28.19	$25.89 (run-rate)[1]
Total operating cost per tonne of mill feed (US$/t)	**$118.26**	**$108.97 (run-rate)[1]**
Sustaining capital per tonne of mill feed (US$/t)	$35.33	$20.79 (run-rate)[1]
Total cost per tonne of mill feed (US$/t)	**$153.59**	**$129.76 (run-rate)[1]**
Expansion capital expenditure ($m)	$187	
Sustaining capital expenditure ($m)	$429	
Total capital expenditure with closure costs ($m)	**$628**	
Cash cost per ounce AuEq ($/oz)[3]	$546	$527 (run-rate)[1]
All-in sustaining cost per ounce AuEq ($/oz)[4]	$674	$604 (run-rate)[1]
Cash cost per ounce gold ($/oz)[3]	$275	$220 (run-rate)[1]
All-in sustaining cost per ounce gold ($/oz)[4]	$444	$325 (run-rate)[1]
Base Case Economic Analysis at US$1,600/oz Gold, US$4.00/lb Copper and US$20.00/oz Silver		
After-tax NPV0%	$1.8 billion	
After-tax NPV5%	$1.3 billion	
IRR (%) and Payback Period (years)	N/A (Self-Funded)	
Economic Analysis at $2,000/oz Gold, US$4.00/lb Copper and US$20.00/oz Silver		
After-tax NPV0%	$2.5 billion	
After-tax NPV5%[5]	$1.8 billion	
IRR (%) and Payback Period (years)	N/A (Self-Funded)	

1.Run-rate excludes the final partial calendar year of production.

2.Excludes 2H 2026 Stage 4 commissioning and initial ramp-up stage.

3.Cash costs are net of by-product credits and are inclusive of mining costs, processing costs, site G&A and refining charges and royalties.

4.AISC includes cash costs plus estimated corporate G&A, sustaining costs and accretion.

5.Net present value is calculated utilizing mid-year discounting.

1.13.2 Kora 2022 PEA Mining Methods

Introduction

Entech were engaged by K92 Mining Inc. (K92) to complete a Preliminary Economic Study (PEA) on underground mining of the Kainantu Gold Mine. The PEA targets an expansion to a peak processing rate of 1,700,000 tpa with an 11-year mine life. The PEA assumes a same start date of the life of mine schedule, of 1 January 2022.

Geotechnical

A total of 3,662 m of drill core was geotechnically logged from core photographs in detail for the Kora underground mining assessment during Q1 and Q3 2021. The drillholes were selected from existing exploration holes. From this total, 1,946 m, was logged as part of an initial campaign, with a second campaign of 1,716 m completed in October 2021.

In addition to the 3,662 m, 472.3 m of core was validated and logged associated with the Judd orebody.

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 K92 staff and provided to Entech.

Rock property testing specific to the orebodies was unable to be sampled and sent for testing again due to covid restrictions, resourcing availability in country, and available drill holes for sampling. 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 remain a significant information gap for the project and should be addressed as soon as practical.

The drill core logging data was analysed by Entech and forms the basis of this study. This information has allowed for characterization of the rock mass, assessment of stable stoping span predictions and estimates of dilution factors.

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 utilized for visualization 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 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.

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 pastefill plant, and open stoping with cemented pastefill with both bottom-up and 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 and extracted continuously 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 both the Kora and Judd orebodies can be classed as 'good ground'. Table 1.13.2 outlines the recommended stope spans and dilution estimates for the Kora and Judd orebodies.

Table 1.13.2 Summary of Assessed Stoping Parameters for Kora and Judd

Orebody	Parameters	Hangingwall	Footwall
K1	Allowable Strike Length	16 m	20 m
	Dilution	0 - 0.5 m	0 - 0.5 m
K2	Allowable Strike Length	31 m	19 m
	Dilution	0 - 0.5 m	0 - 0.5 m
Judd	Allowable Strike Length	35 m	35 m
	Dilution	0 - 0.5 m	0 - 0.5 m

The presence of the FGZ and pervasive structure throughout the rock units will dominate stope wall behaviour with the possibility of slabbing, sliding and unravelling failure types occurring 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 will be the key to minimising stoping performance issues.

Mining Method Selection

A mining method selection process was completed by generating a long list of potential mining methods, which were shortlisted. Trade-off studies were completed on mining methods, materials handling options, production rate analysis, and ventilation strategies. Scenario modelling was used to estimate inventories, costs, cashflows and net present values for each mining method.

The mining method selection process resulted in Avoca and modified Avoca being selected for mining prior to commissioning of the paste fill plant in Q2 2024, and longhole stoping with pastefill for the remainder of the mine life.

Mine Design and Physicals

On selection of the mining method, Entech completed stope optimizations on the Mineral Resource. The PEA includes Measured, Indicated, and Inferred Mineral Resources.

K92 selected a 4.5 g/t AuEq stope cut-off grade to carry forward for the final mine design and PEA LOM scheduling. This selection was supported by NPV analysis, cut-off grade estimates, trade-off study results, as well as aligning with K92's objectives for inventory size and grade.

Dilution of 0.5 m on both the footwall and hanging wall of the stope shapes (1.0 m total) at contained Mineral Resource grade was applied during the optimization phase. An additional 0.5m of dilution at contained Mineral Resource grade was applied to stope shapes that were within 2 m from the fault gouge zone. These dilution parameters were based on Entech's geotechnical study.

An additional 2.5% dilution at waste grade was applied to pastefilled stoping from overmining of paste fill. An additional 5% waste rock dilution at waste grade was applied for Avoca stoping methods from overmining of waste rock fill. Paste fill and Avoca dilution are based on benchmark and site dilution data. The LOM average stope dilution is 20%.

To account for ore losses a mining recovery factor of 90% has been applied to Avoca stopes and 95% has been applied to LHS stopes with paste fill. The 5% lower recovery factor for Avoca stoping accounts for ore loss where ore and waste material mix and become subeconomic to mine.

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 development design can be found in Figure 1.13.1 below, followed by Figure 1.13.2 detailing the stope extraction methods.

Figure 1.13.1 Isometric View of the Kainantu Underground Development Layout

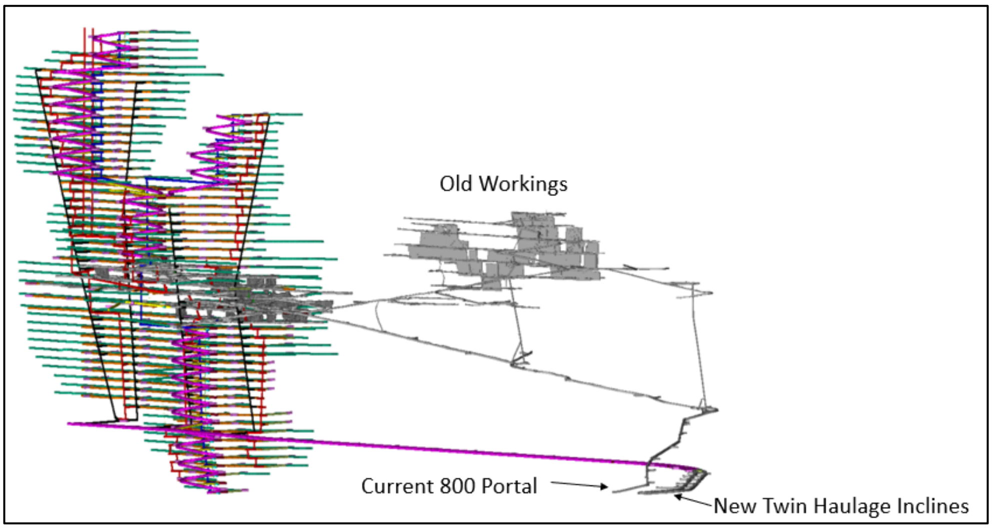
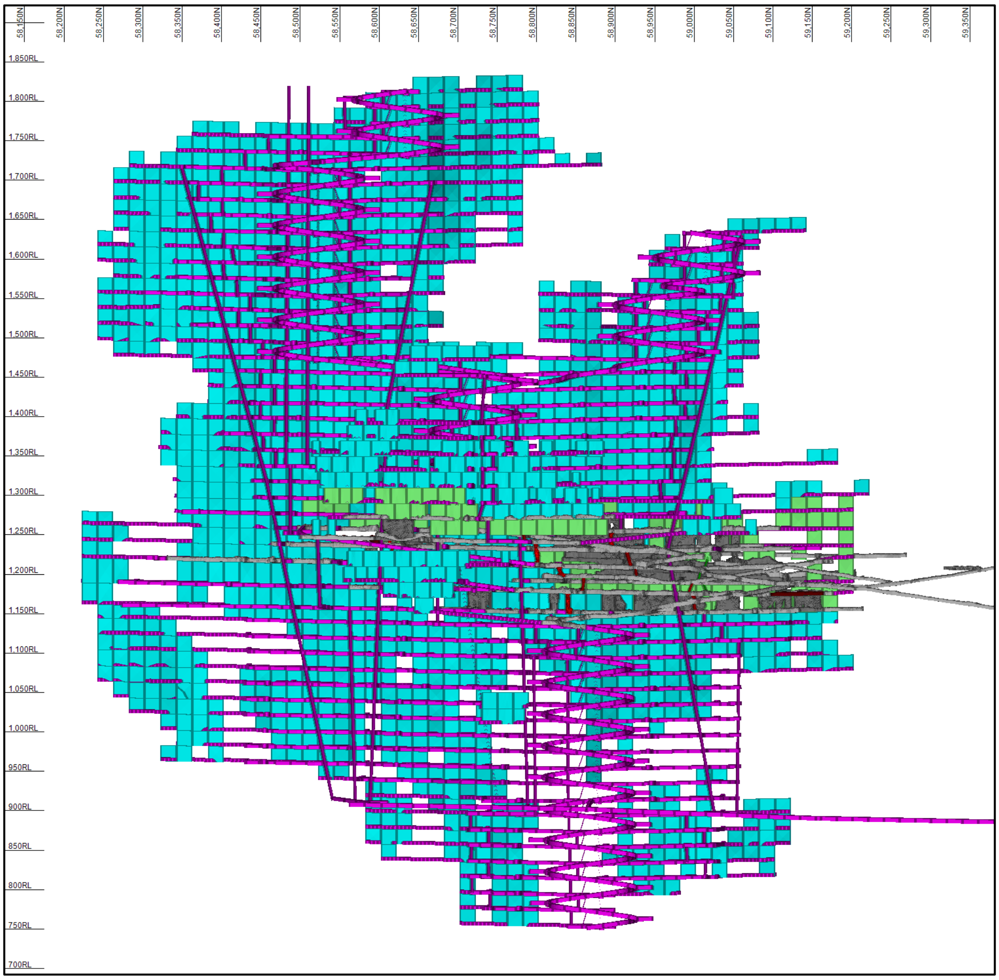
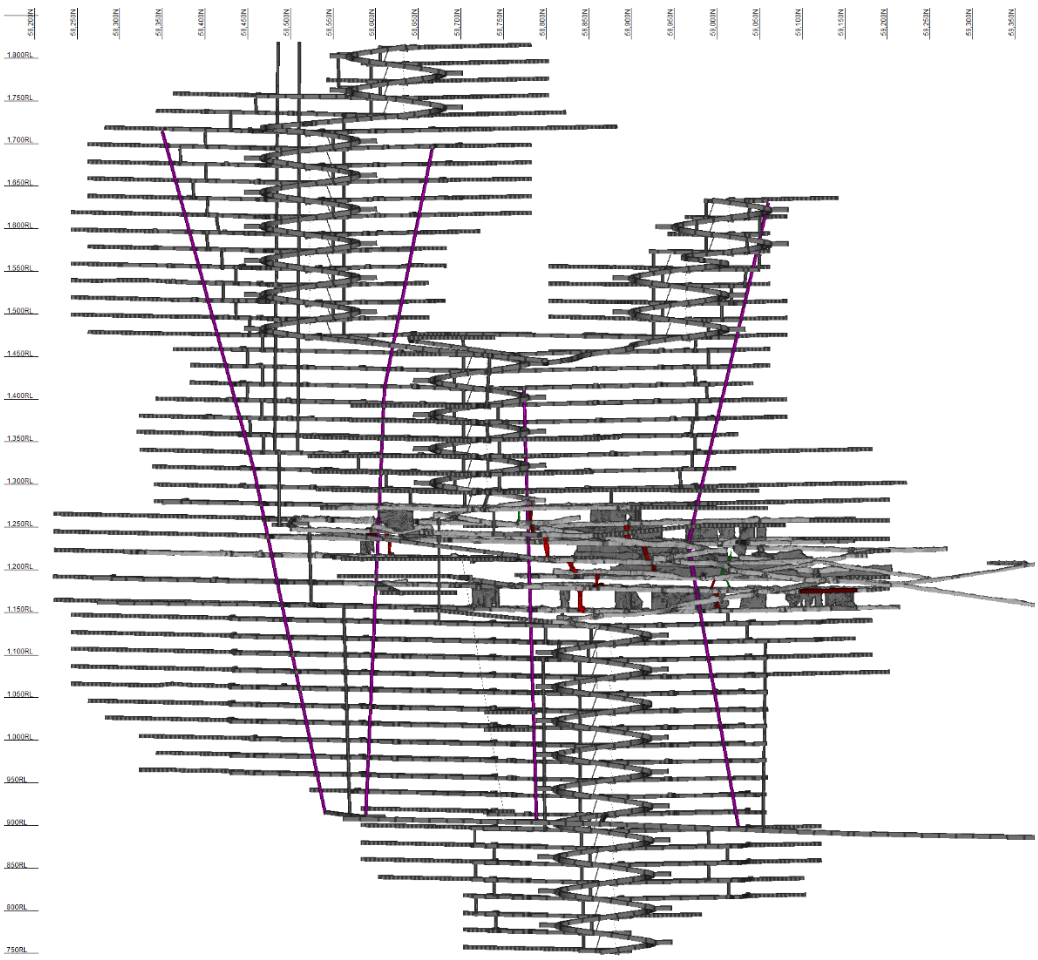


Figure 1.13.2 Looking West – LHS with Fill (Blue), Avoca (Green), As-builts (Grey)



The trade-off study quantitative and qualitative analysis resulted in the selection of the PEA utilising an ore pass network to increase production rate and efficiency, while future proofing the design. The ore pass network is shown in Figure 1.13.3 below.

Figure 1.13.3 Looking West - Development Design (Ore Passes in Pink)



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 peak rate of 1,700,000 t per annum (1.7 Mtpa). The mine life is 11 years with a 6-year production ramp up (inclusive of 2022), with sustained production of 1.7 Mtpa for 3 years, and final year production of 0.2 Mtpa. A summary of the key physicals per annum is shown in Table 1.13.3.

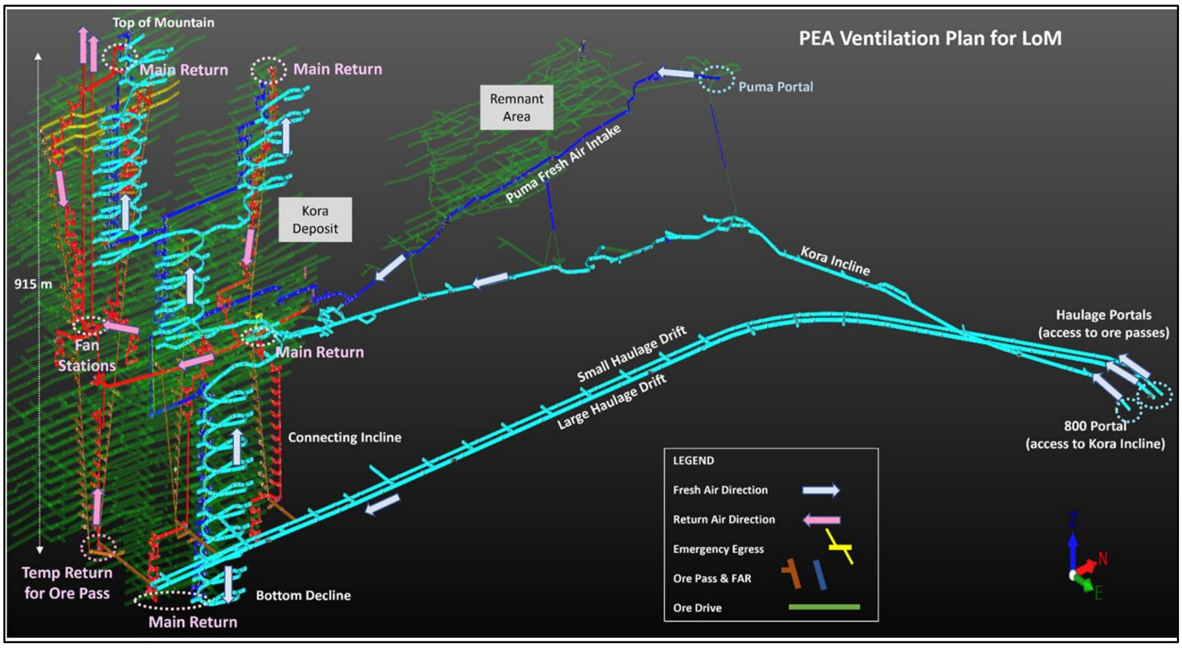
Table 1.13.3 Physicals Per Annum

		Total/Av	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Lateral Development Total	**m**	**110,945**	**9,020**	**13,192**	**14,292**	**14,433**	**14,435**	**14,436**	**14,430**	**13,420**	**3,285**		
Vertical Development Total	**m**	**36,695**	**1,328**	**2,050**	**4,233**	**2,914**	**3,615**	**4,678**	**5,355**	**4,175**	**4,835**	**2,777**	**735**
Ore Profile													
Ore Tonnes	t	12,155,527	450,568	524,687	835,166	1,225,312	1,518,036	1,701,871	1,707,267	1,668,776	1,455,243	844,924	223,678
Au Grade	g/t	6.98	9.53	7.90	6.64	6.47	7.61	7.66	6.04	6.51	5.51	7.13	14.02
Au Ounces	oz	2,727,923	138,046	133,213	178,254	254,979	371,205	419,396	331,347	349,033	257,869	193,750	100,830
Cu Grade	%	1.20	0.8	0.8	1.2	1.3	1.1	1.0	1.2	1.4	1.5	1.2	0.9
Cu Tonnes	t	145,361	3,438	4,320	9,882	15,786	16,117	17,558	19,869	23,957	22,192	10,129	2,112
Ag Grade	g/t	22.85	14.61	14.97	22.69	23.21	18.80	18.77	23.17	29.60	30.45	22.26	15.13
Ag Ounces	oz	8,930,020	211,598	252,485	609,194	914,184	917,597	1,027,064	1,272,019	1,587,964	1,424,534	604,562	108,820

Ventilation

The PEA plan to ventilate the underground sees fresh air entering the mine through four portals in the side of the mountain hosting the Kora deposit, with air exhausting out two exhaust raises daylighting at the top of the mountain. The Puma portal will be repurposed as an intake, which is currently the main exhaust for the existing workings. Figure 1.13.4 illustrates the plan.

Figure 1.13.4 LOM Ventilation Plan Profile Facing Northwest



Fresh air enters all portals and converges at the mid-point, where production is currently occurring. From here an incline continues to the top and bottom of the deposit. Exhausting air links all production zones to underground fan stations, located at the base of two 480 m raises to surface.

Table 1.13.4 contains a list of the mobile equipment intended for use at K92 (or equivalent unit) and shows the peak number of vehicles anticipated with the subsequent total flowrate.

Table 1.13.4 Peak Mobile Equipment Flowrate Calculation

Diesel Unit	Assumed Model	Engine Power Rating (kW)	Flowrate Requirement (m^3/s) *	Count	Total Flowrate (m^3/s)
Truck	Sandvik TH545	515	26	9	232
Loader	Sandvik LH517	310	16	10	155
Charge-up	Getman	120	6	4	24
Development Drill **	Sandvik DD421-60C	110	6	8	0
Production Drill **	Sandvik DL431-7C	119	6	4	0
Cable Bolter**	Sandvik DS421C	119	6	1	0
Grader	12K Grader	134	7	1	7
Water Cart	Volvo A30D	242	12	1	12
Fibrecrete Sprayer	Normet Spraymec SF 050 D	90	5	2	9
Agitator	Normet LF 700 transmixer	170	9	3	26
IT	Liugong	203	10	5	52
LV	Toyota Landcruiser	151	8	16	124
Total Flowrate for Diesel (m^3/s)					**640**
Leakage @ 15%					96
Total Flowrate Including Leakage (m^3/s)					**736**
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.**

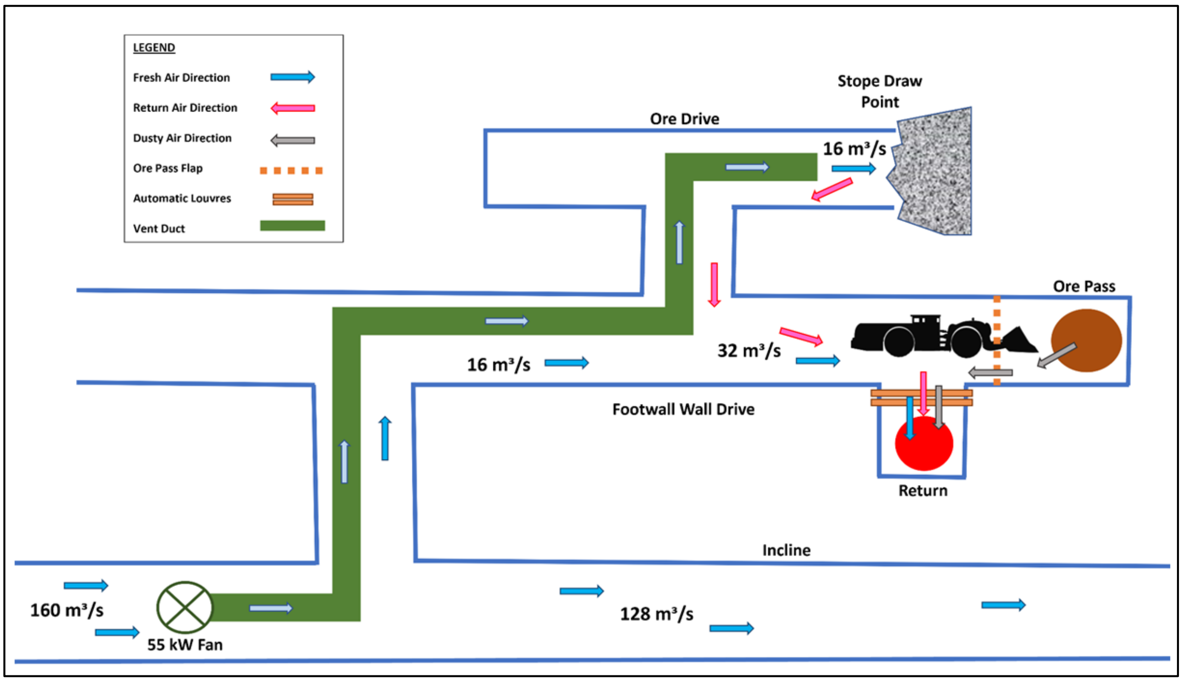
*****Minimum flowrate for activities relying on secondary air. The combination of two fans refers to activities in parallel.**

Production level design will combine secondary air with primary air depending on mining activity.

Development headings will be forced vented with 55 kW fans 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 on 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 at the end of the drive where ore will be tipped into ore pass. Any dust generated from the ore pass will directly report to the return air raise without polluting the incline. See Figure 1.13.5 for a typical ventilation plan.

Figure 1.13.5 Example of Production Level Vent Plan



Mining Fleet and Personnel

The Kainantu mine is an owner miner model whereby K92 runs all aspects of the mining operation, barring a small proportion of activities which utilize mining contractors. As such K92 employ the site personnel, and directly purchase infrastructure, equipment, and consumables as required.

Based on current site productivities, K92 utilized Entech's mine plan physicals to estimate the fleet and personnel requirements. The labour force is a mixture of expatriate employees and Papua New Guinean nationals. Entech reviewed all K92 equipment and personnel estimates and found them appropriate for the mine plan.

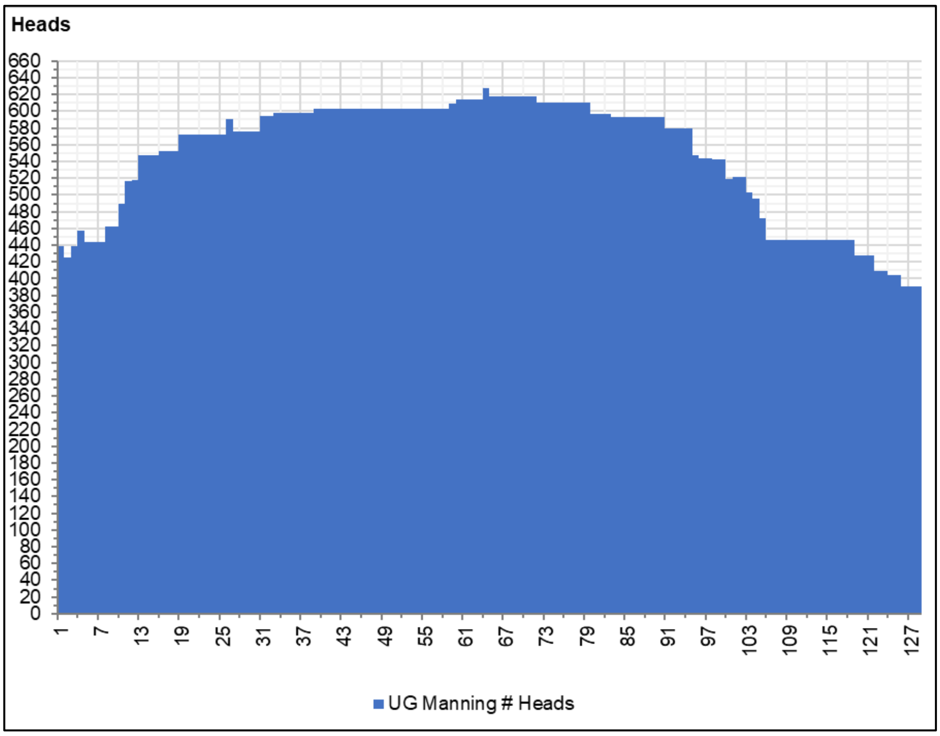
The peak fleet requirements are shown in Table 1.13.5 below.

Table 1.13.5 Peak Equipment Requirements

Equipment	Quantity
Primary	
Development Drills (DD421-60C)	8
Loaders (LH517)	10
Trucks (TH545)	9
Production Drills (DL421-7C/DL431-7C/DS 421C Bolter)	5
Slot Drill - Rhino 100 (or equivalent)	1
Production Charge-up (Getman)	2
Development Charge-up (Getman)	2
Ancillary	
Spraymec (6050WP)	2
Agitator	3
IT 856H Development	3
IT 856H Production	2
12K Grader	1
Scissor Lift (Ultimec)	1
Light Vehicles	16
Raisebore	1

The peak total underground personnel requirements are shown in Figure 1.13.6 below.

Figure 1.13.6 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 was provided to a mining contractor, who provided a pricing estimate for these activities.

Mining Capital Costs

The estimated capital costs are summarized in Table 1.13.6.

Table 1.13.6 Mining Capital Cost Totals

Description	Unit	Value
Infrastructure	USD (M)	14.4
Decline	USD (M)	28.5
Cap Access	USD (M)	11.4
Ventilation	USD (M)	46.4
Escapeway	USD (M)	3.7
Other Lateral Development	USD (M)	66.8
Fleet	USD (M)	77.8
Operators and Maintenance	USD (M)	92.0
Capital Mine Services	USD (M)	24.3
Capital Mine Overheads	USD (M)	18.1
Total Capital	**USD (M)**	**383.5**

A breakdown of the capital unit costs is shown in Table 1.13.7.

Table 1.13.7 Mining Capital Unit Costs

Description	Unit	Value
Infrastructure	$ / t ore	1.18
Decline	$ / t ore	2.34
Capital Access	$ / t ore	0.94
Ventilation	$ / t ore	3.82
Escapeway	$ / t ore	0.31
Other Lateral Development	$ / t ore	5.50
Fleet	$ / t ore	6.40
Operators and Maintenance	$ / t ore	7.57
Capital Mine Services	$ / t ore	2.00
Capital Mine Overheads	$ / t ore	1.49
Total Capital Cost	**$ / t ore**	**31.55**

Mining Operating Costs

The estimated operating costs are summarized in Table 1.13.8.

Table 1.13.8 Mining Operating Cost Totals

Description	Unit	Value
Operating Access	USD (M)	18.7
Ore Drive	USD (M)	88.0
Stope	USD (M)	246.1
Operators and Maintenance	USD (M)	207.7
Operating Mine Services	USD (M)	59.2
Operating Mine Overheads	USD (M)	48.2
Surface Haulage	USD (M)	68.7
Grade Control	USD (M)	35.7
Total Operating	**USD (M)**	**772.4**

A breakdown of the operating unit costs is shown in Table 1.13.9.

Table 1.13.9 Mining Operating Unit Costs

Description	Unit	Value
Operating Access	$/t ore	1.54
Ore Drive	$/t ore	7.24
Stope	$/t ore	20.25
Operators and Maintenance	$/t ore	17.09
Operating Mine Services	$/t ore	4.87
Operating Mine Overheads	$/t ore	3.97
Surface Haulage	$/t ore	5.65
Grade Control	$/t ore	2.94
Total Operating	**$/t ore**	**63.54**

1.13.3 Kora 2022 PEA Infrastructure

Evaluating different solutions to store the PEAs tailings, including the increased capacity in the existing TSF or in a newly constructed tailings facility. The critical constraints were identified from the several options evaluated; for instance, the time required to implement the solution, understanding the foundation conditions, tailings characterizations, suitable borrow material availability, and site limitations, discussed further below.

First, the time required to implement a new facility for the PEA Kora Expansion shall be considered at least three years from now, considering the site investigation, options assessments, and the development of the engineering, construction, procurement activities and regulatory aspects. Second, the currently poor understanding of the foundation conditions for a new TSF location due to the geological and geotechnical complexities typical of PNG would not necessarily ensure that selected sites would be suitable for installing a new facility. Third, there are additional learnings to be understood of the characteristics of the tailings, and the expected variation for those generated from Kora in the coming years create uncertainties for suitable mechanical dewatering solutions. Further, PNG's highly weathered soils do not generally provide optimal conditions for earthwork, so borrowing material to build a new dam is currently considered uncertain. Finally, the site limitation due to the remote location, limited availability of skilled labour, and logistics difficulties may constrain most of the best available technologies from being implemented.

The evaluation outcome for TSF options suggests two potential solutions. First, a conventional TSF with a 25m height rockfill dam embankment built with borrowed material and thickened tailings discharge subaerial into the TSF. Second, the classification of tailings coarse and fine particles through hydrocyclones, coupled with the construction of a 15m dam using geotubes filled with the fine fraction of classified tailings and stacked on top of each other; the coarse fraction of tailings will be stored as a sand stockpile within the impoundment created by the geotubes dam. The high permeable coarse fraction would quickly drain by gravity and is expected to reach a higher settled density than the current tailings stored in the TSF. While the fine fraction is considered the most complex tailings, the deposition of that fraction within the geotubes would allow their encapsulation and ability to build the dam to create the impoundment to store the coarse fraction.

Both solutions will provide enough volume to store the tailings generated for the PEA Kora Expansion after the capacity of the existing TSF at stage 3 exhaust by 2027. For the purpose of this assessment, K92 have adopted the implementation of conventional TSF with a rockfill dam as the preferred solution.

For the PEA case, the Kumian Camp undergoes further expansion. This includes an additional 2 sixty-four bed blocks and an additional 10 two person ensuite blocks, additional to what is described under section 18.9 in this report.

1.13.4 Kora 2022 PEA Paste System

To satisfy the backfill requirements for the PEA scenario a peak backfill requirement of 70,000 m^3/mth is required. Utilising all available tailings the backfill system would have an 'instantaneous' production rate of 174 m^3/h, 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 is considered achievable.

PEA mining scenario and 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 expected to remain unchanged.

Relative to the flowsheet described for the DFS, the only change, to accommodate the increased production rate, is duplication of the vacuum disc filter. In addition, it is also necessary to increase the capacity of various system components, including most pumps and the paste mixer. No changes are required to the storage / mixing tanks, overland pipelines, or the binder addition system.

With the similar system no increase in personnel is expected to be required, relative to the DFS model. This results in a slight reduction in operating cost for the PEA relative to the DFS.

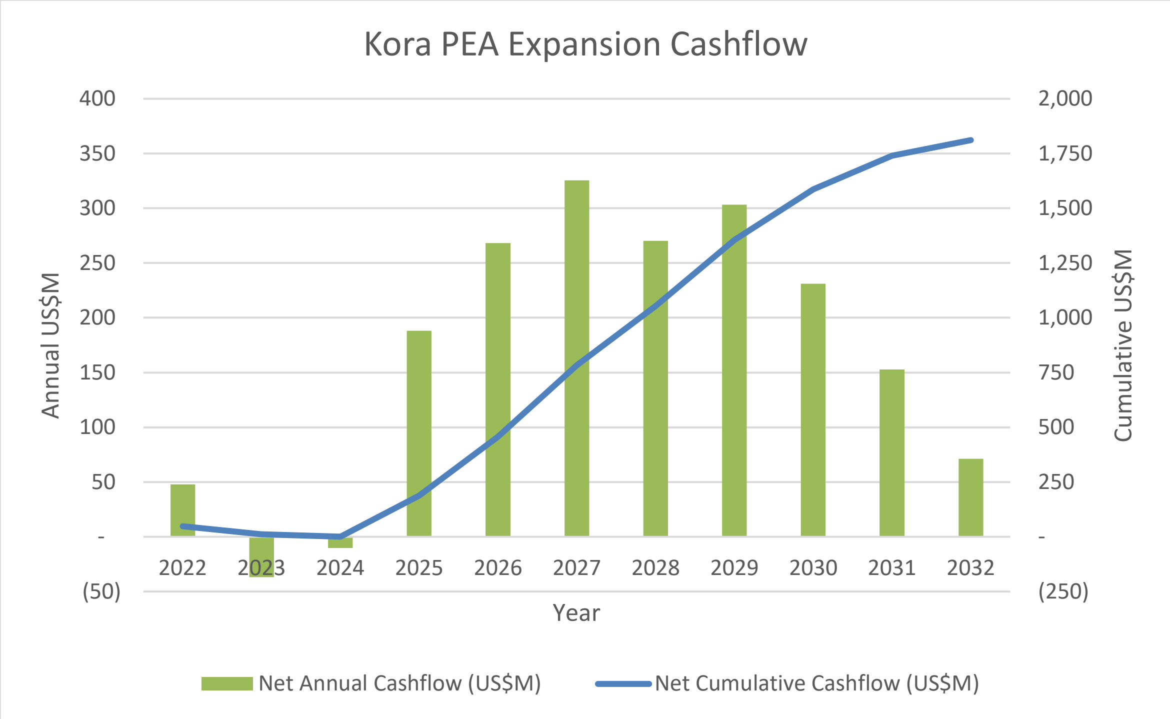
1.13.5 Kora 2022 PEA Economic Analysis

An economic analysis has been carried out for the expanded 1.7 Mtpa project using a cash flow model, similar to that carried out for the 1.2 Mtpa project.

The financial model indicates that the PEA project has a post-tax Net Present Value (NPV) of USD1,325M 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 1.13.7 shows the post–tax annual and cumulative cash flow for the project over the LOM.

Figure 1.13.7 PEA Cumulative Cash Flow



1.14 Interpretation and Conclusions

The DFS evaluates an expansion of mining and processing to a run-rate throughput of 1.2 Mtpa, representing a 140% increase from the Stage 2A run-rate of 500,000 tonnes per annum (tpa). This expansion is referred to as the Stage 3 Expansion and involves on-site treatment of ore by a new standalone 1.2 Mtpa process plant, utilizing single stage crushing, SAG and ball milling, gravity, and flotation recovery.

The DFS and mineral reserve statement is derived from the Company's Mineral Resource estimate for Kora (effective date of October 31, 2021) and Judd (effective date of December 31, 2021), with Kora depleted based on mining actuals until 31 December 2021, and does not incorporate post-resource-estimate drilling results.

Table 1.14.1 DFS Highlights

US Dollars unless otherwise stated	Life of Mine (starting January 2022)	Stage 3 Expansion (Q3 2024 onwards)
Production		
Mine life (years)	7 years	
Total mill feed (000s tonnes)	6,153	
Average mill throughput (tonnes per annum)	879 ktpa	1.2 Mtpa (run-rate)(1)
Total Metal Production		
AuEq (000s ounces)	**1,544**	**1,049**(2)
Gold (000s ounces)	1,224	799(2)
Copper (mlbs)	114	89(2)
Silver (000s ounces)	2,773	2,164(2)
Peak Annual Production		
Year		2026
AuEq (000s ounces per annum)		309
Average Annual Metal Production		
AuEq (000s ounces per annum)	**221**	**291 (run-rate)**(1)
Gold (000s ounces per annum)	175	224 (run-rate)(1)
Copper (mlbs per annum)	16	24 (run-rate)(1)
Silver (000s ounces per annum)	396	574 (run-rate)(1)
Average Grade		
AuEq grade (g/t)	**8.4 g/t**	
Gold grade (g/t)	6.7 g/t	
Copper grade (%)	0.9%	
Silver grade (g/t)	18 g/t	
Average Recovery		
Gold Recovery (%)	93%	
Copper Recovery (%)	95%	
Silver Recovery (%)	80%	
Costs		
Mining cost per tonne (US$/t)	$66.54	$61.97 (run-rate)(1)
Processing cost per tonne (US$/t)	$17.36	$15.32 (run-rate)(1)
G&A cost per tonne (US$/t)	$32.43	$28.88 (run-rate)(1)
Total operating cost per tonne of mill feed (US$/t)	**$116.34**	**$106.17 (run-rate)(1)**
Sustaining capital per tonne of mill feed (US$/t)	$35.50	$19.26 (run-rate)(1)
Total cost per tonne of mill feed (US$/t)	**$151.83**	**$125.43 (run-rate)(1)**
Expansion capital expenditure ($m)	$177	
Sustaining capital expenditure ($m)	$218	
Total capital expenditure with closure costs ($m)	**$402**	
Cash cost per ounce AuEq ($/oz)(3)	$574	$554 (run-rate)(1)
All-in sustaining cost per ounce AuEq ($/oz)(4)	$716	$634 (run-rate)(1)
Cash cost per ounce gold ($/oz)(3)	$366	$313 (run-rate)(1)
All-in sustaining cost per ounce gold ($/oz)(4)	$545	$416 (run-rate)(1)
Base Case Economic Analysis at US$1,600/oz Gold, US$4.00/lb Copper and US$20.00/oz Silver		
After-tax NPV0%	$729 million	
After-tax NPV5%	$586 million	
IRR (%) and Payback Period (years)	N/A (Self-Funded)	
Economic Analysis at $2,000/oz Gold, US$4.00/lb Copper and US$20.00/oz Silver		
After-tax NPV0%	$1,051 million	
After-tax NPV5%(5)	$855 million	
IRR (%) and Payback Period (years)	N/A (Self-Funded)	

1. Run-rate excludes the final partial calendar year of production.

2. Excludes 2H 2024 commissioning and initial ramp-up stage.

3. Cash costs are net of by-product credits and are inclusive of mining costs, processing costs, site G&A and refining charges and royalties.

4. AISC includes cash costs plus estimated corporate general and administration costs, sustaining costs and accretion.

5. Net present value is calculated utilizing mid-year discounting.

1.14.1 Mining and Mineral Reserve Estimate

The FS 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 utilized 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 1.2 Mtpa Treatment Plant

Design criteria, process flow sheets and a mass balance for a 1.2 Mtpa Process Plant treating material from a copper-gold sulphide deposit have been developed. Capital Cost Estimates (Capex) and Operating Cost Estimates (Opex) were prepared for the Processing Plant, Secondary Back-up Power Station, and power reticulation.

The total installed capital estimate for the 1.2Mtpa Processing Plant is estimated to be US$80.2mil including a contingency allowance. A new standalone Power Station for the 1.2Mtpa Processing Plant is estimated to cost US$9.6M inclusive of contingency and a further US$13.8M is included for Power Distribution inclusive of contingency.

Conventional single stage crushing followed by a traditional SAB milling circuit was chosen in place of the current multistage crushing and ball mill circuit based on the test work and comminution modelling conducted during the study. The milling circuit includes flash flotation and a gravity circuit to capture free gold for smelting on site to produce gold dore.

Conventional sulphide flotation, thickening and filtering is employed to produce a high-grade concentrate which is loaded into shipping containers for transport to smelters.

The overall project schedule from project go-ahead until the first gold pour and project handover is scheduled for 23 months. An implementation schedule was prepared to support the duration of 23 months and is based on a critical path through the supply and installation of the mills.

1.14.3 Tailings Storage Facility

Tailings generated from the production expansion will be stored in the existing Tailings Storage Facility (TSF) and 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.

1.14.4 Paste Plant

To improve underground geotechnical stability, facilitate complete ore extraction and dispose an estimated 46% 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 Kora 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 is capable of servicing 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 Mining

Items for consideration are detailed in the following list:

- Optimization studies to increase NPV through scheduling and design improvements.
- Optimization study to consider ventilation design alternatives for surface raisebore versus underground adits to surface.
- Optimization of ore pass locations, and potential use of truck chutes at their termination points as opposed to rehandle with a loader.
- Optimization study to consider viability of underground and/or land conveyor suitability.
- 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 mineralized zones Judd, K1 and K2 and the FGZ.

- Additional geotechnical testwork is required for all lithologies to confirm FS 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.

1.15.2 1.2 Mtpa Treatment Plant

Geotech investigations are to be completed on the process plant location to confirm and optimize the foundation design parameters for the detail design.

1.15.3 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 Stage 3 (RL 530 m) 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.

1.15.4 Environmental Studies, Permitting and Social or Community Impact

The recommendations for environmental studies, permitting and social or community impact include:

- Investigate and pursue opportunities to accelerate the EIS process to minimize the potential for permitting delays, including commissioning long-lead studies.

- Continue engagement with MRA and CEPA to support efficient and timely project permitting.

- Continue to appropriately support the LTC determination of landholdings within the existing operations leased boundaries to minimize the potential for landownership disputes to delay project permitting and/or execution.

- Implement a comprehensive stakeholder engagement program focussed on supporting project permitting.

1.15.5 Paste Plant

Geotech drilling and investigations are to be completed at the 800 level to confirm and optimize the foundation design parameters for the detail design.

Binder constitutes over 50% of the variable component of the operating costs. At the time of compiling this work limited options were available for supply of Slag (the major binder input) to the Kainantu site. With further investigation considerable upside opportunity exists to reduce operating costs.

2.0 INTRODUCTION

This report is an Independent Technical Report dated 1 January 2022 of the Integrated Development Plan (IDP) for K92 Mining Inc.'s Kainantu Gold Mine Project (the 'Kainantu Project') in Papua New Guinea. The 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 2021, K92 Mining Inc. (K92) requested Entech Mining (Entech), MineFill Services (MineFill)), ATC Williams, Tetra Tech Coffey (Coffey), Metallurgical Management Services Pty Ltd (MMS) and Lycopodium Minerals Pty Ltd (Lycopodium) 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 and mine scoping studies of the Kora Expansion Project (the Project).

Entech were engaged by K92 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.

Lycopodium was engaged to carry out a feasibility study on the potential replacement of the existing processing plant to treat 1,200,000 tpa of ore from the Kora deposit. Including a financial model to provide guidance in relation to the economic viability of the Project.

MMS were engaged by K92 to manage the metallurgical testwork programme including interpretation.

MineFill Services were engaged by K92 to carry out a feasibility study on mine backfill options with a focus on a paste plant.

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

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

K92 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.

2.1 Basis of Technical Report

This Technical Report has been compiled by Lycopodium Minerals Pty Ltd (Lycopodium), Brisbane, Australia, from the sections prepared and signed off by the nine Qualified Persons (QPs – identified below), in order 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, 2, 3, 4, 5, 6, 7, 8, 9, 10, 11 and 19.

- Simon Tear (H&S Consultants Pty Ltd), responsible either wholly or partly for report Sections: 1, 14 and 26.

- Shane McLeay and Patrick McCann (Entech Mining), responsible either wholly or partly for report Sections: 1, 15, 16, 21, 24, 25 and 26.

- Sandra (Sandy) Hunter (Lycopodium Minerals Pty Ltd), responsible either wholly or partly for report Sections: 1, 13, 17, 18, 21, 22, 24, 25 and 26.

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

- Tara Halliday (Tetra Tech Coffey), responsible either wholly or partly for report Sections: 1, 20, 21, 25 and 26.

- Evan Kirby (Metallurgical Management Services Pty Ltd), responsible either wholly or partly for report Sections: 1 and 13.

- Matthew Helinski (MineFill), responsible either wholly or partly for report Sections: 1, 13, 18, 21, 24 and 25.

- Other sections have been provided by the Company.

2.2 Property Inspections by QPs

A summary of the QP site visits is detailed in Table 2.2.1.

Table 2.2.1 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
Simon Tear	21/10/18 – 23/10/18
Shane McLeay	N/A
Patrick McCann	14/10/21 – 21/10/21
Sandra (Sandy) Hunter	28/04/22 – 30/04/22
Ralph Holding	2004
Tara Halliday	N/A
Evan Kirby	09/06/22-12/06/22
Matthew Helinski	N/A

2.3 Effective Dates

The Effective Date of this report is 1 January 2022. There were no material changes to the scientific and technical information of the Project between the Effective Date and signature date of this Report.

2.4 Abbreviations

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	Bilimaio interim landowner association
BLA	Bilimaio 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
F_{80}	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
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
Mtpa	million tonne per annum
NAF	non-acid forming
NAG	net acid generation
NPV	Net Present Value
NQ	Exploration drill size (75. 5mm 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
WSF	Water Storage Facility
μS/cm	micro Siemens per centimetre
μm	Micron (1/1000 of a millimetre)

3.0 RELIANCE ON OTHER EXPERTS

The author of this report is 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.

4.0 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 4.1.1). 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 five exploration licences (EL470, EL693, EL1341, EL2619 and EL2620), one mining licence (ML150), two mining easements (ME80 and ME81), and one licence for mining purposes (LMP78), plus one exploration licence application (ELA2753). Tenements are owned 100% by K92 Mining Limited (K92ML) but there is an understanding in-place for a 5% share (ML150 only) to be divested to the local landowners and the Eastern Highlands Province. Further information on this understanding is detailed in Section 20.3.6 Memorandum of Agreement (MOA). A tenement map is shown in Figure 4.1.1 and tenement details are summarised in Table 4.1.1. The Project as described herein is 100% owned by K92 Mining Limited (K92ML); a company incorporated in Papua New Guinea, which is 100% owned by K92 Holdings (PNG) Limited (K92PNG), a 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 on 23 January 2015 for a period of 10 years to 13 June 2024. 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), each effective until 13 June 2024.
- Licence for Mining Purposes 78 (LMP78), effective until 13 June 2024.
- Exploration Licence 470 (EL470), effective until 4 February 2023.
- Exploration Licence 693 (EL693), effective until 4 February 2023.
- Exploration Licence 1341 (EL1341), effective until 20 June 2020. K92ML have lodged an application for renewal for a further two years.

- Exploration Licence 2619 (EL2619), effective until 22 January 2022. K92ML have lodged an application for renewal for a further two years.
- Exploration Licence 2620 (EL2620); effective until 2 June 2023.

The renewal of ML150, ME80, ME81, and LMP78 occurred immediately prior to the acquisition of K92ML by K92PNG.

Figure 4.1.1 Kainantu Project Location and Tenements

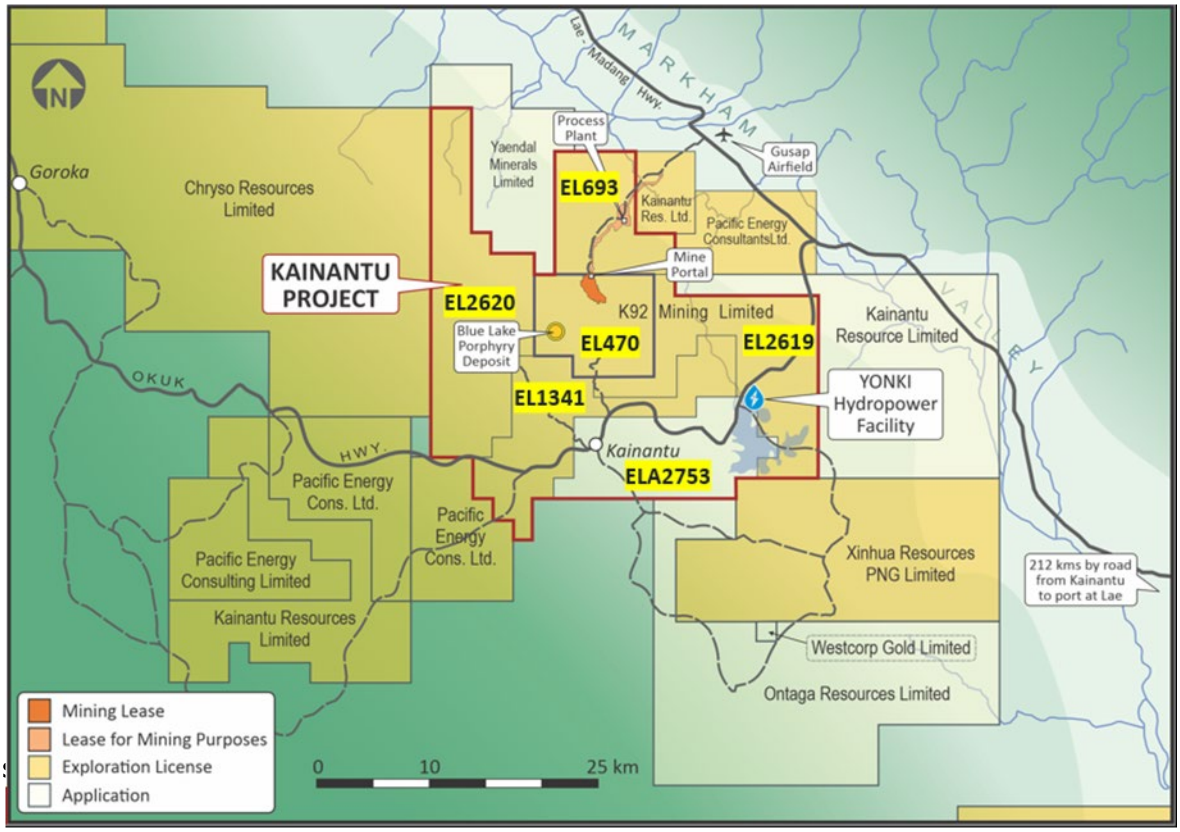


Table 4.1.1 Project Tenure Details

Tenement No.	Grant Date	Expiry Date	Renewal Application Date	Area (km^2)	Owners[#]
EL470	05/07/1982	04/02/2023	Current	92.65	K92ML
EL693	29/12/1986	04/02/2023	Current	95.45	K92ML
EL1341*	21/06/2004	20/06/2020	11/03/2020	146.63	K92ML
EL2619	23/01/2020	22/01/2022	Current	320.54	K92ML
EL2620	03/06/2021	02/06/2023	18/01/2022	201.19	K92ML
ML150	14/06/2002	13/06/2024	Current	2.88	K92ML– 95% Landowners– 5%**
ME80***	14/06/2002	13/06/2024	Current	0.30	K92ML
ME81***	14/06/2002	13/06/2024	Current	0.35	K92ML
LMP78***	14/06/2002	13/06/2024	Current	2.09	K92ML

Source: K92ML (2022)

*** EL1341 are pending renewal as of 21.12.2021**

**** 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 4.2.1 Expenditure Commitments

Tenement No.	Term End Date	Commitment Year 1* (PGK)	Commitment Year 2* (PGK)	Proposed Work Program Budget	
				Unit	Amount
EL470	04/02/2023	2,400,000	2,400,000	PGK	4,800,000
EL693	04/02/2023	500,000	500,000	PGK	1,000,000
EL1341	20/06/2022	600,000	600,000	PGK	1,200,000
EL2619	22/01/2022	120,000	180,000	PGK	300,000
EL2620	02/06/2023	100,000	150,000	PGK	250,000

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.

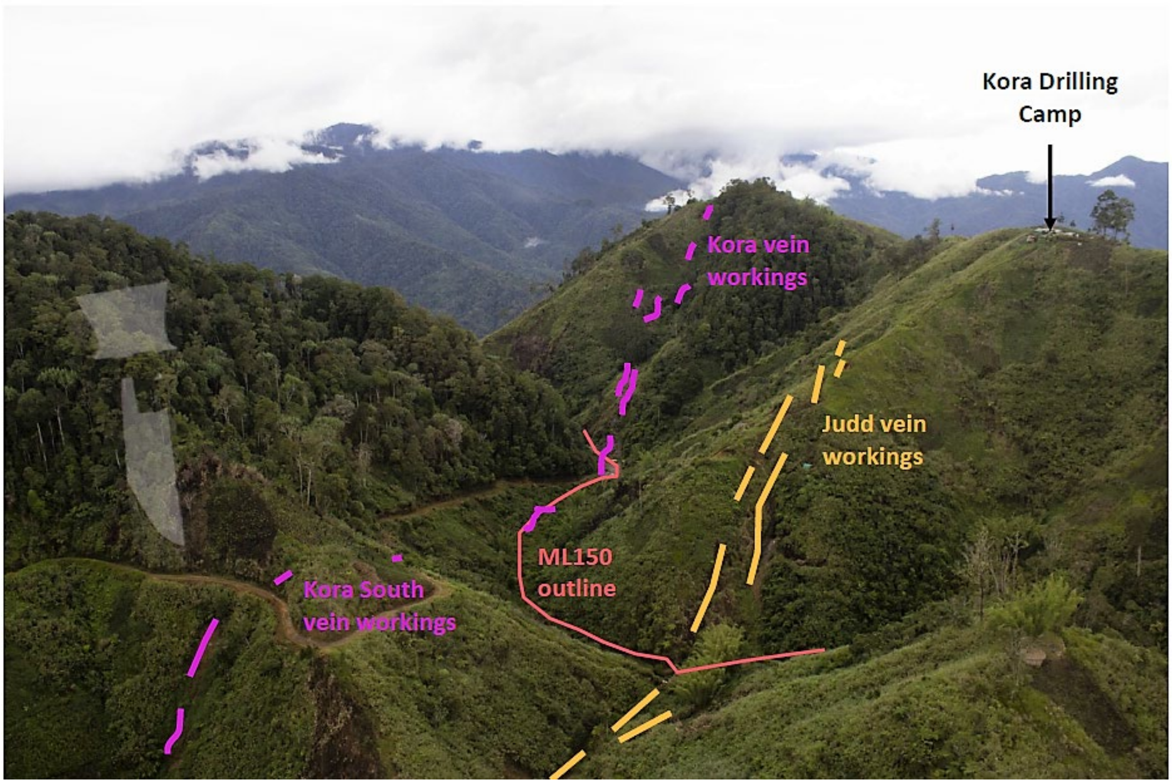
Environmental permitting, mine closure plans, and landowner compensation agreements are discussed in Section 20: 'Environmental Studies, Permitting, and Social or Community Impact' of this report.

5.0 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. Elevations range from 400 m to 1,900 m above sea level. Vegetation is mostly primary rainforest with areas of shifting cultivation in valley floors.

Figure 5.1.1 Kainantu Project Topography



Source: K92ML (2021)

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 / month during the November to April wet season, and 137 mm / month during the dry season. Annual rainfall averages approximately 2,000 mm. Project operation/exploration is subject to the weather; reduced visibility when cloudy prevents operation of helicopters and heavy rainfall or earthquakes can trigger 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 use in emergencies and for charter flights.

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 also 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 K92 in October 2016. In February 2018 K92 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.

6.0 HISTORY

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. There has been a significant amount of exploration on the property by various owners. The operation was on care and maintenance between January 2009 and August 2016 when K92ML commenced rehabilitation of the mine and processing plant.

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 the deposit. Under K92ML, Kora and Eutompi deposits became a part of the up-dip extension to the near surface of Kora Consolidated.

HPL Mineral Resources, reported in accordance with the 2004 JORC Code and Guidelines, were prepared by independent consultants 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. K92 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 MRE for the Kora and Judd deposits which was 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.

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 6.2.1 shows mill production for the life of the mine from 2006 to its closure in 2008. On a qualitative basis a negative reconciliation on grade from grade control to mill production is evident. 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 6.2.1 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 months)	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 2020

K92ML restarted mining operations in the Irumafimpa mining area. Limited mining activities were undertaken in the lower parts of Irumafimpa during 2017, with mineralised material being mined from development headings and from stopes. A small amount of low-grade ore was also recovered from remnant stopes. Table 6.4.1 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 K92 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.

On 13 January 2022, the Company announced record quarterly production in the fourth quarter of 2021 at Kainantu Gold Mine, of 36,145 oz AuEq, or 33,220 oz of gold, 1,048,100 pounds of copper and 28,818 oz of silver. Annual production also achieved a record of 104, 196 oz AuEq or 95,055 oz gold, 3,375,528 lbs copper and 70,792 oz silver.

Table 6.4.1 Kainantu Mill Production 2016 to 2021

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

**** 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 6.4.2.

Table 6.4.2 Summary of Operations Timeline for the Kainantu Project

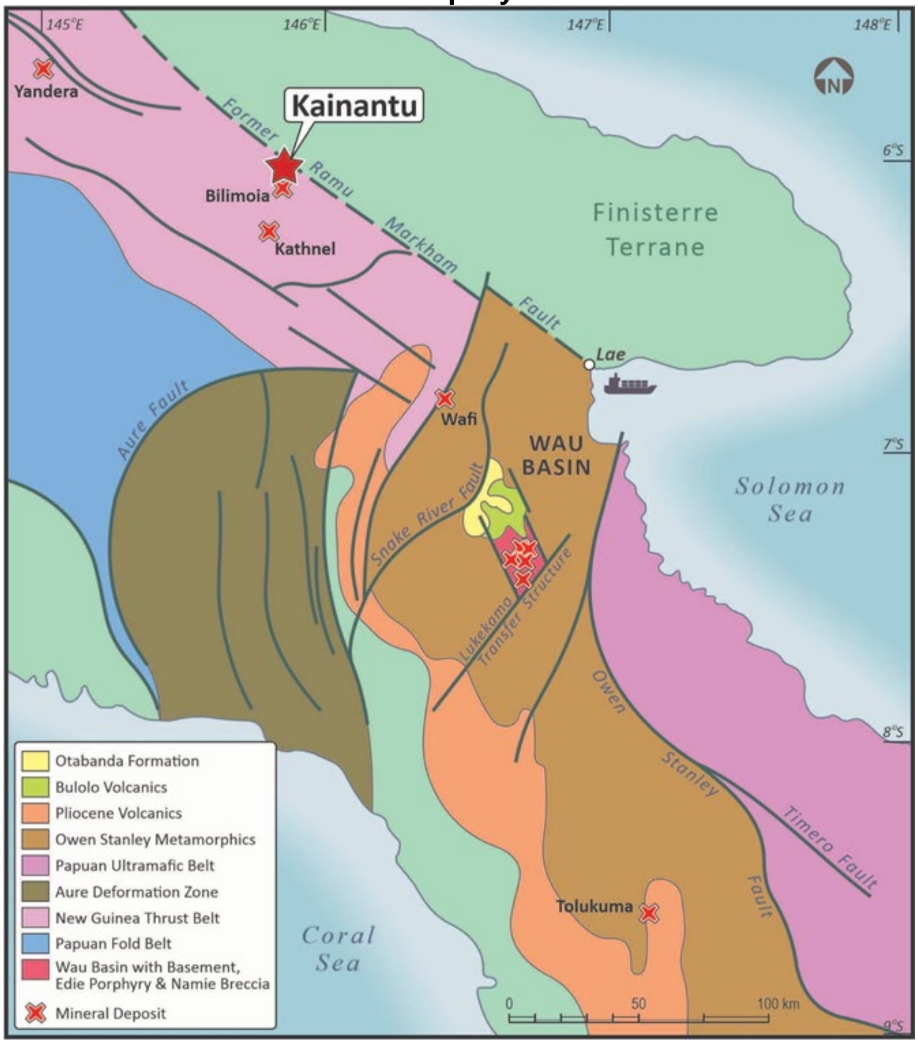
From	To	Mine Operations History (ML150)
January 2004 2005	October 2007	Highlands Pacific DFS approved by Mineral Resources Authority. Kainantu Gold Mine operated by Highlands Kainantu Limited (HKL).
November 2007 January 2008	December 2008	Barrick purchased the Kainantu project. Mining was suspended 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 11 June 2014 which closed 6 March 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, 1235 Drive commences as a vent drive, followed by drilling from underground on the Judd Vein system.

7.0 GEOLOGICAL SETTING AND MINERALISATION

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 7.1.1). 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).

Figure 7.1.1 Regional Geology of Papua New Guinea, Showing Location of Kainantu Property



Source: K92ML (2021)

7.2 Property Geology

The Kainantu area is underlain by rocks of the Early Miocene Bena Formation, comprising pelite, psammite, conglomerate and marl beds metamorphosed to greenschist to amphibolite grade (Table 7.2.1). 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 dykes form small high level crowded feldspar porphyry dykes and diatreme breccias associated with gold and copper mineralisation. A north-northeast trending transfer structure transects the area, with associated mineralisation, alteration and porphyry complexes aligned along it.

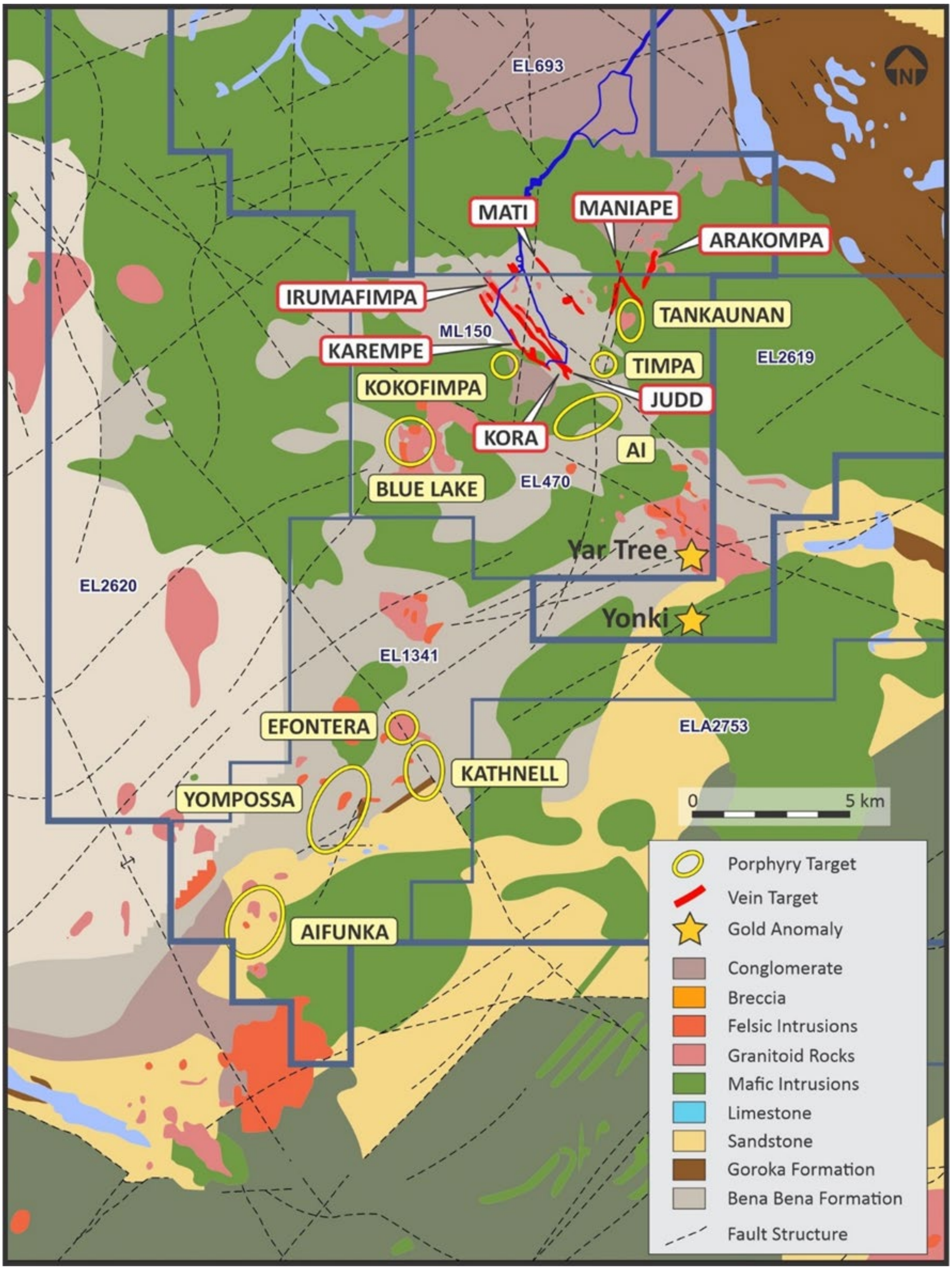
Table 7.2.1 Main Regional Rock Units Identified Within the Kainantu Area

Age	Rock Units
Recent Quaternary	Kainantu Formation – basal fluvial conglomerate, 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 mineralisation accompanying isoclinally folded faults and breccias.

7.3 Mineralisation Overview

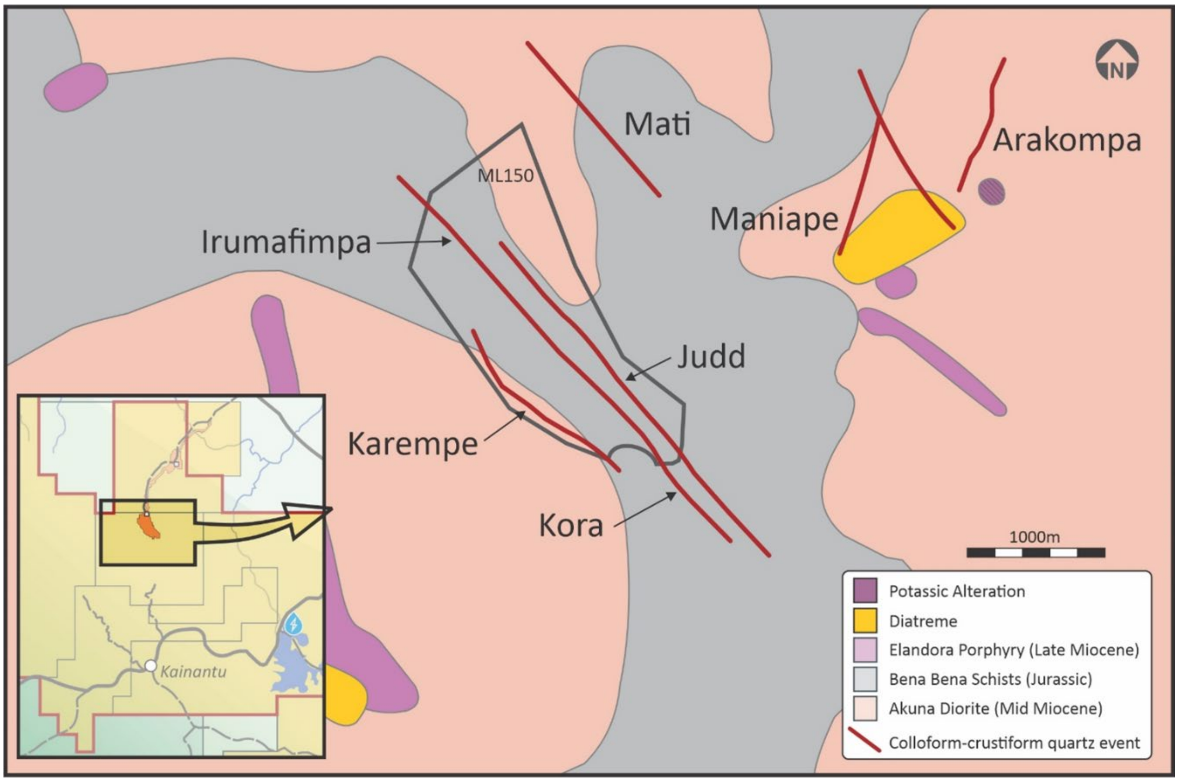
Dominant host rock of the Kora Consolidated-Irumafimpa vein systems is the highly sheared and deformed Bena Bena formation, composed of low grade metamorphosed phyllites and amphibolites, intruded by the Elandora porphyry at the northwest end of the vein system. Mineralisation on the property includes gold, silver, and copper in low sulphidation epithermal Au-telluride veins (Irumafimpa), Au-Cu-Ag quartz-sulphide veins of Intrusion Related Gold Copper ('IRGC') affinity at Kora Consolidated, indications of porphyry Cu Au systems (Blue Lake) and alluvial gold.

Figure 7.3.1 Kainantu Property Geology and Known Vein and Porphyry Deposits and Prospects



The Kora Consolidated-Irumafimpa mineral zone occurs in the centre of a large mineralised 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 mineralisation. The Kora Consolidated-Irumafimpa (including Kora, Eutompi and Kora North) vein deposits have been demonstrated from drilling results and underground development to be a continuous mineralised structural system.

Figure 7.3.2 Location Plan Kora, Irumafimpa and Judd Vein Systems



Source: K92ML (2021)

The Kora Consolidated mineral zone 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 mineralisation between K1 and K2 known as the Kora Link lode and about 90 to 150 m to the east lies the parallel Judd lode. The Judd lode (J1) as well as J2 to J4 mineralised zones appear to be a vein system similar to the Kora Consolidated zone, but as yet has not been fully defined by drilling.

Kora Consolidated vein system is a broad northwest trending mineralised zone more than 2.5 km long and up to 60 m wide in which individual veins vary from less than one metre wide with pinch and swell features over short distances (Au telluride lodes at Irumafimpa) to more continuous veins up to several metres wide (Au, Cu – rich quartz and sulphide lodes, IRGC).

Other less advanced prospects on the property include epithermal Au veins similar to Irumafimpa, IRGC veins similar to Kora, porphyry Cu-Au systems (Blue Lake), skarn Cu, Pb and Zn mineralisation and alluvial gold.

Mineralisation style, host rocks, dimensions and geological continuity for the Kora-Irumafimpa vein deposit and the other vein and porphyry prospects for the Kainantu Project are described in the 'Independent Technical Report, Mineral Resource Estimate Update, Kora and Judd Gold Deposits, Kainantu project, Papua New Guinea', effective date 20 January 2022 which is filed on SEDAR.

7.4 Kora Consolidated-Irumafimpa Vein Systems

The Kora Consolidated-Irumafimpa vein systems is interpreted to contain two stages of mineralisation (Corbett, 2009, his work used the Irumafimpa mine working exposures and Barrick's Kora and Eutompi drill results). The earliest is a sulphide-rich Cu-dominant stage. 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), reaching a vein intrusive style of mineralisation at Kora Consolidated containing Au and Cu mineralisation with minimal tellurides. The alteration and mineralisation paragenesis recognised in the Kora-Irumafimpa vein system is summarised below in Table 7.4.1.

Table 7.4.1 Mineralisation 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 mineralisation comprises coarse-grained idiomorphic quartz and pyrite (typically euhedral) veins with base metals. Volumetrically this early mineralisation appears to be the most abundant mineralisation. At Kora Consolidated the mineralisation comprises massive pyrite veins to pyritic breccias, grading to pyrite-chalcopyrite-bornite veins characterised by elevated Zn, Pb, Sn, W, Bi, and Sb in an intrusive vein style mineralisation. Higher copper grades, in the order of 2% Cu, occur at Kora (Coote, 2018; Muller et al., 2019). There appears to be a lateral zonation to lower copper grades at Irumafimpa.

Stage 3 mineralisation 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. 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 Consolidated. Kora Consolidated is characterised as a vein intrusive style mineralisation containing minimal tellurides.

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

7.5 Host Rocks

Dominant host rock of the Kora Consolidated-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 northern end of the vein system.

7.6 Controls

The structural history of the Kora Consolidated-Irumafimpa area has been documented by Blenkinsop (2005), this body of work was done prior to the discovery of Kora North. The Kora Consolidated-Irumafimpa vein systems are made up of the contiguous Kora, Eutompi, Kora North and Irumafimpa structures. Veins are breccia veins with abundant clasts of both altered wall rock and earlier stages of vein mineralisation, 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 mineralisation Table 7.4.1.

At Kora Consolidated both the sulphide copper dominant and quartz gold dominant mineralisation occur along the same NW trending sub-vertical structure. This is likely to be a long-lived structure, which was reactivated at several different stages in time. The quartz gold dominant mineralisation shows modest variations in dip (from sub-vertical to locally -60° dip) and strike this believed to be due to the formation of dilation from pre-existing structures that later was enhanced by structural dilatancy from continued mineral fluid injection.

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

7.7 Dimensions and Continuity

The current mineralisation is part of a broad northwest trending mineralised zone more than 2.5 km long and up to 60 m wide in which individual veins vary from less than one metre wide with pinch and swell features over short distances (Au telluride lodes at Irumafimpa) to more continuous veins up to several metres wide (Au, Cu – rich quartz and sulphide lodes) and 100's of metres of geological continuity at Kora.

Historical and more recent exploration has identified and subdivided several shoots within the lodes, defining the Kora, Kora North, Eutompi and Irumafimpa deposits. Minor modifications to the lode margins have been interpreted from the 2020-2021 drilling results with the southern phasing out of K1 at depths below the 1205 level, and the extended continuity of the K2 lode, which remains open to the south at both shallow and deeper levels. The interpretation has given K2 the dominant role for the vein system. The Kora Link lode has been re-interpreted as an amalgamation of smaller mineralised veins into a single zone juxtaposed with both K1 (on its hanging wall) and K2 (on its footwall).

The Kora Consolidated comprises two parallel, steeply west dipping, north-south striking quartz-sulphide vein systems, K1 and K2, within an encompassing dilatant structural zone hosted by phyllite. An additional structure, the Kora Link, has also been defined for part of the area between K1 and K2. The current Kora Consolidated resource estimate area covers an area of approximately 1,250 metres along strike by 1,050 to 1,150 m vertically. Kora Consolidated is along strike to south from the previously mined Irumafimpa deposit.

The Geological continuity of the lode system is excellent, whereas the gold grade has variability as is typical of this style of mineralisation, a drill spacing of 25 to 30 m centres identifies with confidence to Indicated Resource category status.

7.8 Judd Vein System

The Judd lode is a narrow, (4.9 m average width in the MRE 2022) intrusive related quartz-sulphide, Au-Cu-Ag vein system, similar to Kora Consolidated located approximately 90 to 150 m east of and parallel to Kora Consolidated on ML150. The system consists of four known narrow veins with significant inter-vein separation, steeply dipping and with a similar strike direction as Kora. The vein of most interest is the western vein, J1 and at the close of date of the MRE 2022, this has been subject to development drives on two levels (1,235 and 1,265 level) totalling 692 m and 197 face sample lines for 1,478 samples. This work indicated significant geological continuity with variable grade continuity, significant mineralisation with potential for economic extraction, material from the development drives was treated by the process plant and achieved similar recoveries for gold and copper to that achieved with Kora material. K92ML has carried out diamond drilling from both underground and surface to define part of the Judd J1 vein system in the development area. Underground diamond drilling has concentrated on the J1 vein, with some holes being extended to intersect the other veins (J2, J3, and J4) previously documented by Barrick and HPL and which remain under-explored.

The Judd vein system was partially tested by Barrick surface holes that were drilled to test the Kora Consolidated 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 mineralisation, a quartz dominant, Au-rich component, and a sulphide dominant, Cu-rich vein style. Surface mapping and sampling has indicated a mineralised strike length of over 2.5 km.

7.9 Host Rocks

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.

7.10 Controls

The Judd veins are intrusive vein style and are typically brecciated with abundant clasts of both altered wall rock and earlier stages of vein mineralisation. The veins are tectonic in nature and are sometimes vuggy in places.

The structures containing the quartz sulphide mineralisation at Judd strike NNW and dip at 75 to 80 degrees to the west. This is likely to be a structure, which was reactivated at several different stages in time. The quartz gold dominant mineralisation shows modest variations in dip and strike and is believed to be due to the formation of dilatancy from pre-existing structures that later were enhanced by continued mineral fluid injection. Minor late-stage faults sometimes display the mineralisation approximately subparallel to the mineralisation.

7.11 Dimensions and Continuity

The dimensions of the wireframe interpretation used in MRE are 800 m in strike length and 500 m to 700 m in the dip direction. The Judd vein system is interpreted where it has been sporadically intercepted in diamond drill holes from surface and underground in the ML150 and passing into EL470 to the south for a total distance of 2.5 km. This shows potential that is yet to be developed.

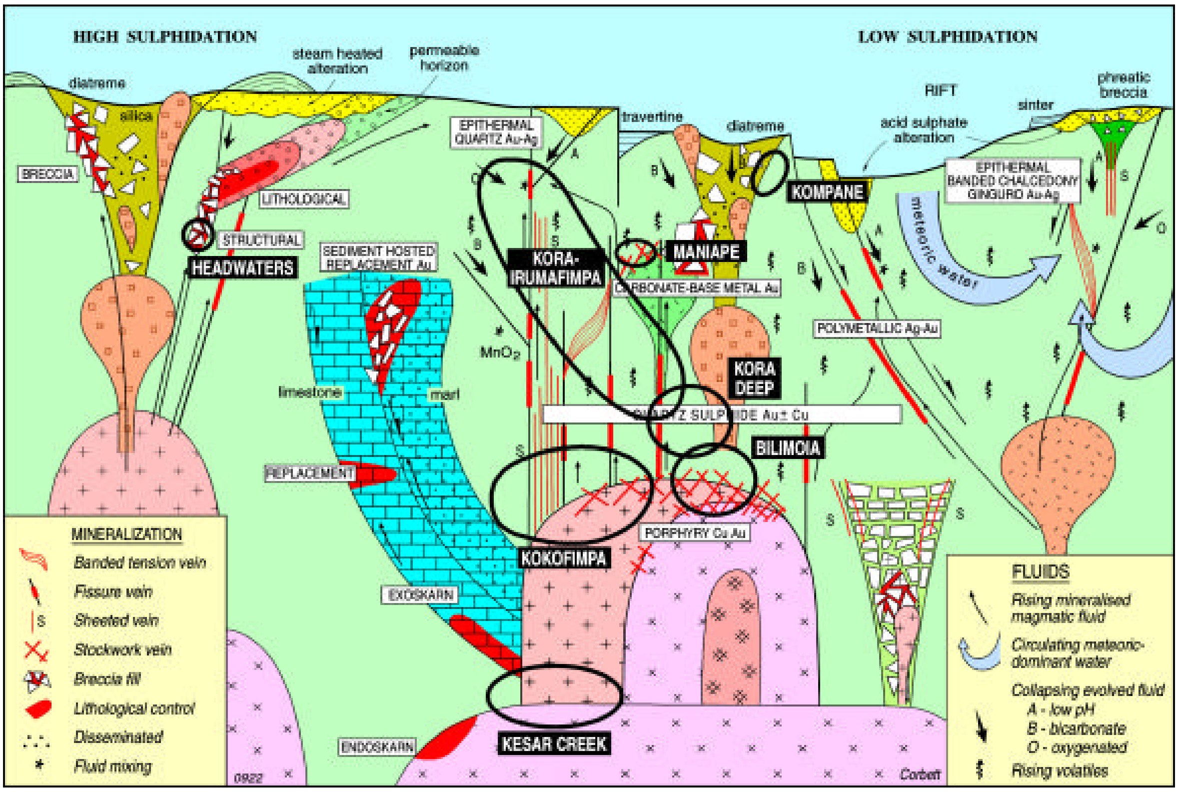
The J1 gold grade continuity and mineralisation pinches and swells along the drive. The gold grades vary with ranges of similar grades typically between 5 up to 30 m along the drive, the variability is usual for this type of mineralisation.

8.0 DEPOSIT TYPES

Gold-copper deposits within the SW Pacific Magmatic Arcs have been classified into groups by Corbett and Leach (Corbett and Leach, 1997, Corbett, 2009):

- Porphyry-related (including gold skarn).
- High sulphidation gold-copper. PFS NI43-101 Section
- Low sulphidation (including sediment-hosted replacement).
- Intrusion related gold deposits.
- Carbonate-base metal Au deposits.

Figure 8.0.1 Conceptual Model for Porphyry and Related Low and High Sulphidation Mineralisation



Source: Modified by K92ML after Corbett 2009

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 8.0.1.

9.0 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 1 Mtpa 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 2015-2021

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 Drive. Following discovery of the Kora North mineralisation and the delineation of an initial Mineral Resource, the emphasis of K92ML's drilling aimed to extend the Kora North mineralization above and below that 2018 Mineral Resource. Drilling in conjunction with underground mining has continued in this area with expansions to the mineral resource documented in the 2022 NI43-101 report 'Independent Technical Report, Mineral Resource Estimate Update, Kora and Judd Gold Deposits, Kainantu project, Papua New Guinea', effective date 20 January 2022. Drilling since 2020 has concentrated on infilling the Resource to upgrade to Measured and Indicated, a minor amount of drilling has been used to extend the mineralization at depth to the south of the 2020 mineral resource estimate.

K92ML's exploration team has prioritised several targets (listed below in Table 9.2.1) for follow-up work.

Table 9.2.1 K92ML Priority Exploration Targets 2021

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)

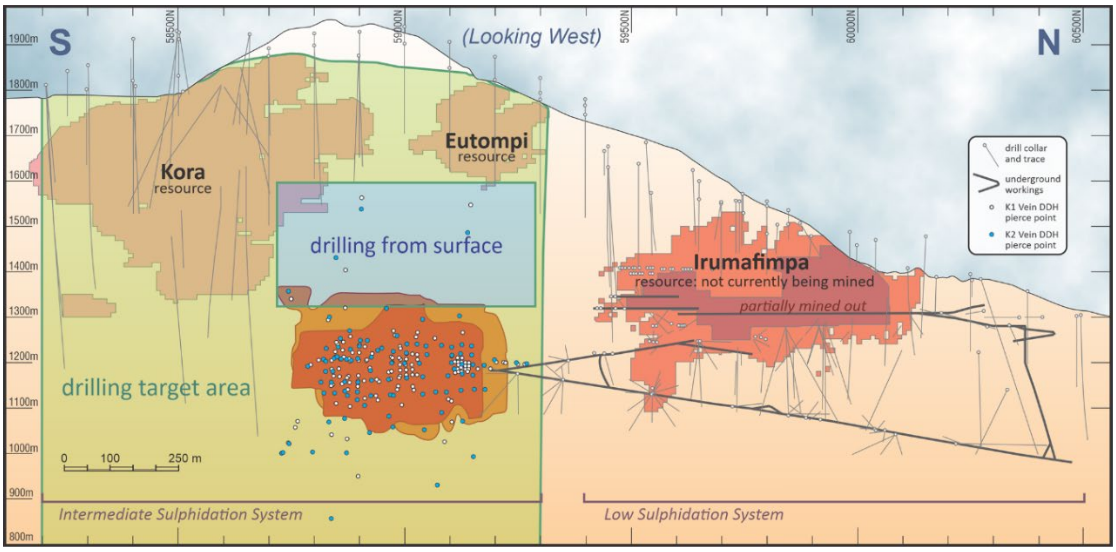
In May 2017, K92ML commenced underground diamond drilling at Kora North. Diamond drill hole, KMDD0009, intersected what K92ML interpreted as an extension of the Kora / Eutompi vein system approximately 130 m from the Kora Drive. The intersection of 5.4 m at 11.68 g/t gold, 25.5 g/t silver and 1.33% copper 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 1 Mtpa from the Kora Gold Deposit, Kainantu Project, Papua New Guinea, dated 27 July 2020 which are 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 in 2020 and a consolidation and upgrade of the Mineral Resource for Kora Consolidated and a maiden Mineral Resource for the J1 Judd lode.

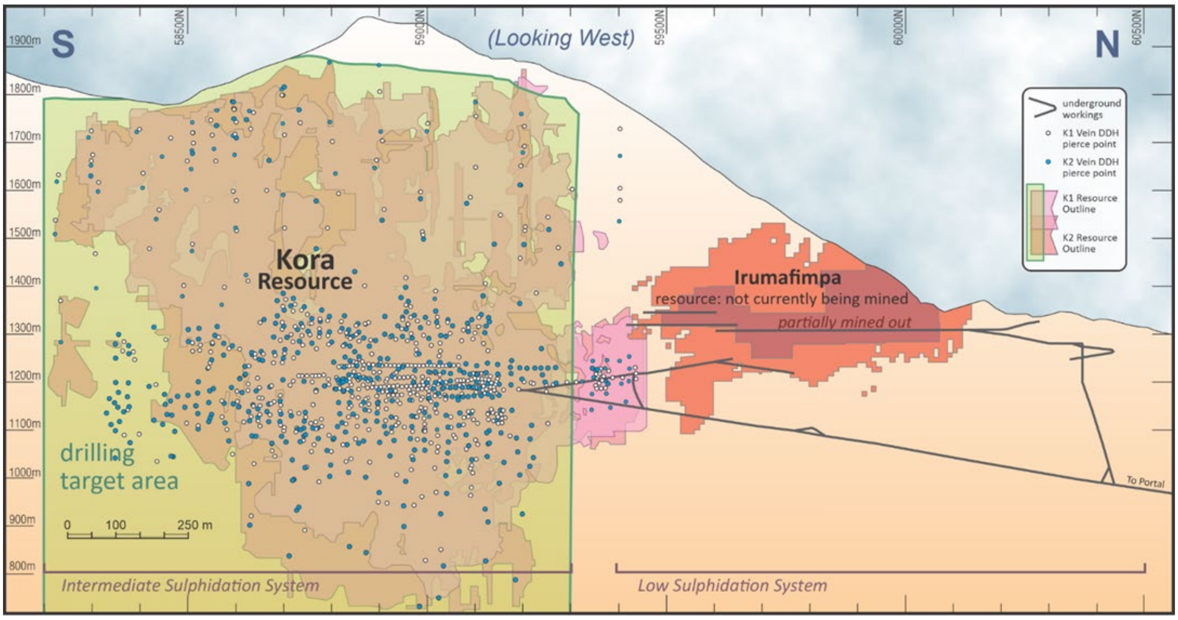
Figure 9.3.1 is a schematic longitudinal section showing the location of the Kora diamond drilling as of April 2020 at the time of the previous (2020) MRE and Figure 9.3.2 shows the Kora drilling as of October 2021. Drilling was designed to be of sufficient density to raise confidence in the resource classification.

Figure 9.3.1 Mine Lease Long Section - Irumafimpa, Kora and the Drilling Target Area as of the Resource Estimate April 2020



Source: K92ML (2020)

Figure 9.3.2 Mine Lease Long Section - Irumafimpa, Kora, Drilling Target Area and Total Intercept Pierce Points as of October 2021



Source: K92ML (2021)

10.0 DRILLING

10.1 Kora Consolidated / Judd- Drilling

The updated Mineral Resources for Kora and Judd detailed in the 'Independent Technical Report, Mineral Resource Estimate Update, Kora and Judd Gold Deposits, Kainantu project, Papua New Guinea', effective date 20 January 2022 are based on diamond drill hole samples from both surface and underground drilling along with face sampling of underground development drives.

Table 10.1.1 provides summary details of the sampling for the overall deposit area. The majority of the recent K92ML drilling (March 2020 – Dec 2021) has been infill drilling to increase the amount of Measured and Indicated Resource for the mineral resource estimate. The drilling has also achieved some resource expansion by extending the Kora mineralisation along strike to the south. Drilling into Judd was aimed at both expanding and defining the Mineral Resource from underground, after confirming economic mineralisation while installing a vent drive along the J1 Lode.

Table 10.1.1 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	9,564.20	115.2
			Face	461	1,499.30	3.3
K92ML	Oct 2018 - Feb 2020	Surface	DD	16	7,390.00	461.9
			DDwedge	3	719.7	239.9
		UG	DD	96	21,224.50	221.1
			DDwedge	7	935.2	133.6
			Face	312	1,657.50	5.31
K92ML	Mar 2020- Oct 2021	Surface	DD	21	6,155.10	293.1
		UG-Kora	DD	231	36,152.64	156.5
			DDwedge	3	471.7	157.2
			Face	509	2,798.59	5.5
	Mar 2020-Dec 2021	UG_Judd	DD	49	8,935.60	182.4
			Face	193	1,059.84	5.5
Kora Total Drilling				**569**	**110,707.04**	
Kora Total Face				**1,282**	**5,955.39**	
Grand Total				**618**	**119,642.64**	
Grand Total Face Sampling				**1,475**	**7,015.23**	

(DD = diamond drilling; UG = underground)

Drilling and sampling methods are described in detail in the 'Independent Technical Report, Mineral Resource Estimate Update, Kora and Judd Gold Deposits, Kainantu project, Papua New Guinea', effective date 20 January 2022 which is filed on SEDAR.

11.0 SAMPLING 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 transported to the core yard, a centre line marked on the core for reference for sampling purpose to avoid bias by the geologist, half core is sampled, and the other half retained in the tray. Mineralized zones were sampled as well as 5m either side of the mineralized zone. Maximum sample interval was 1m and minimum 0.5m. Intervals were determined by a qualified Geologist on geological mineralized boundaries. This included 5 at 1 metre intervals either side of the mineralization. The entire half-core sample is to be to be dried in a monitored oven, so the heat remains between 95°C and 113°C. After 10 hours, the samples were removed from the oven and allowed to cool and reduced to minus 75 micron with LM5 before the assay portion is split off. All samples are to be analysed at Astrolabe's Madang Laboratory. Gold was determined by 50 g fire assay and other elements such as Cu, Pb, Zn and As by acid digest, AA finish.

11.2 Barrick

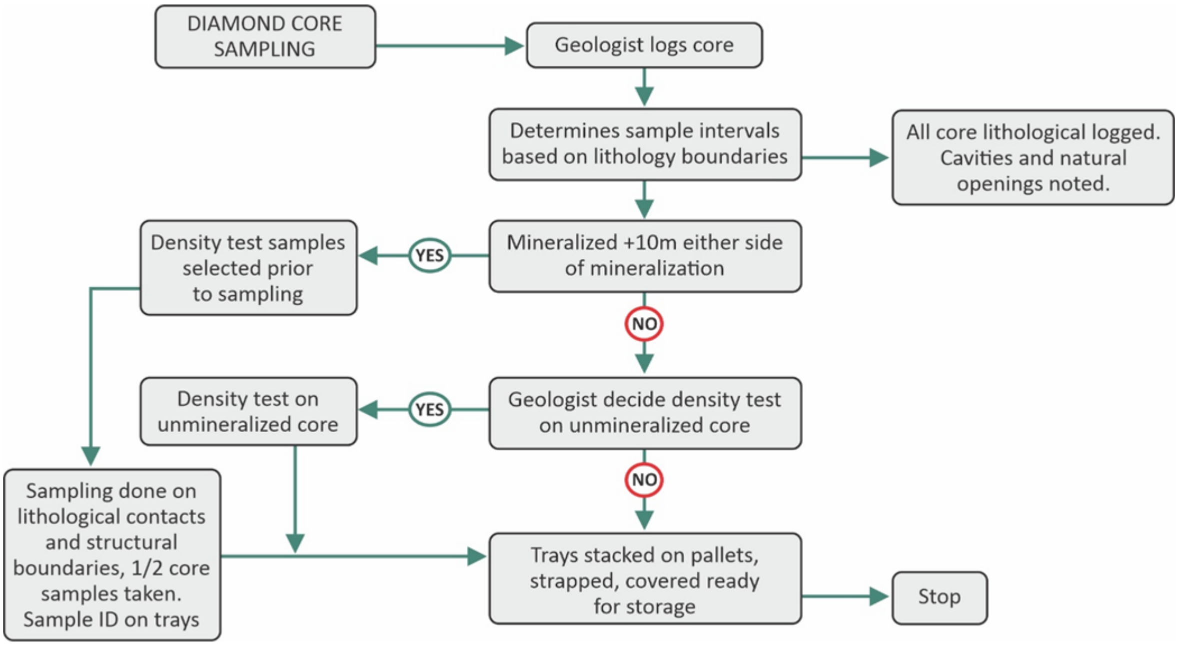
The drill core handling procedures used by Barrick were reported by Nolidan Mineral Consultants in 2015. His findings are as follows: "All drill core was logged, photographed (wet and dry), then cut and sampled at Barrick's Kumian core yard. Logging data entry was completed using an in-house developed 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 is presented in Figure 11.3.1.

Figure 11.3.1 Diamond Core Sampling Process Flow Diagram



The core tray labelling and installation 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. Sample intervals were guided by geological contacts / boundaries. A minimum sampling width of 0.1 m and a maximum of one metre were used. The smaller sample intervals were utilized to sample individual sub-veins / stringers and sulphide intercepts. Core was sampled as a minimum to greater than 5 m 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 lodes was usually left unsampled. All mineralized occurrences were 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 remainder 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 for dispatch to the on-site assay laboratory managed by Intertek Testing Services (PNG) LTD, an independent accredited laboratory.

K92ML has a standard underground face sampling procedure in place. Face samples, under geological control, are taken across the full face of both the exposed lode system and any waste rock, with sample intervals ranging from 0.1 to 1 m in width depending on what the geologist decided. Two samples are taken per interval at waist and knee height and the corresponding widths recorded. Sample lengths are <1.5 m with samples approximately 3.5 kg in size. 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. Two samples are taken per interval with the samples 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 October 2019).

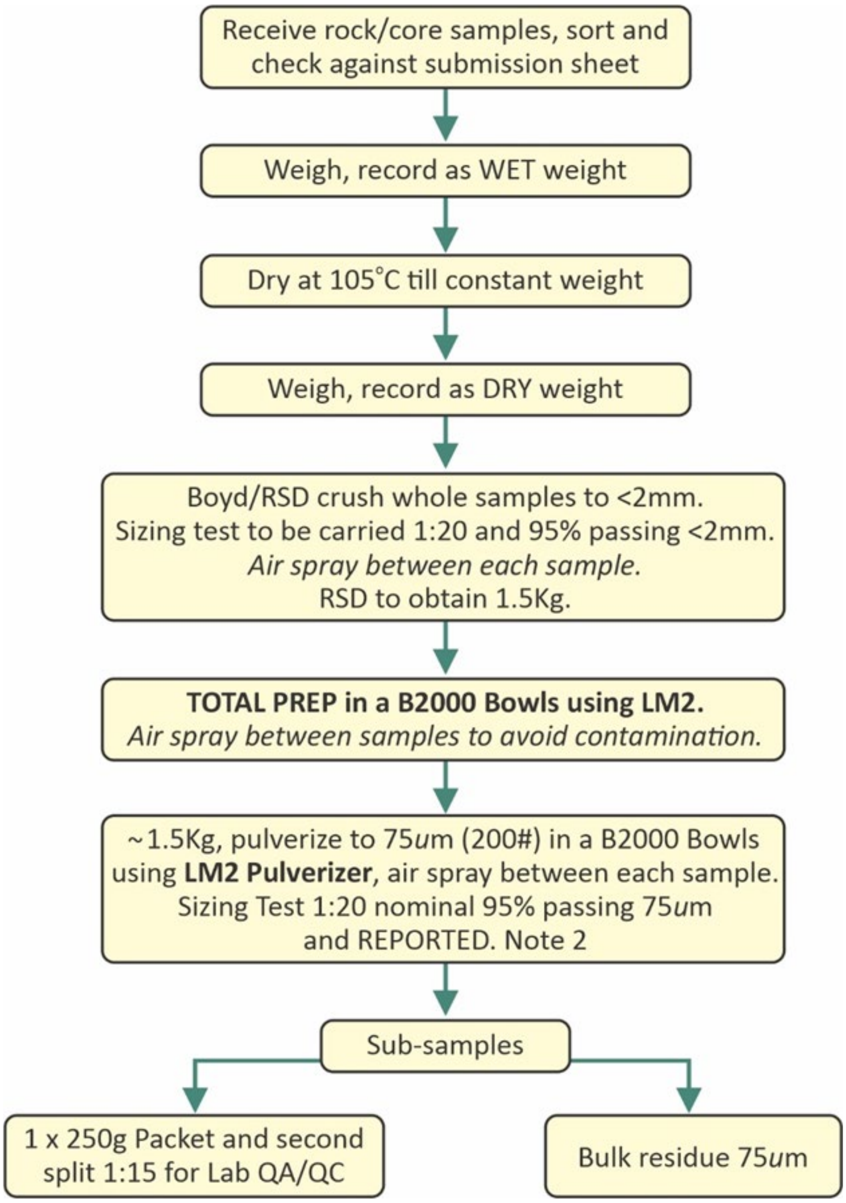
A twin hole programme was completed to confirm the Barrick assays as no QAQC data could be found from their drilling.

11.3.2 Sample Preparation

Intertek Laboratory services provides on-site laboratory facilities. The sample preparation for both the drill core and face samples is described below and as a flow diagram presented in Figure 11.3.2.

- Samples are sorted and dried at 105ºC overnight.
- Jaw crushed to 5 mm.
- Secondary jaw crush using a Boyd crusher to 2 mm and then rotary split to give 1 kg.
- Pulverization using an LM2 mill of the 1 kg sample to 90% passing 106 microns.
- Duplicate splits 1 in 30.

Figure 11.3.2 Sample Preparation Procedure



11.4 Density Measurements

11.4.1 Historic Density Measurements

Previous density test work 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).

Density testing between October 2018 and March 2020 were summarized 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 1 Mtpa 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.

11.4.2 Current Density Measurements

The K92ML density determination method of 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, dried 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 and moisture content and density calculated. Trials, of using cling film, to wrap the core prior to submersion in water were abandon as this tends to buoy the sample giving less weight to the sample. Typically, moisture content was less than 1% by weight. Results of the density tests were entered into the Access database and using the Surpac software the density results were extracted as sample points from the database within each of the lodes, i.e. K1, K2, Kora Link and J1. Results of the density testing statistics are summarized in Table 11.4.1 for the four lodes. As a result, default density values were assigned to each lode, with the value 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.

Table 11.4.1 Summary of Density Data for Each Lode

Descriptive Statistics	K1	K2	Kora Link	J1
Mean	2.84	2.84	2.74	2.71
Median	2.73	2.71	2.58	2.63
Standard Deviation	0.44	0.44	0.41	0.33
CV	0.15	0.16	0.15	0.12
Minimum	2.20	2.29	2.31	2.32
Maximum	4.71	5.27	4.80	3.80
Count	564	527	222	77

11.4.3 Sample Analysis

The analytical method is detailed as follows:

- 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).
- Cu/Ag:
  - 3 acid digest at 180°C (Nitric, Perchloric and Hydrochloric mix)
  - Diluted with water and read by AAS.
- CS:
  - Read by combustion furnace.
- Fluorine:
  - Following carbonate fusion samples are read on a specific ion meter referenced against standards.

11.4.4 QAQC Programme and Results

Since its acquisition of the project in 2016 K92 has implemented a QAQC programme. K92ML's documented QAQC programme for the drilling comprises standards, blank standards, laboratory duplicates, and second laboratory checks. The face sampling prior to September 2018 had no QAQC samples inserted although the sample suites were inserted in between drill hole sampling suites. From September 2018 onwards the same QAQC protocols were applied to the face sampling.

The K92 2018 QAQC programmes comprised the use of standards including blank samples, certified reference materials, 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 improved dust extraction system in the sample preparation location in following year.

The K92 2018-2020 QAQC programmes comprised the use of standards including blank samples, certified reference material, laboratory duplicates and second laboratory check assays.

The K92 2020-2021 QAQC 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 four pairs that was completed by K92ML in 2021 with variable and unbiased set of results, twinning previous historical surface drilling.

11.4.5 QAQC Summary

The QAQC programme has included the use of a range of CRMs for gold, copper and silver, laboratory duplicates, 2nd laboratory checks and 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% in standards with values under 1.5% copper and at 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 and accuracy of analysis of +/-3 g/t for the K92ML laboratory and the relative low level of some of the standards (1- 5 ppm).

Blanks comprise a locally sourced dacite or phyllite fragment generally >5 cm in size. The assay results for the blank indicate possible very minor contamination but the levels are low enough to be considered as not significant. The level of contamination where gold is concerned has markedly improved over time.

Laboratory duplicates comprise a second crushed pulp of the original sample analysed by the K92ML laboratory with the results showing no issue with the sample homogenization and subsequent acid digest and analysis. A good range of assays values have been included in the sample selection.

The second laboratory checks show no obvious bias. However, there is variation in results, this is not considered significant at this stage. A good range of assays values has been included in the sample selection. No coarse reject samples have been tested. Hole twinning has comprised four pairs of an original Barrick diamond drill hole with a K92ML diamond twin drill hole. 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, may 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 2018-2021 K92ML QAQC programme is of an acceptable level and conforms to common industry practice.

QAQC data is detailed in Section 11 of the 'Independent Technical Report, Mineral Resource Estimate Update, Kora and Judd Gold Deposits, Kainantu project, Papua New Guinea', effective date 20 January 2022.

12.0 DATA VERIFICATION

12.1 Site Visits

Andrew Kohler is a full-time employee of K92ML. He visited the Kainantu site in September 2021 and prior to that was regularly at the mine site on a roster basis that was completed in March 2021. Visits to the site in 2022 occurred in May, August and October. Prior to his role change he was employed as the Underground Mine Geology and Mine Exploration Manager. Mr Kohler 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. Detailed verification of the processes took place.

A site visit to Kora North was completed by Simon Tear of H&SC Consultants Pty Ltd in October 2018. The visit 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.

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 opinion of the respective QPs that the collar coordinates, downhole surveys, lithologies, and assay results are considered suitable to support the mineral resource estimation.

13.0 MINERALS PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

K92 engaged Lycopodium to prepare a report including cost estimates for a new processing plant located adjacent to the existing plant to accept ore from the Kora deposit. The design throughput for the new plant is 1.2 Mtpa.

This section covers the testwork undertaken and used as the basis for the process design, which was managed by Metallurgical Management Services Pty Ltd.

13.2 The 2021 Testwork Program

13.2.1 Overview

A series of tests were conducted on samples from K92 current plant operations, mining and development activities, and exploration drilling. The sequence of testwork was as follows:

- Sample receiving, preparation and preliminary compositing:
  - Stream samples (final concentrate, cleaner tailings, and final tailings) from plant operations whilst processing individual ores.
  - Belt cuts of mined ore, K1, K2 and Judd.
  - 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 K92 plant.

- 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 the three individual ore type and the Master Composite samples:
  - 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.

In parallel with the laboratory testwork, there was continued process development work at the operating 400 ktpa K92 Concentrator. This plant optimization work was supervised by K92 management.

13.2.2 Samples Tested

A full list of the samples tested with head analyses for the main elements of interest is shown below in Table 13.2.1. 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.

Table 13.2.1 List of All Samples Used in the DFS 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
Plant 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
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

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 Overview of Mineralogy in the Context of Plant Operations and Samples Evaluated

Chalcopyrite and pyrite are the dominant sulphides in the plant feed and flotation can achieve high recoveries of both minerals. In practice, lime is used for pH control in flotation and at high pH, pyrite can be depressed for selective copper flotation. Well liberated gold mineralization, as both free gold and telluride, is strongly recovered over the full range of pH used in the plant

There are physical factors limiting flotation of value minerals. Strongly floating minerals that are fully liberated and in the ideal size range for flotation will be efficiently recovered. Less strongly floating minerals, composites with weakly floating minerals, and particles that are either too coarse or too fine for flotation will float more slowly and will not achieve the same recovery rates. Some very fine mineral particles will be entrained in the liquid that makes up the froth and will report to concentrate in this way.

The processing plant incorporates a flash float cell treating a portion of the mill cyclone underflow. Compared with conventional float cells, it operates with a high froth loading, shallow froth depth and brief froth residence time. This cell can recover fast floating gold and sulphide mineral particles from the cyclone underflow. The shallow froth layer and short froth residence time allow it to recover larger particles than a conventional cell.

To some extent, the mineralogy of plant samples is a result of operating practices and management control actions. Plant operating objectives have been to maximize gold recovery and meet a target gold grade in final concentrate whilst maintaining moderately selective copper flotation. The K2 belt cut ore sample analysis shown in Table 13.2.1 has the highest copper grade and selective flotation of copper has been necessary to meet gold in concentrate targets. This is reflected in the high level of chalcopyrite in final concentrate plant sample and the displacement of pyrite and silicate gangue to the K2 tailings streams.

Plant samples have been evaluated by both QEMSCAN automated scanning electron microscopy and bulk mineralogy methods. QEMSCAN, with its electron microprobe analysis capability and image analysis software is a powerful tool for identifying minerals and quantifying parameters such as grain size and degree of locking of mineral particles. It has some limitations regarding gold mineralogy due to the relatively low (in absolute terms) concentrations of gold in all samples. Furthermore, light elements such as fluorine cannot be determined by electron microprobe analysis.

Additional information on gold and fluorine mineralogy has been provided by Laser Ablation ICP-MS spot analyses of selected sulphide and gangue mineral grains. Bulk mineralogical methods have also been applied to investigate the nature and mode of occurrence of gold in the plant samples.

13.3.2 QEMSCAN Mineralogy

Mode of Occurrence of Gold from QEMSCAN Mineralogy

Grains of native gold and gold tellurides were observed in all samples. The number of particles observed in each sample was much in line with the gold grade of the sample.

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.

Sulfide Mineralogy from QEMSCAN Results

Table 13.3.1 summarizes the mineralogy of chalcopyrite and pyrite in the flotation plant products from individual ore types. These results were obtained from QEMSCAN automated scanning electron microscopy and image analysis. Most of the chalcopyrite recovered to final concentrate is well liberated. However, about 50% of the pyrite recovered to final concentrate is in composite particles with some chalcopyrite. Pyrite liberation in the cleaner tailings and final tailings is much higher at between 75% and 80%. This indicates some degree of selective flotation of chalcopyrite over pyrite for all three ore types.

Table 13.3.1 Chalcopyrite and Pyrite in Flotation Plant Products

Mineralogical Parameter	K1	K2	Judd
Final Concentrate			
Chalcopyrite			
Mass %	30	63	17
Grain Size P_{80} µm	126	20	38
Percentage Well Liberated %	81.5	90	81,5
Main Locking Mineral	pyrite	pyrite	pyrite
Pyrite			
Mass %	22.9	8.0	10
Grain Size P_{80} µm	96	20	31
Percentage Well Liberated %	50%	50%	80%
Main Locking Mineral	chalcopyrite	chalcopyrite	silicates, chalcopyrite
Cleaner Tailings			
Chalcopyrite			
Mass %	1.36	5.9	2.5
Grain Size P_{80} µm	35	49	43
Percentage Well Liberated	67.4	82	75
Main Locking Mineral	pyrite and silicates	pyrite and silicates	pyrite and silicates
Pyrite			
Mass %	15	16	10
Grain Size P_{80} µm	39	50	42
Percentage Well Liberated %	80	75	80
Main Locking Mineral	silicates	silicates, chalcopyrite, bornite	silicates, chalcopyrite
Final Tailings			
Chalcopyrite			
Mass %	0.09	2.54	0.38
Grain Size P_{80} µm	56	62	45
Percentage Well Liberated %	50	76	75
Main Locking Mineral	pyrite and silicates	pyrite and silicates	pyrite and silicates
Pyrite			
Mass %	15	16	10
Grain Size P_{80} µm	54	59	55
Percentage Well Liberated %	80	75	80
Main Locking Mineral	silicates	silicates, chalcopyrite, bornite	silicates, chalcopyrite

Occurrence and Distribution of Mineral Species in Plant Flotation Products

The main findings for the distribution of mineral species are summarized below in Table 13.3.3. General observations are as follows:

- Chalcopyrite is the main copper bearing mineral in all samples.

- Minor amounts of copper minerals bornite, chalcocite / covellite and tennantite-tetrahedrite are also present.

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

- Pyrite is less abundant whilst chlorite and feldspars are more abundant in all three Judd products compared with K1 and K2 products.

The K1 belt cut sample has the highest sulphur grade as reflected by high levels of pyrite in all three K1 plant samples. The gold grade of the K1 concentrate was high and the concentrate contains higher levels of pyrite and silicate gangue compared with the K2 and Judd samples.

The mineral distributions in cleaner tails and final tails samples reflects the above trends.

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

The main findings for grain size distribution, are summarized below in Table 13.3.2. 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 13.3.3 of the 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 13.3.2 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 13.3.3 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

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 13.3.4. General observations are as follows:

- Copper accountability between chemical analyses and analyses calculated from QEMSCAN results was generally good indicating that copper containing minerals have been correctly identified and quantified.
- The proportion of copper hosted by chalcopyrite is highest in the final concentrate samples. However, the proportion of copper hosted by chalcopyrite varies 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 generally strongly floating and tends to be recovered efficiently to concentrate. Bornite can have quite variable flotation characteristics and is prone to poor recovery if it has been subjected to any degree of oxidation or weathering. The secondary sulphide minerals chalcocite and covellite require the use of a sulphurising agent for efficient recovery. Without such a reagent, they can appear to be weakly floating.

Table 13.3.4 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

Mode of Occurrence of Fluorine in the Plant Samples

ALS Mineralogy made the following note in connection with fluorine contents of the various mineral species:

"The fluorine deportment data above is indicative only due to: (a) the F content in muscovite and chlorite was assigned based on the average values of ten grains analysed by electron probe micro analysis. These average values carry a large uncertainty because the F content in highly variable and more grains are to be analysed to improve the accuracy; (2) other minerals in the samples might be F-bearing, for example, Fe-Mn-oxides/oxyhydroxides, but no F content was assigned."

Variations in mode of occurrence of fluorine are summarized below as follows:

- Fluorine accountability between chemical analyses and analyses calculated from QEMSCAN results was reasonably good for the K1 samples. However, it was quite poor for K2 and Judd samples. In all cases, QEMSCAN estimates were lower than corresponding chemical analyses .

- Most of the fluorine content of all samples is associated with muscovite. The K1 and Judd samples contain about 90% and about 80% respectively muscovite associated fluorine. Furthermore, for these samples, the proportions of muscovite association are similar across final concentrate, cleaner tails and final tails.

The K2 samples seem quite different with over 20% of the final concentrate fluorine in fluorite and andalusite type minerals, compared with less than 10% for K1 and Judd

13.3.3 Laser Ablation ICP-MS Analysis of Sulphide Grains

In a separate exercise, grains of chalcopyrite and pyrite were micro-probed for gold using laser ablation ICPMS. Results for the three ore types were as shown below in Table 13.3.5. 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. The results indicate that solid solution and sub-micron gold in pyrite and chalcopyrite is a minor contributor to gold losses in final tailings.

Table 13.3.5 Average Gold Contents of Pyrite and Chalcopyrite Grains by LAICPMS Probing

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

13.3.4 Bulk Mineralogy

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 13.3.6. 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.

Table 13.3.6 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	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	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	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	5.7	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

It is interesting to note that all three final concentrate samples are out of marketing specification regarding fluorine content.

Screen Fire Assay / Leachwell Evaluation

Samples of the flotation products were subject to a combined screen fire assay and Leachwell cyanide soluble gold determination.

In this procedure, a weighed portion of about 800 grams of sample was ring mill pulverized and then screened at 106 µm. Any metallic gold particles larger than about 10 µm would be flattened during pulverization and would report to screen oversize. The mass yield to screen oversize was between 0.7% and 2.7% by weight. The entire screen oversize was sent to fire assay for gold determination.

A 500-gram portion of screen undersize was then subject to accelerated cyanide leaching using Leachwell reagent. The dissolved gold was then determined. A 50-gram portion of the washed residue was then subject to fire assay. The analyses were calculated to represent the initial sample weight and then interpreted as 'Coarse Gold', 'Cyanide Soluble Gold' and 'Refractory Gold'.

In practice, most of the coarse gold will be recoverable by recoverable by gravity concentration. Most of the cyanide soluble gold will occur as fine free gold particles, whilst most of the refractory gold will occur as gold telluride particles and as fine free gold locked in sulphides or silicates.

Sequential Gold Leach Evaluation

Samples of the flotation products were subject to a four-stage sequential gold leach and fire assay procedure. The four stages of the procedure are as follows:

- An intensive cyanide leach followed by solution analysis for gold. This determines the free gold content of the sample.

- The residue goes to hydrochloric acid leaching followed by an intensive cyanide leach and solution analysis for gold. This determines the carbonate and pyrrhotite locked gold content of the sample.

- The second stage residue goes to leaching with hot Aqua Regia. This determines the sulphide locked gold content of the sample.

- The third stage residue goes to fire assay to determine the silicate locked gold content of the sample.

Table 13.3.7 shows a comparison between the 'Free and slow leaching gold' from the sequential leach tests, and the sum of the 'Coarse gold' and 'Cyanide soluble gold' from the screen fire assay tests. The head-grades determined by the two methods are also shown. Clearly there is some scatter between results that should nominally be the same. The differences are probably the result of the following factors:

- Sampling errors associated with products containing particulate free gold.
- Minor differences in response of the gold mineralization from the three ore types to the screen fire assay and sequential leach procedures.

Table 13.3.7 Cyanide Soluble Gold and Head Grade Comparison Between Screen Fire Assay and Sequential Leach Results

Product	Screen Fire Assay			Sequential Leach		
	Coarse and Cyanide Soluble Gold		Head	Free and Slow Leaching Gold		Head
	g/t	%	g/t	g/t	%	g/t
K1 Final Concentrate	184.8	29.9	617.45	322.3	52.0	619.6
K1 Cleaner Tails	15.6	87.9	17.76	6.75	77.5	8.71
K1 Final Tails	3.84	48.4	7.93	2.60	74.2	3.50
K2 Final Concentrate	18.1	24.8	72.88	7.08	9.8	72.2
K2 Cleaner Tails	4.25	43.7	9.73	3.81	45.7	8.33
K2 Final Tails	2.83	54.2	5.23	2.09	56.9	3.68
Judd Final Conc.	27.7	50.4	55.04	22.1	65.7	33.6
Judd Cleaner Tails	4.77	84.8	5.62	2.7	84.3	3.15
Judd Final Tails	0.94	77.7	1.21	0.8	74.8	1.03

Despite the scatter, there is a general pattern of indicated gold occurrence between samples from the three different ore types. The results are interesting in several respects:

- The results show relatively low levels of coarse gold in all three K1 products. The K2 and Judd final tails samples both contain significant levels of coarse gold. The Judd final concentrate and final tails samples contain more coarse and cyanide soluble gold than corresponding K1 and K2 samples. The K2 samples contain the lowest levels of free and slow leaching gold.

  The K1 and Judd cleaner tails samples both contain high proportions (>80%) of cyanide soluble gold. This is probably fine liberated or partially liberated gold particles. The K2 samples show the lowest proportions of cyanide soluble gold. Most of the gold in the final tails samples occurs as free gold and a relatively small proportion is silicate locked.

- There are high levels of refractory gold in all three final concentrates. This could be gold occurring as the telluride mineral. The sequential leach method indicates that a high proportion (61.7%) of the gold in the K2 Final concentrate is locked in silicates. This is much in line with the screen fire assay result that shows 75% refractory gold.

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 13.4.1 whilst parameters derived from the SMC results are shown in Table 13.4.2. Bond test results are shown in Table 13.4.3.

Table 13.4.1 SMC Test Results

Sample	Dwi kWh/m³	Dwi %	MI Parameters			SG t/m³
			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 13.4.2 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 13.4.3 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				

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

13.5.1 Overview and Rationale of the Testwork

A 20 kg portion of final concentrate was obtained from the K92 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 ground to simulate concentrate produced from a flotation circuit incorporating rougher concentrate regrinding.

The ground samples were received and 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.

13.5.2 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 parameters for fine grinding, sizing, and design 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 then 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. The 3 mm bead top size was less able than the 5 mm top size to break the coarse particles in the concentrate. Keramos recommended the use of 5 mm beads for a K92 concentrate regrind application.

The Relationship Between Specific Energy and Grind Size

Results of the Signature Plot are presented in Figure 13.5.1, which shows data for grind sizes produced by a range of specific grinding energy input. The calculation shown in Figure 13.5.1 indicates that a Specific Grinding Energy of 6.2 kWh/t is required to achieve a grind P_{80} of 30 µm.

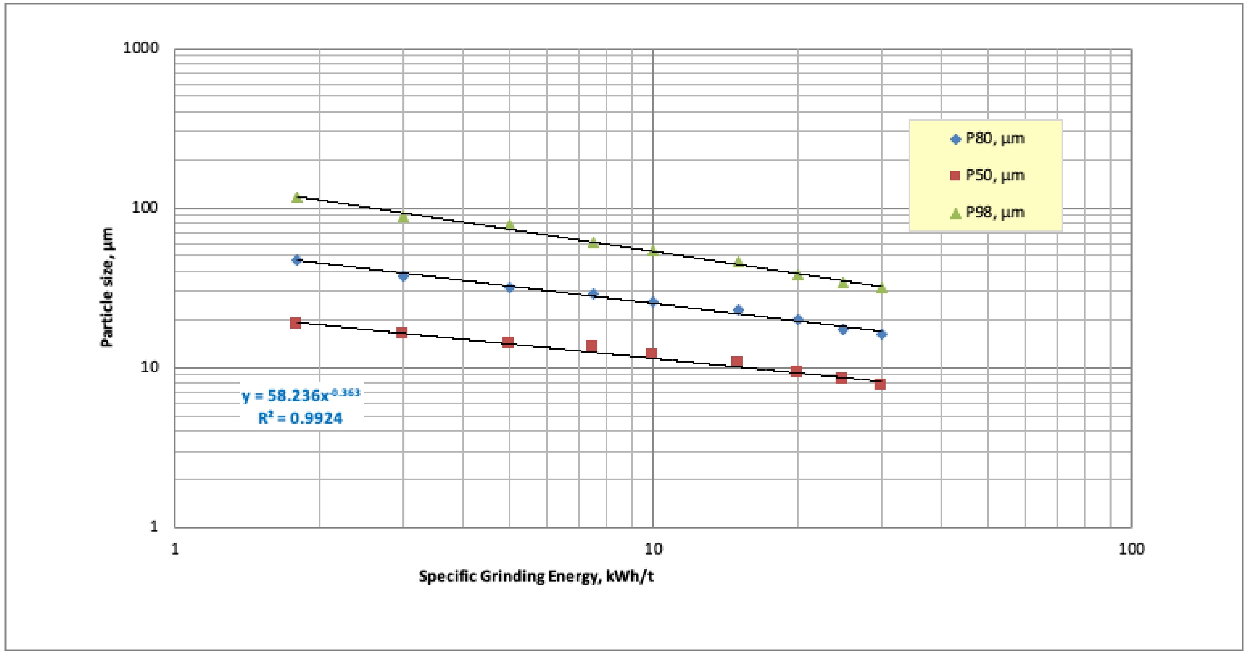
Figure 13.5.1 Metso SMD Signature Plot Results Showing Grind Size Data at a Range of Specific Energy

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



13.5.3 Concentrate Thickening Testwork

Overview

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

The test work showed that the concentrate samples can be successfully thickened to high densities. In dynamic thickening tests, with 0.25 t/(m^2*h) solids loading, the unground 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 ground concentrate sample gave underflow densities of 61.8% to 63.8% solids (w/w) with yield stresses ranging from 38 Pa to 69 Pa.

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 13.5.1.

Table 13.5.1 Recommended Thickener Design Parameters

Process Stream	Unground Concentrate P_{80} 76 µm	Ground 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

Flocculant Selection and Feed Dilution Tests

Prior to the dynamic thickening tests, four different BASF flocculants were evaluated in static measuring cylinder tests. The best results in terms of settling rate and overflow clarity were obtained with BASF Magnafloc 155. Dosage rates of between 10 and 50 g/t of this product gave settling rates of about 140 m/h.

Feed dilution tests were conducted using MOG's optimum dilution test method. This determines the maximum feed solids concentration for efficient flocculation with MOG's Vane Feed well.

Batch Dynamic Thickening Test Method Used by the MOG Laboratory

The samples were dynamically thickened in a 99 mm diameter rig. For each test, stock slurry was drawn from an agitated vessel by a variable speed peristaltic pump and diluted with Perth tap water using a second pump. The pH of the slurry was adjusted to 11 by using lime.

Flocculant was added at a set flow rate but at a range of concentrations to deliver the required dosage for each test. Stock flocculant solution was hydrolysed in Perth tap water to 0.25 % w/v (2.5 g/L), and then diluted as required. The diluted flocculant was dosed half into the diluent line and half into the diluted feed line further downstream.

The flocculated solids bed was raked continuously at 2 r/min in conjunction with two stationary pickets to aid de- watering. Overflow samples were collected at a bed height of 120 mm, just at the opening of the feed well discharge. Underflow samples were taken with a positive displacement pump.

The suspended solids content of overflow samples was determined by filtration and then drying and weighing filter discs. The solids concentration of underflow slurry samples was determined by weighing a known volume of slurry, followed by drying and weighing the dry solids.

All rheological measurements were carried out using a Thermo Haake VT550 rheometer and an 'OK600' 4 blade vane. For each dynamic test underflow sample, a simple un-sheared vane yield stress was measured.

The underflow yield stress was measured to generate data for sizing of the thickener rake drive. It was not intended to be used for any other purpose such as for determining pumping requirements.

Dynamic Thickening Tests

All tests were conducted at a Solids Loading of 0.25 t/(m^2*h) and using BASF Magnafloc 155 flocculant. There were three test runs on the unground sample and four test runs on the ground sample.

For the unground sample, the first run used a flocculant dose of 10 g/t flocculant. This achieved an underflow density of 69.5% solids (w/w) with a slurry yield stress measured at 77 Pa. The overflow clarity was very good, with less than 100 mg/L suspended solids.

The second run was conducted at the same conditions as the first run, though the flocculant dose was reduced from 10 to 5 g/t. The underflow density decreased from 69.5% to 68.7% solids (w/w) and the yield stress slightly decreased from 77 to 59 Pa. The overflow quality was again very good, with less than 100 mg/L suspended solids.

For the third run the flocculant dose was reduced from 5 g/t to 2 g/t. The underflow density decreased from 68.7% to 66.7% solids (w/w) and the yield stress from 59 to 39 Pa. The overflow quality was good, at 120 mg/L suspended solids.

For the ground sample, the first run used a flocculant dose of 30 g/t. This achieved an underflow density of 63.7% solids (w/w) with a slurry yield stress measured at 69 Pa. The overflow quality was very good, at less than100 mg/L suspended solids.

For the second run the flocculant dose was reduced from 30 g/t to 20 g/t. The underflow density slightly increased from 63.7% to 63.8% solids (w/w) and the yield stress slightly decreased from 69 to 65 Pa. The overflow quality was again very good, at less than 100 mg/L suspended solids.

For the third run the flocculant dose was reduced to 10 g/t. The underflow density decreased from 63.8% to 63.5% solids (w/w) and the yield stress from 65 to 63 Pa. The overflow quality was again very good, at less than 100 mg/L suspended solids.

For the fourth run the flocculant dose was reduced to 5 g/t. The underflow density decreased from 63.5% to 61.8% solids (w/w) and the yield stress from 63 to 38 Pa. The overflow quality was again very good, measuring less than 100 mg/L suspended solids.

Based on their analysis of the results, MOG recommended thickener sizing based on results for flocculant additions of 10 g/t and 20 g/t for the unground and ground concentrate.

13.5.4 Pressure Filtration Testwork

Overview

Pressure filtration testwork was performed by the MOG laboratory in Perth on thickened slurry samples from the preceding thickener tests. The objectives were to establish design parameters for filtering 18 t/h of concentrate and producing filter cake with a maximum of 10% moisture content. It was envisaged that the concentrate would be filtered in a 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.

From the testwork results, the MOG laboratory report recommended filter sizing based on the data shown in Table 13.5.2.

Table 13.5.2 Recommended Pressure Filtration Parameters

Filter Feed	Unground Concentrate	Ground 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 testwork report.

Outline Details of the Test program

A chamber thickness of 45 mm was initially tried with the unground 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 and ground samples are shown below in Figure 13.5.2 and Figure 13.5.3. These graphs have been copied from the MOG laboratory report.

Figure 13.5.2 Filtration Results on the Unground Sample

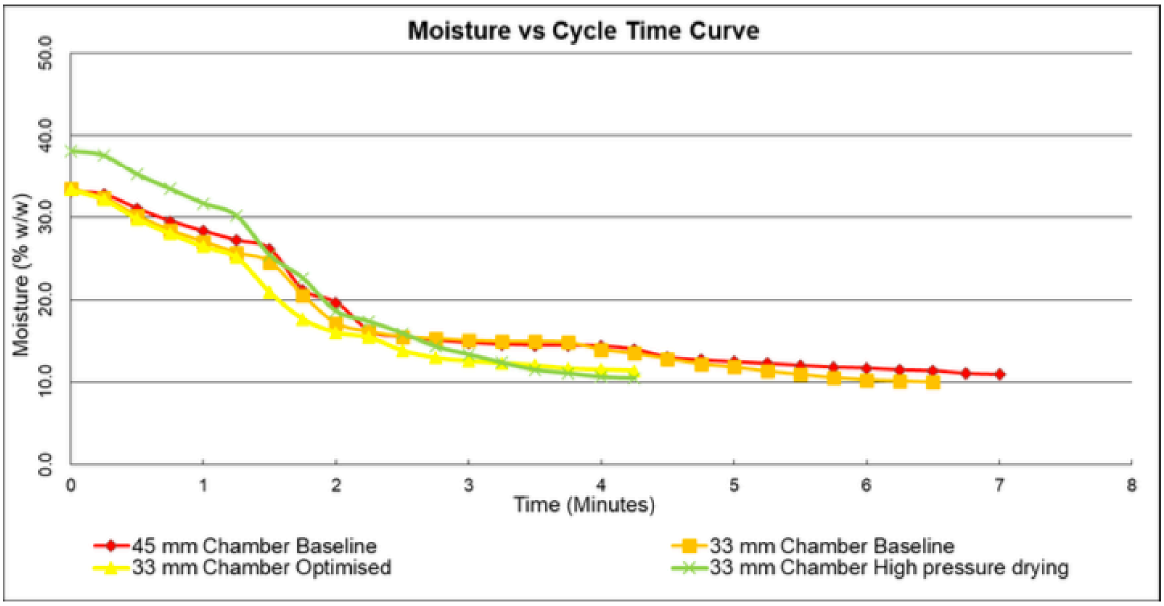
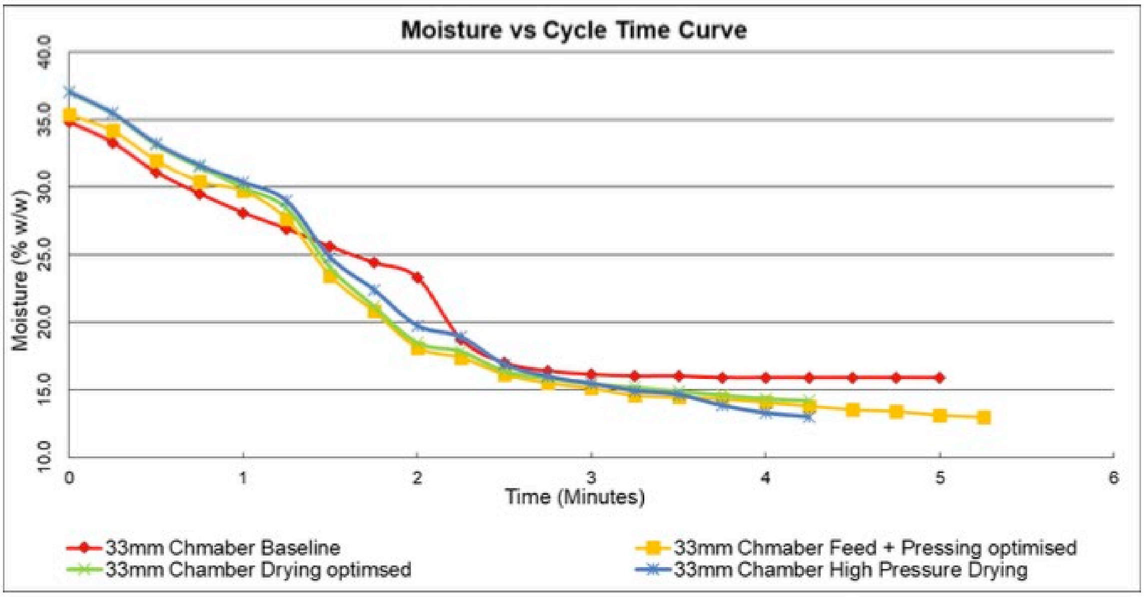


Figure 13.5.3 Filtration Results on the Ground Sample



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

13.5.5 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

The rheology measurements indicated that both samples had a yield stress point, consistent with the Bingham fluid model. Shearing of the pulp before testing generally resulted in a measured yield stress reduction.

Rheology Results

Rheology results are shown in the figures below.

In Figure 13.5.4, for the unground sample, there is a large increase in shear stress between 69% solids (w/w) and 74% solids (w/w). In Figure 13.5.5, the measured yield stress increases rapidly beyond about 76% solids (w/w).

In Figure 13.5.6, for the ground sample, there is a large increase in shear stress between 57% solids (w/w) and 66% solids (w/w). In Figure 13.5.7, the measured yield stress increases rapidly beyond about 62% solids (w/w).

Figure 13.5.4 ALS Results for the Unground Sample, Shear Stress versus Shear Rate

A23132 – Kora 106 µm Composite

SHEAR STRESS vs SHEAR RATE

Sample ID	Slurry Temp. (ºC)	Pulp Density % Solids (w/w)	Haake Viscotester IQ Shear Stress @ Shear Rate (Sec^{-1}) 4.2 (Pa)	7.4 (Pa)	13.1 (Pa)	21.9 (Pa)	38.9 (Pa)	67.4 (Pa)	119.2 (Pa)	209.5 (Pa)
Kora 106 µm Composite	22	55	2.5	2.5	2.8	3.1	3.2	3.5	4.1	5.7
		63	8.3	8.9	8.9	9.2	9.6	10	11	13
		69	24	24	24	25	26	27	29	33
		74	470	454	472	467	394	309	270	250
		79	OVR	OVR	OVR	OVR	OVR	OVR	OVR	OVR

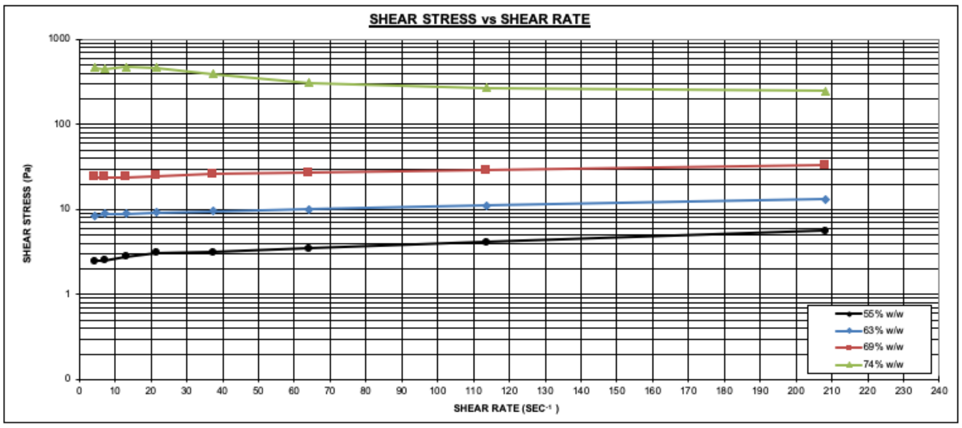


Figure 13.5.5 ALS Results for the Unground Sample, Measured Yield Stress versus Solids Concentration

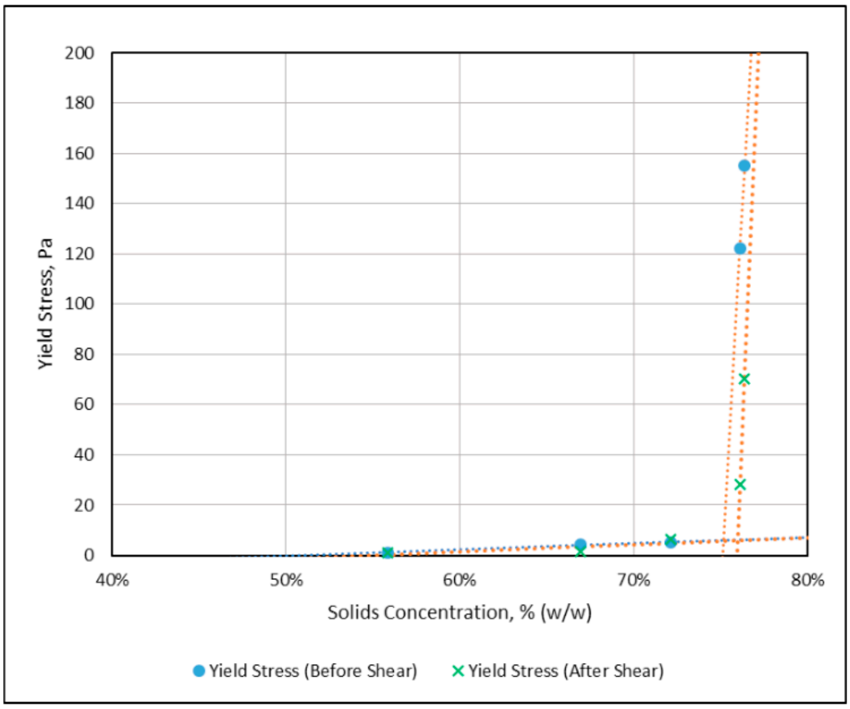


Figure 13.5.6 ALS Results for the Ground Sample, Shear Stress versus Shear Rate

A23132 – Kora 30 µm Composite

SHEAR STRESS vs SHEAR RATE

Metallurgy

Sample ID	Slurry Temp. (ºC)	Pulp Density % Solids (w/w)	Haake Viscotester IQ Shear Stress @ Shear Rate (Sec^{-1}) 4.2 (Pa)	7.4 (Pa)	13.1 (Pa)	21.9 (Pa)	38.9 (Pa)	67.4 (Pa)	119.2 (Pa)	209.5 (Pa)
Kora 30 µm Composite	20	43	0.40	0.42	0.48	0.64	0.64	0.85	1.4	3.3
		51	1.9	1.8	2.0	2.2	2.2	2.6	3.2	4.9
		55	3.8	4.1	4.1	4.4	4.7	5.0	5.9	7.7
		57	5.0	5.1	5.2	5.4	5.7	6.0	6.9	8.7
		57	6.0	5.8	5.9	6.1	6.3	6.6	7.5	9.0
		66	340	351	362	348	329	307	310	317

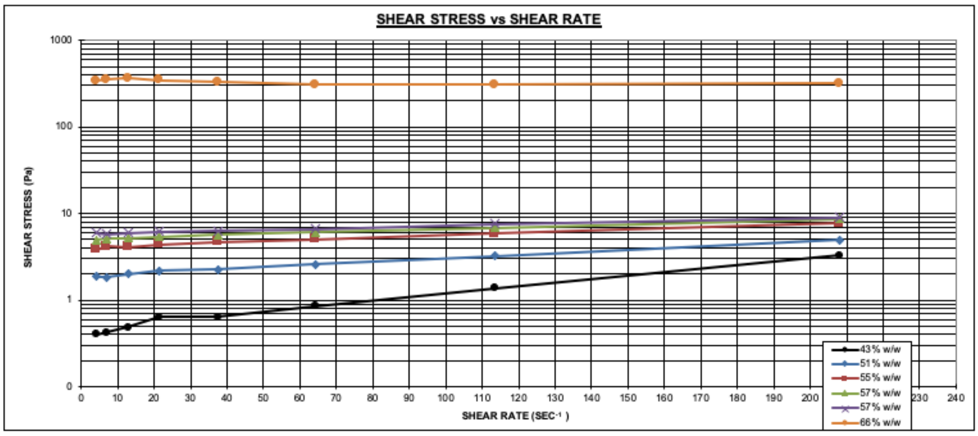
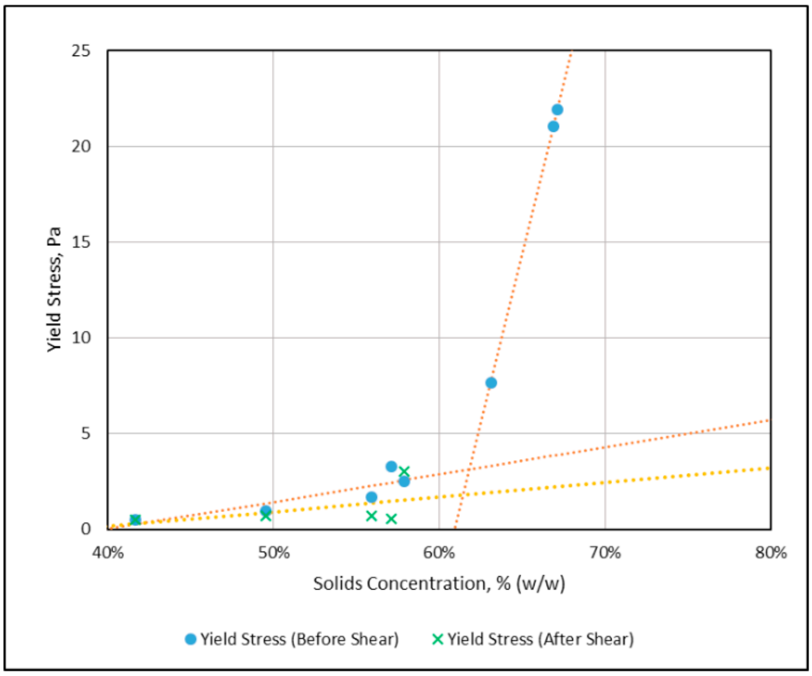


Figure 13.5.7 ALS Results for the Ground Sample, Measured Yield Stress versus Solids Concentration



13.6 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 on 7 May. 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.

13.7 Gravity Recoverable Gold Tests, on the Three Individual Ore Type Samples

13.7.1 Processing Considerations

The mineralogical evaluations reported in Section 13.6 indicate that all three ore types contain native gold particles of wide-ranging size. During grinding a proportion of these particles will be liberated.

Gold particles have a higher SG and lower breakage rate than rock mineral particles. As a result of these two factors, gold particles will tend to build up in the circulating load, and gold going to rougher flotation will have a finer size distribution than rock minerals.

Liberated gold particles in the cyclone underflow stream can be recovered by froth flotation and/or particle density-based concentration. Specialized flotation cells known as 'flash flotation cells' are used to recover well liberated gold and value mineral particles from the cyclone underflow.

Recovery of liberated gold particles based on particle density is commonly referred to as 'gravity concentration', with the gold thus recovered being termed gravity recoverable gold (GRG). In the existing K92 processing plant, a portion of the cyclone underflow goes to a flash flotation cell, and another portion goes to a centrifugal gravity concentrator. Both the flash float cell and the centrifugal concentrator will recover GRG. However, 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 K92 flash flotation cell to recover gold particles of various sizes has not been evaluated by testwork. However, testwork on other flash float installations has shown excellent recoveries of gold particles up to a size of about 100 microns. Flash float recovery of progressively larger gold particles decreases, 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-Johnson1, A Bax, W P Staunton, J McGrath and J Eksteen, World Gold Conference / Brisbane, Qld, 26 - 29 September 2013.

Conventional float cells are much less capable than flash flotation cells regarding coarse particle gold 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. It is noted that gold recovery in conventional rougher flotation cells decreases rapidly at particle sizes greater than 75 µm.

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.

13.7.2 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. Sequential passes through the Knelson used the conditions shown in Table 13.7.1.

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. Results were plotted on graphs showing the total cumulative gravity gold recovered (from coarse to fine) versus particle size. The lower line shows the gravity gold recovered in the first pass whilst the upper line shows the total gravity gold recovered in all three passes.

Table 13.7.1 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

13.7.3 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 13.7.2. 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 13.7.3. Gold distributions by size for the K1, K2 and Judd ores are shown below in Figure 13.7.1, Figure 13.7.2, and Figure 13.7.3 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).

Table 13.7.2 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 13.7.3 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 13.7.1 K1 Composite: Size Distribution of Gravity Recoverable Gold

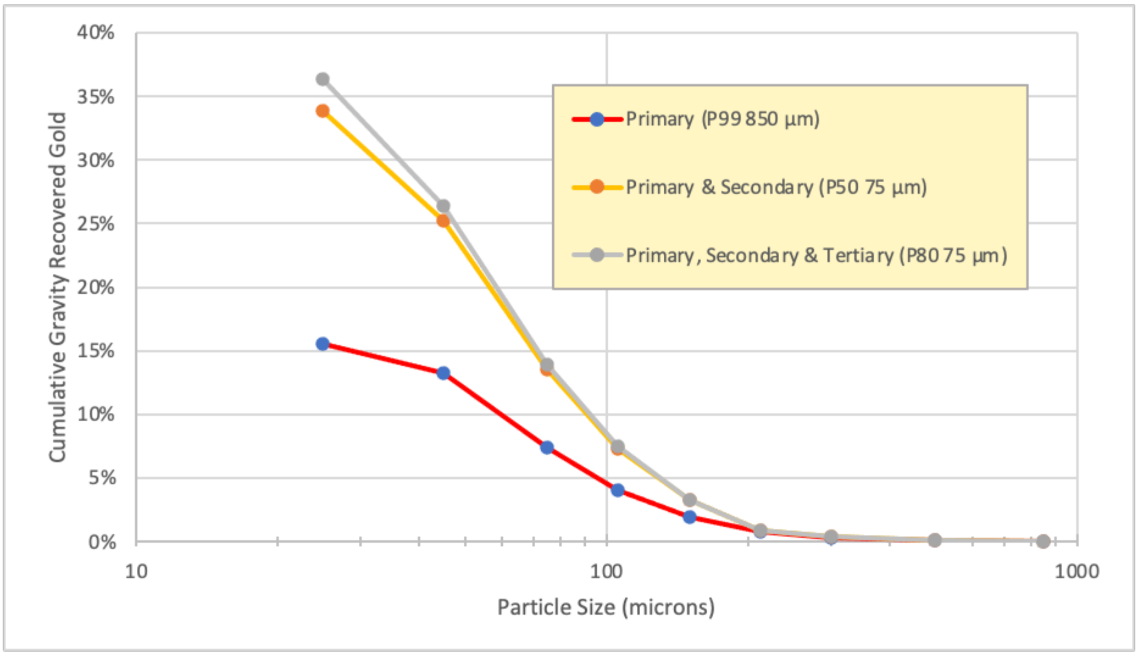


Figure 13.7.2 K2 Composite: Size Distribution of Gravity Recoverable Gold

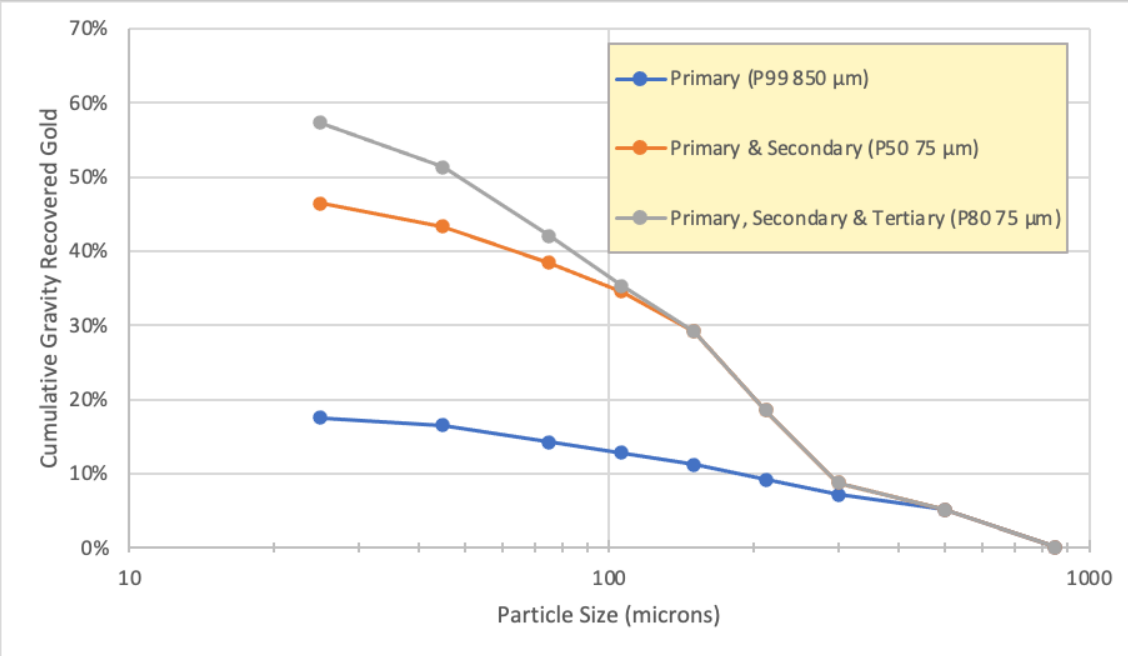
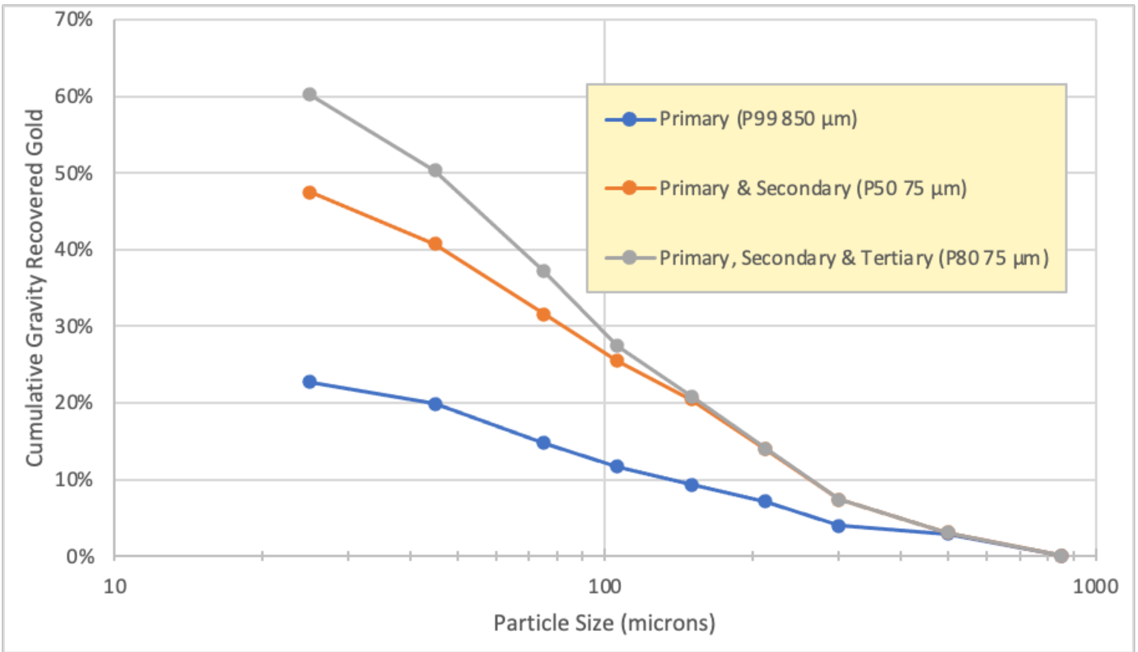


Figure 13.7.3 Judd Composite: Size Distribution of Gravity Recoverable Gold



13.7.4 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.

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

13.8.1 Overview and Rationale Behind the Testwork

Batch flotation Tests were conducted on the three individual ore composites and the Master Composite. 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}). Flotation tests used one-kilogram portions of sample and were conducted using a 2.5 litre laboratory float cell, with a rotor speed of 800 rpm.

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.

This batch testwork was then to be followed up by two or three locked cycle flotation tests on the master composite.

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_{80s} 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).

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 open circuit and closed circuit first cleaner tails.

13.8.2 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.

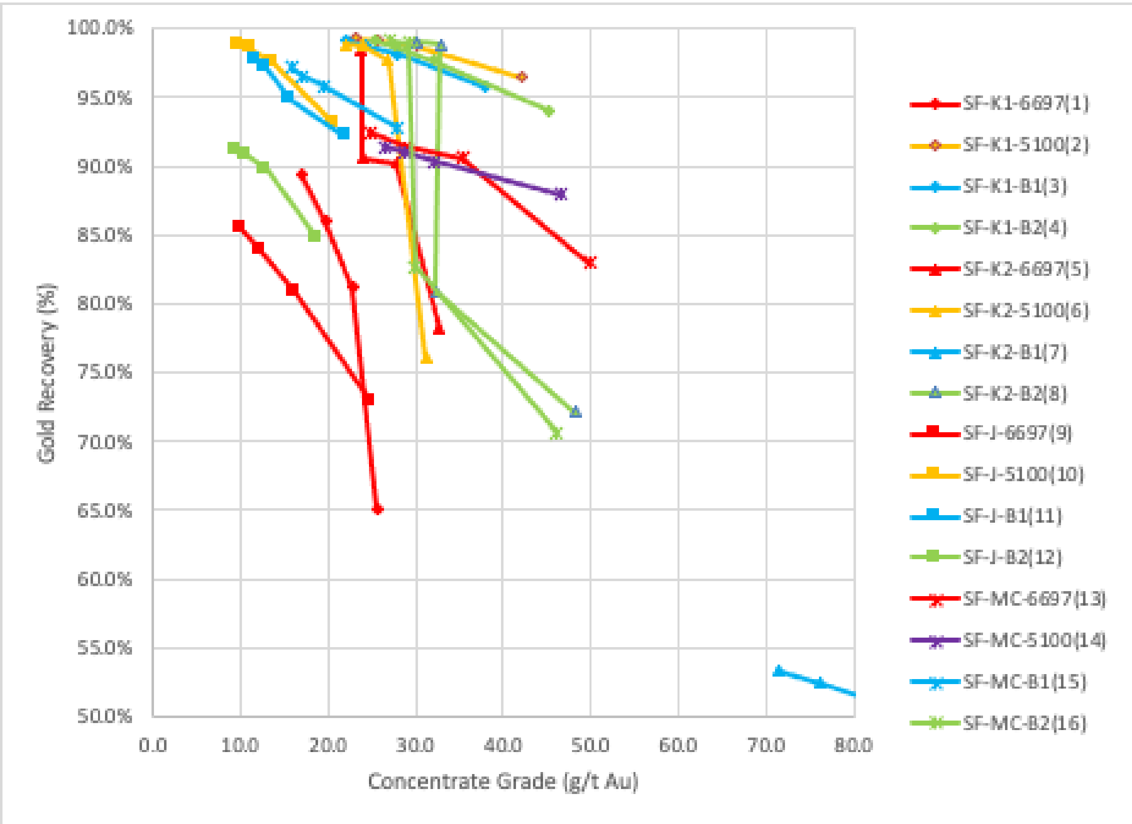
Results of the Collector Comparison Tests

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 13.8.1. 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 13.8.1 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 Al_2O_3 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 13.8.1.

Table 13.8.1 Comparison of Collectors - Total Performance Points

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

13.8.3 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 13.8.2. 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 13.8.2 Grind Comparison - Total Performance Points

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

13.8.4 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.

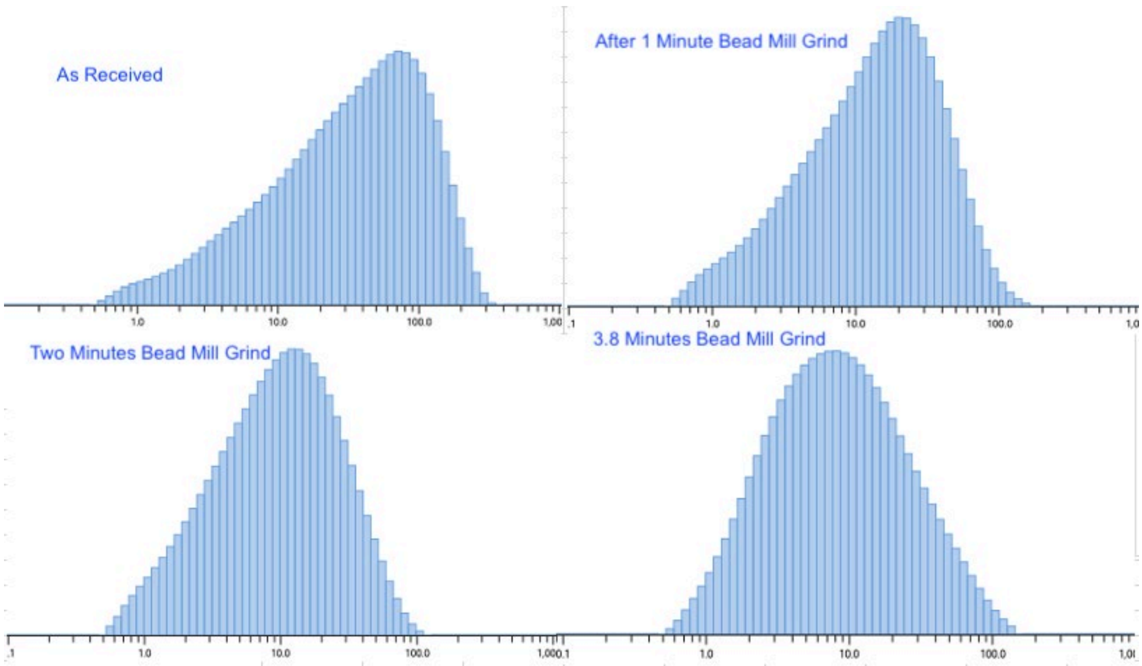
Selection of Rougher Concentrate Regrind Intensity

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 13.8.2, 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 13.8.2 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 fourth 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 K92 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.

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 was shown in Figure 13.8.1.

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.

13.9 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.

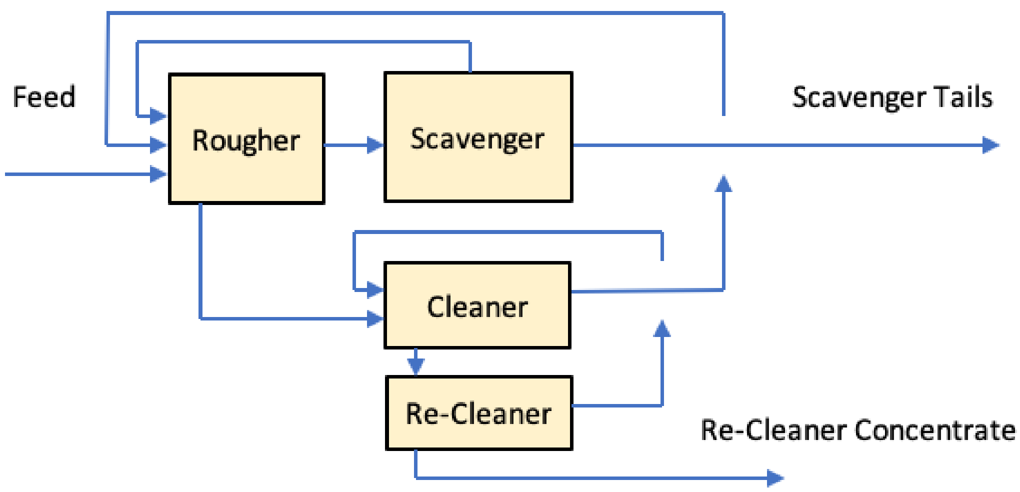
The K92 Expansion Project will benefit from operating data from the existing plant. There are practical limits to the grind and other flotation parameters that can be investigated without adversely affecting operations. However, full-scale plant operating data has much greater authority than laboratory 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 13.9.1, 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'.

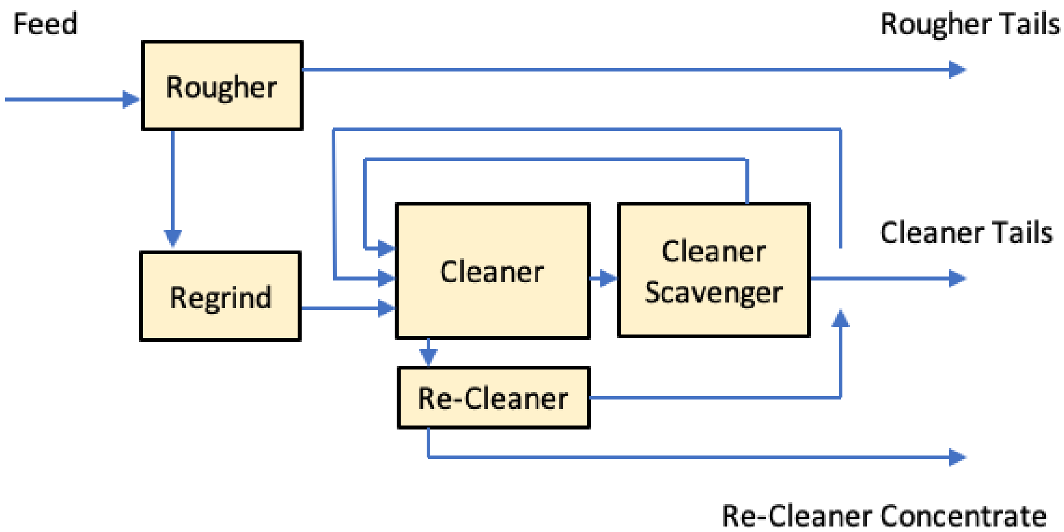
Figure 13.9.1 Flowsheet for the Closed-Circuit Cleaner Tails, No Regrind LCT Test



The second circuit incorporates regrind of rougher concentrates and has open circuit cleaner tails as shown in Figure 13.9.2. 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 13.9.2 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 13.9.1 and Table 13.9.2. The procedures are very similar to the rougher cleaner recleaner batch test.

Table 13.9.1 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 13.9.2 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**

After some discussion with K92 Operations, DOW 250 was selected at the frother rather than the MIBC used in previous tests. It had been noted that MIBC is rarely used in industry due to carcinogenic risks and the fact that it is highly flammable.

In the tables, lime additions are shown for the first tests 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 13.9.3. 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 13.9.4 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 13.9.3 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 Rougher	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 Rougher	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 Rougher	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 Rougher	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 13.9.4 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 13.9.7. Similarly copper recovery results are shown in Figure 13.9.7. 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 13.9.7 are slightly more scattered than the gold results. From the data shown in Table 13.9.4, this is due to some low recoveries associated with the 4 minute regrind of rougher concentrate.

Figure 13.9.3 Gold Recovery versus Concentration Ratio Results for the K1, K2, Judd, and Master Composites (Locked Cycle and Rougher Cleaner Recleaner Tests)

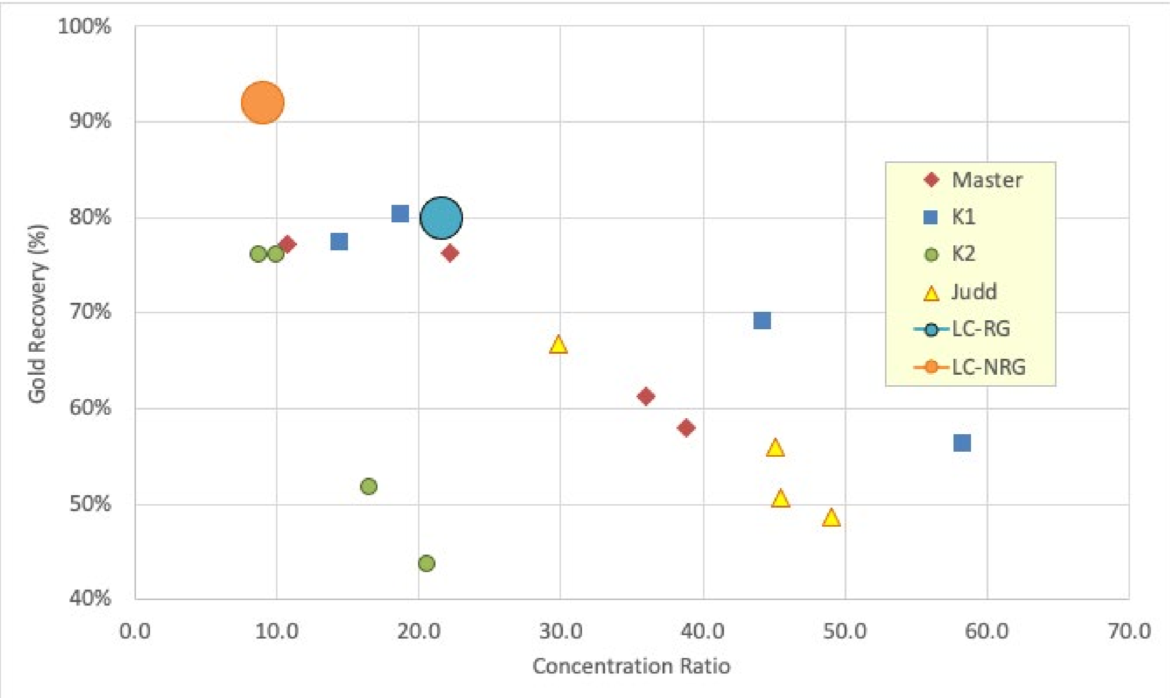
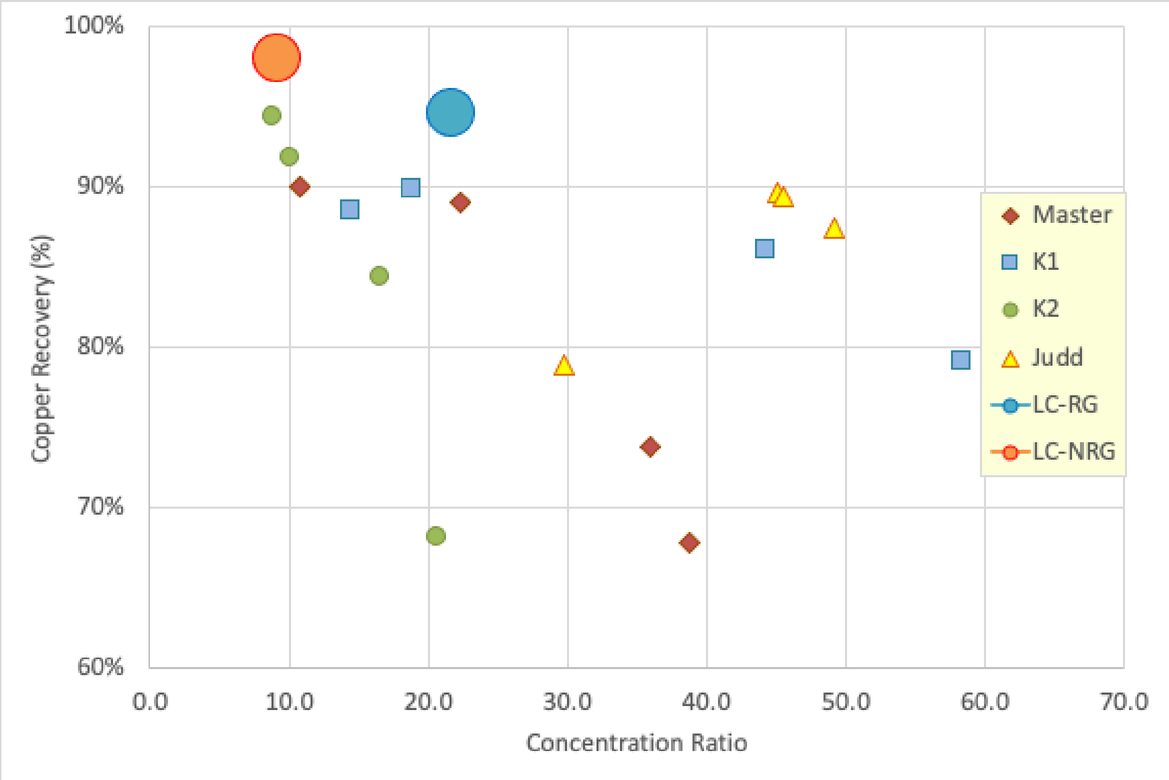


Figure 13.9.4 Copper Recovery versus Concentration Ratio Results for the K1, K2, Judd, and Master Composites (Locked Cycle and Rougher Cleaner Recleaner Tests)



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.

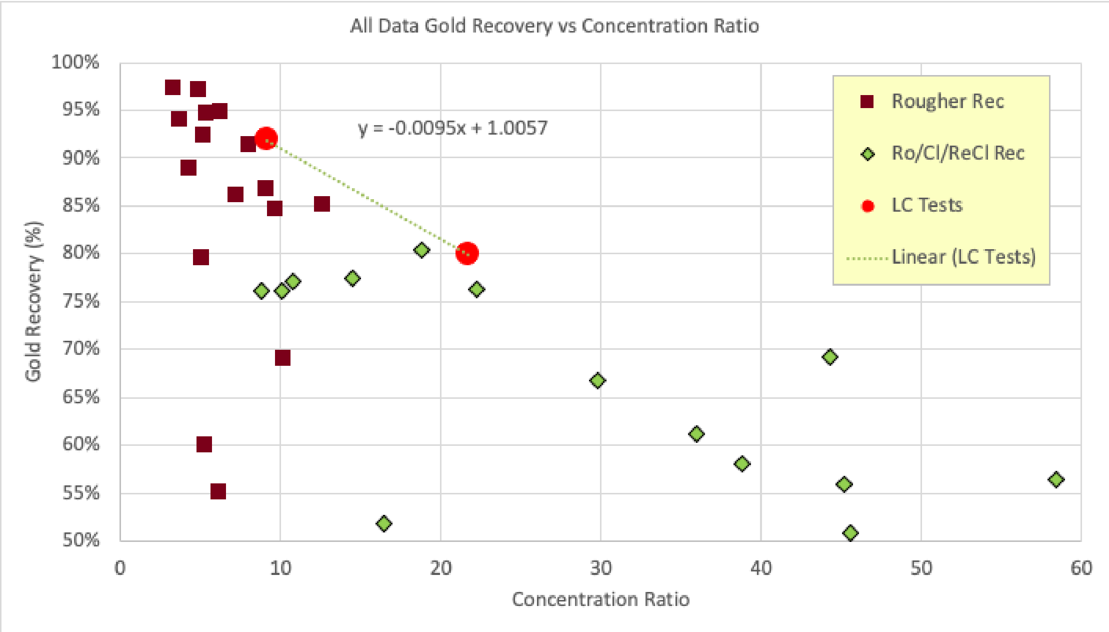
Recovery versus Concentration Ratio Evaluation of the Rougher Regrind Cleaner Flotation Results

In Figure 13.9.7, 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:

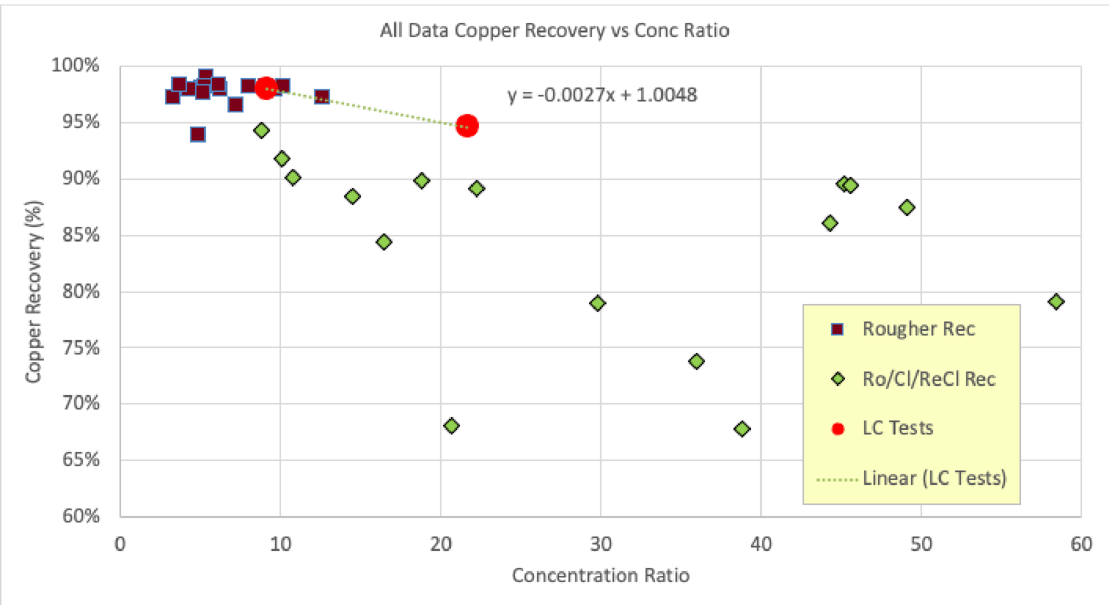
Equation 1: Gold Recovery % = 100 * (-(0.0095 * CR) + 1.0057)

Figure 13.9.5 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.

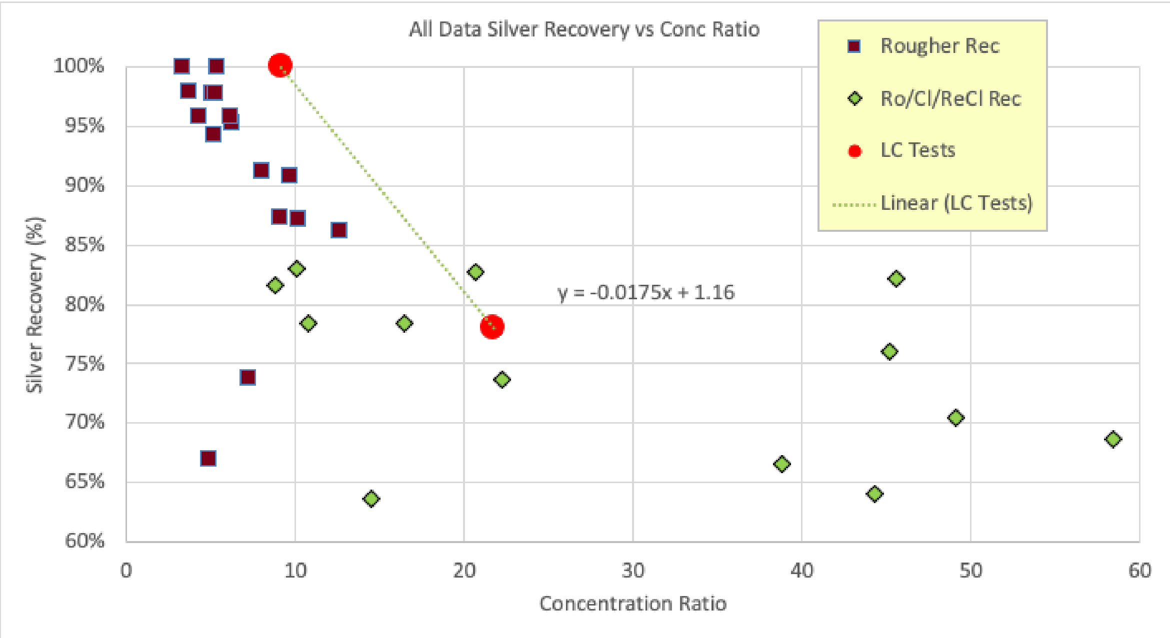
Figure 13.9.6 Copper Recovery Results from the Locked Cycle Tests Shown with the Rougher, Cleaner Recleaner Results



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 13.9.7 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 K92 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.

13.10 Variability Testwork on Eight Composites of Diamond Drill Core

Overview

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.

It could be concluded that none of the variability samples show a problematic response to flotation recovery.

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 13.10.1. 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 13.10.1 and Figure 13.10.2 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 13.10.1 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 13.10.1 Gold Recovery versus CR for All Rougher Floats Using Blend 1 Collector

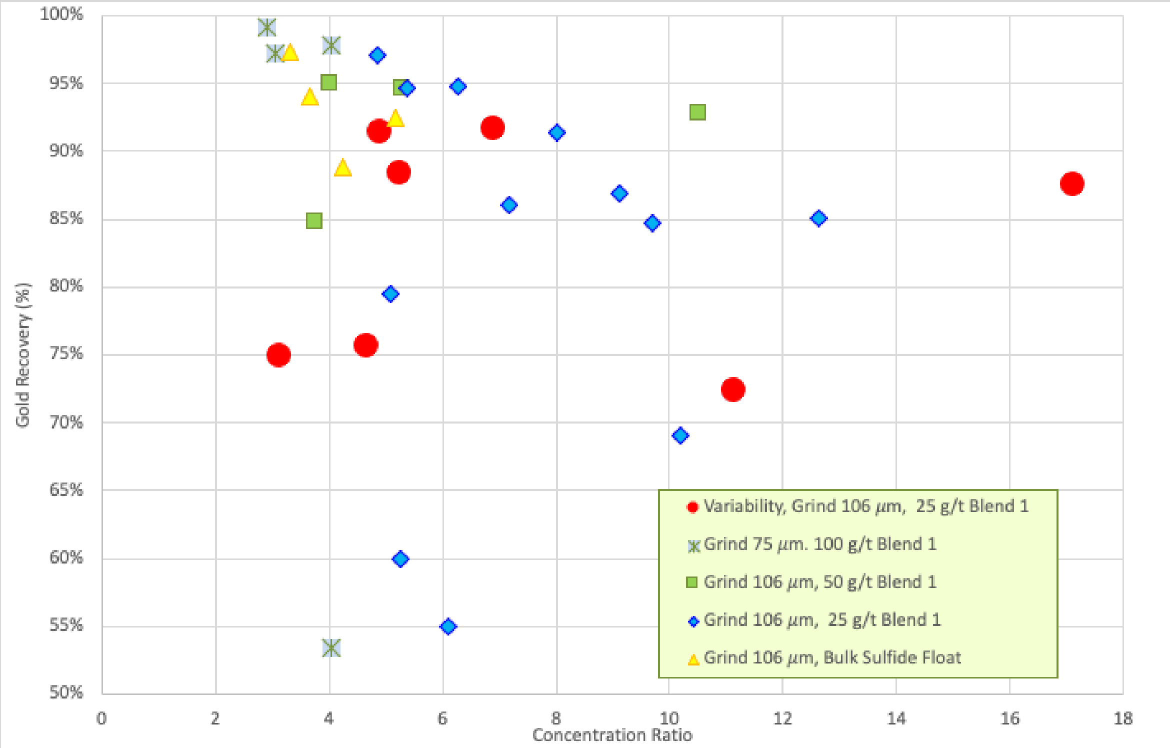
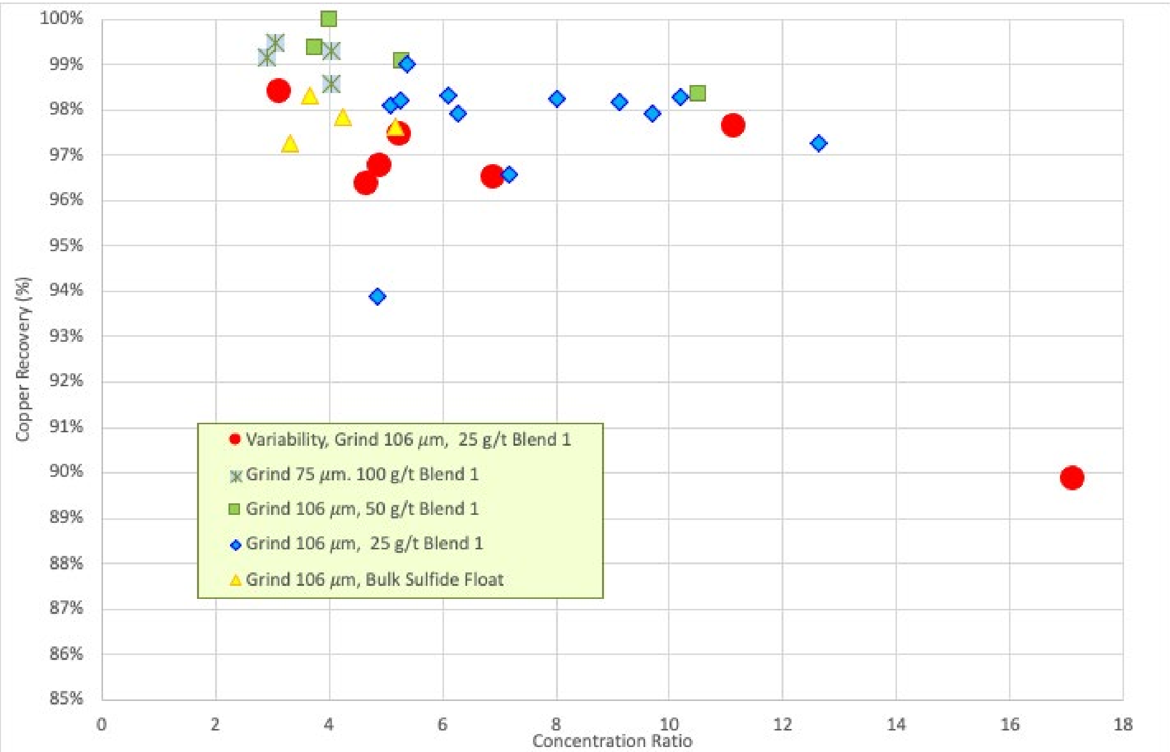


Figure 13.10.2 Copper Recovery versus CR for All Rougher Floats Using Blend 1 Collector



Rate of Flotation Comparisons

The flotation rates of gold and copper for the variability samples are shown in Figure 13.10.3 and Figure 13.10.4 respectively. Results for the belt cut samples (K1, K2, Judd, and the Master Composite) are also shown for comparison.

Figure 13.10.3 Gold Flotation Kinetics of the Variability Samples

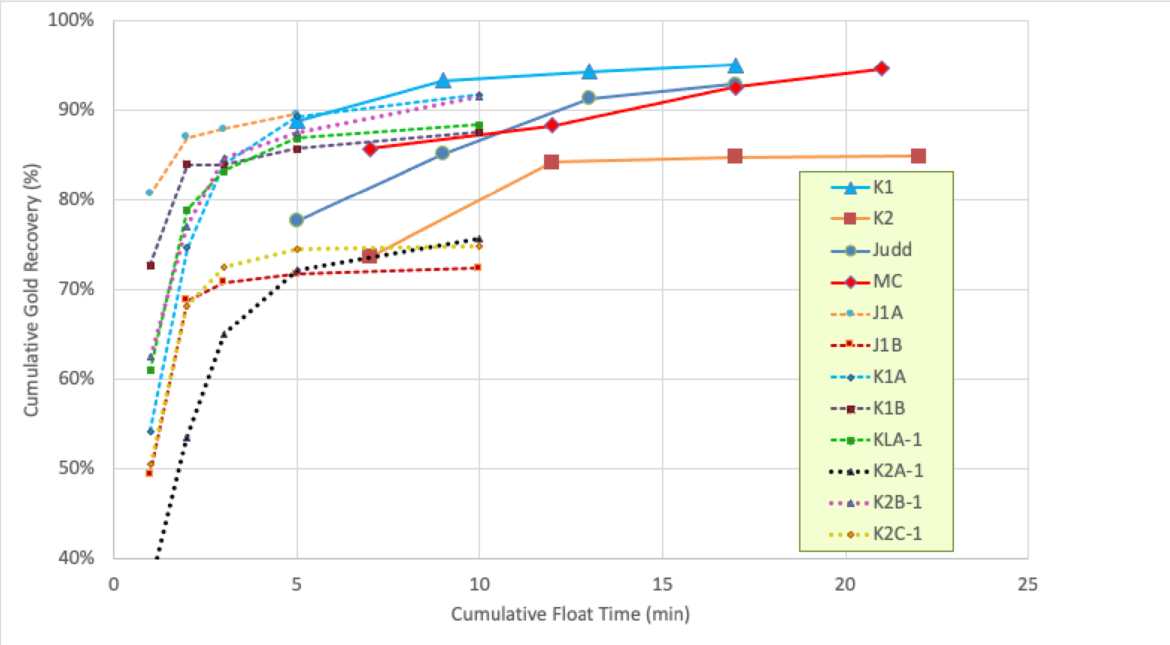
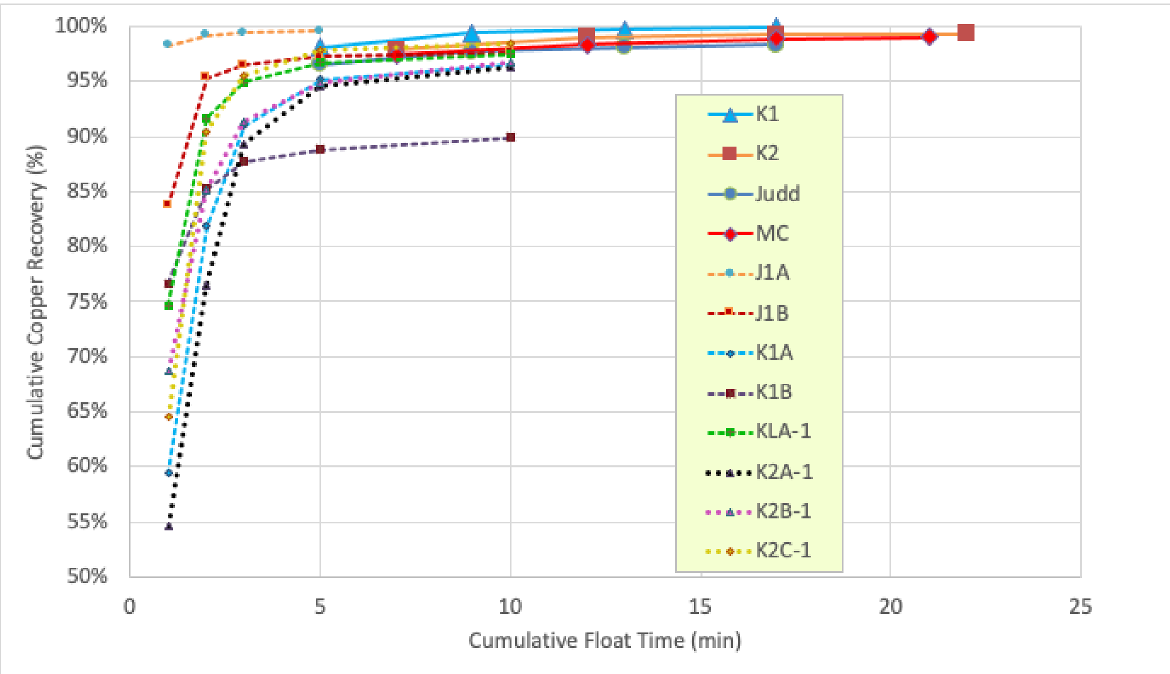


Figure 13.10.4 Copper Flotation Kinetics of the Variability Samples



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 13.8.3. 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'.

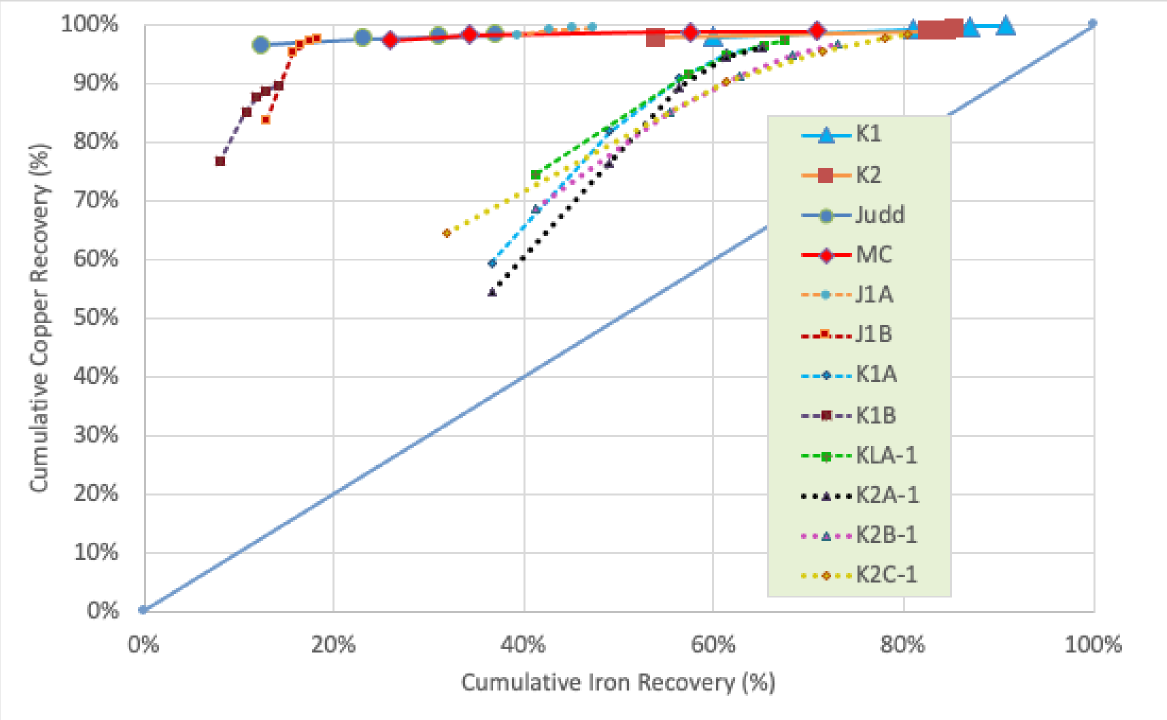
Selectivity for Copper versus Iron Flotation

Figure 13.10.5 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 13.10.1 that the two samples with the most selective flotation response have the highest fluorine in concentrate levels (of the variability samples).

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

Figure 13.10.5 Copper versus Iron Flotation Selectivity for the Variability Samples



13.11 Evaluation of Current Operating Results

Ore Sources

Over the last two quarters of operations, ore has been sources as shown in Table 13.11.1.

Table 13.11.1 Ore Sources During Q4 2021 and Q1 2022

	Q4 2021	Q1 2022
K1	49.4%	46.9%
K2	16.7%	16.0%
J1	33.8%	35.1%
Other	0.0%	2.0%

Operating Data

Operating data for the existing K92 Concentrator is shown below in Table 13.11.2. This is quarterly data for the period from January 2021 to March 2022, with a single month of data shown as 2022 Q2. At the beginning of the period, the plant was being fed ore from the K1 orebody and a small quantity of ore from mining development work on the K2 orebody. The proportion of feed from K2 progressively increased and development ore from Judd came in towards the end of the period.

For the first two quarters of 2021, both mining and concentrator operations were severely affected by personnel shortages caused by the COVID pandemic. The situation started to improve in the third quarter of 2021 and more tonnes were mined. 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 commissioned in November 2021. Parallel operation of the gravity gold recovery and flash flotation can be expected to result in improved overall gold recovery.

Table 13.11.2 K92 Existing Plant, Quarterly Operating Data from January 2021 to March 2022

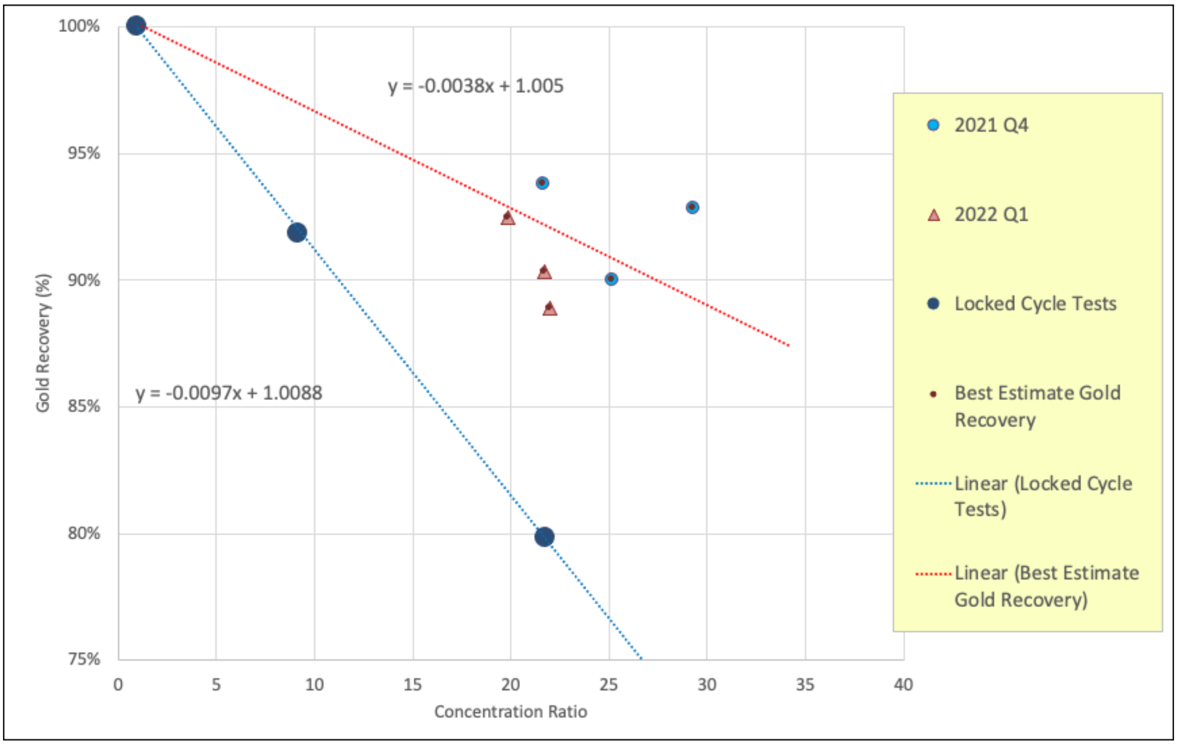
	2021				2022
	2021 Q1	**2021 Q2**	**2021 Q3**	**2021 Q4**	**2022 Q1**
Milled Tonnes (t)	73,221	75,667	87,621	99,713	99,611
Mill Throughput (tph)	41.4	38.5	45.4	53.9	54.9
Head Grade Au (g/t)	8.51	10.32	9.03	11.16	8.30
Head Grade Cu (%)	0.31	0.76	0.48	0.51	0.76
Contained Metal Au (oz)	20,027	25,094	25,439	35,776	26,580
Contained Metal Cu (t)	226	572	418	512	760
Recovery Au (%)	88.7%	88.3%	86.1%	92.9%	90.9%
Recovery Cu (%)	86.2%	87.2%	87.2%	92.9%	91.1%
Recovery Ag (%)	54.7%	64.6%	77.3%	85.5%	76.2%
Concentrate Tonnes (t)	4,873	4,228	3,365	4,009	4,699
Concentrate Grade Au (g/t)	147.6	163.0	202.6	257.7	156.5
Concentrate Grade Cu (%)	5.3	11.8	10.9	11.8	14.7
Concentrate Grade Ag (g/t)	62.9	109.7	182.4	218.9	168.0
Concentration Ratio	15.0	17.9	26.0	24.9	21.2

Gold Recovery versus Concentration Ratio: Comparisons with the Locked Cycle Testwork

Gold recoveries and concentration ratios from the 2021-Q4 and Q1-2022 plant data are shown with the results of the locked cycle tests in Figure 13.11.1. Two results have been extracted from Table 13.9.4, covering rougher-cleaner-recleaner flotation for the two locked cycle tests. A third point, corresponding to 100% recovery when the concentration ratio is unity, has been added. A linear regression line has been drawn through these points. The regression line is a valid estimate of closed-circuit flotation performance based on the locked cycle floats.

All quarterly operations data points plot above this regression line. The gold recovery performance the plant is clearly well above the locked cycle test performance. 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.

Figure 13.11.1 Gold Recovery versus Concentration Ratio for Plant Operations and the Locked Cycle Tests



The difference in recovery is about eight percentage points at a concentration ratio of 15. This can be taken as coarse gold recovery by a combination of flash flotation and centrifugal concentration.

As indicated in Figure 13.11.1, the recovery difference appears to increase with increasing concentration ratio. This could be also explained by coarse gold recovery considerations; at higher concentration ratios, it will be increasingly difficult to recover coarse gold particles to the froth product in conventional flotation. Flotation conditions to achieve high concentration ratio will adversely affect coarse particle recovery.

Recommended Relationship Between Gold Recovery and Concentration Ratio

The new concentrator design will combine process knowledge from both the operation of the existing plant and the current 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 unusually 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 deliver better performance (in terms of value recovery versus CR) than the existing concentrator.

A best estimate of gold recovery performance is required for financial evaluation. This estimate must still be robust and slightly conservative. However, as noted above, it must also show equal or higher performance than currently achieved in the existing plant. Based on detailed consideration of monthly and quarterly data, MMS Pty Ltd recommends performance predictions based on the red dashed line, 'Linear (best estimate of gold recovery)' shown in Figure 13.11.1. This is the least squares best fit line for the gold recovery data for Q4 2021 and Q1 2022, that has been forced to go through to point for 100% recovery at a CR of 1.0.

This line corresponds to the equation:

Equation 4: Gold recovery (%) = 100 * ((-0.0038 * CR) + 1.005)

For example, with the reserve ore head grade of 6.7 g/t and a concentration ratio of 20, the gold recovery will be 92.9% and the concentrate grade with no gravity gold recovery will be 124.5 g/t. After allowing for 15% gold recovery by gravity, the effective head grade will be 5.695 g/t and the concentrate grade will be 105.8 g/t.

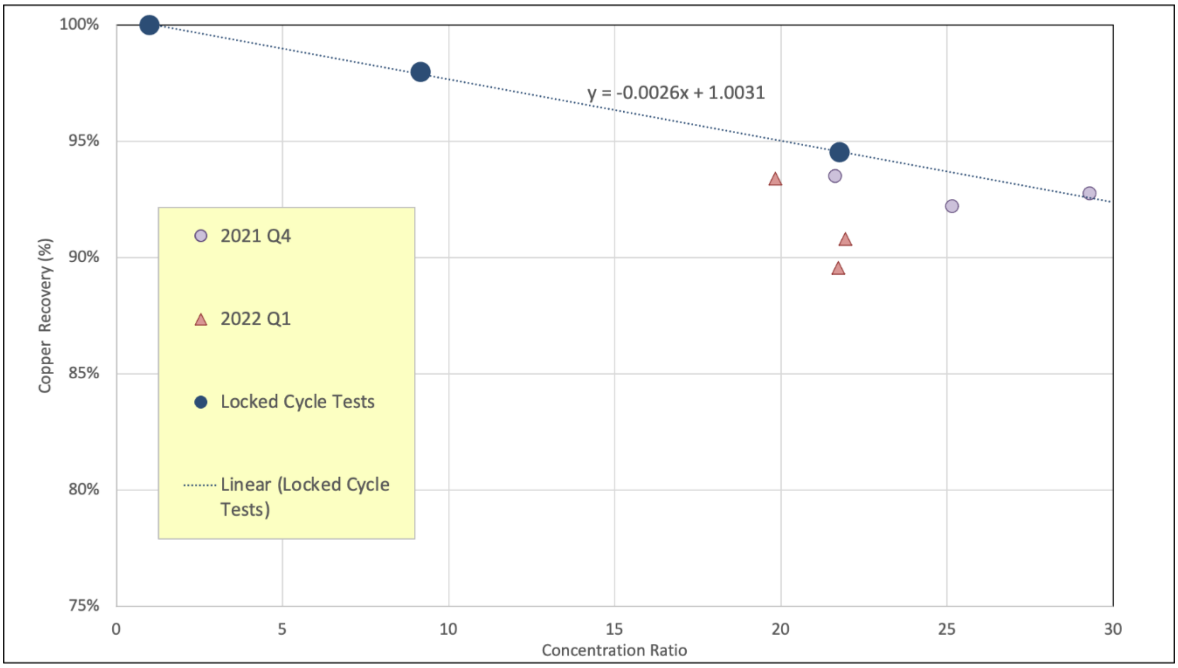
With the plant controlled to achieve a target gold value in concentrate (e.g., target range 100 to 110 g/t), the predicted gold recovery will depend on the head grade of ore fed to the plant. Suitable calculations will translate the grade and tonnage of ore in the monthly mining schedule to gold contained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

Copper Recovery versus Concentration Ratio: Comparisons with the Locked Cycle Testwork

Copper recoveries from monthly plant operating data (separated into quarterly intervals) plus the results of the locked cycle tests are shown below in Figure 13.11.1. Two locked cycle results have been extracted from Table 13.9.4 these cover cleaner flotation for the two tests. A third point, corresponding to 100% recovery when the concentration ratio is unity, has been added. The linear regression line drawn through these points is a valid estimate of closed-circuit flotation performance.

All except one of the monthly operations data points plots below this regression line. However, the copper recovery difference between plant and laboratory results is much smaller than for gold recovery. The copper recoveries achieved in the plant are about five percentage points below the locked cycle test performance.

Figure 13.11.2 Copper Recovery versus Concentration Ratio for Plant Operations and the Locked Cycle Tests



Recommended Relationship Between Copper Recovery and Concentration Ratio for the DFS

A best estimate of copper recovery performance is required for financial evaluation of the project. This estimate must still be robust and slightly conservative. However, as noted above, it must also show higher performance than currently achieved in the existing plant. Copper recovery performance can be estimated using the same considerations as used for gold. In the case of copper, the existing plant has no inherent recovery advantage over laboratory flotation.

MMS Pty Ltd recommends performance predictions based on the blue dashed line, 'Linear (Locked Cycle Tests)' shown in Figure 13.11.2. This is the regression line from the locked cycle test results.

This line corresponds to the equation:

Equation 5: Copper recovery (%) = 100 * ((-0.0026 * CR) + 1.0031)

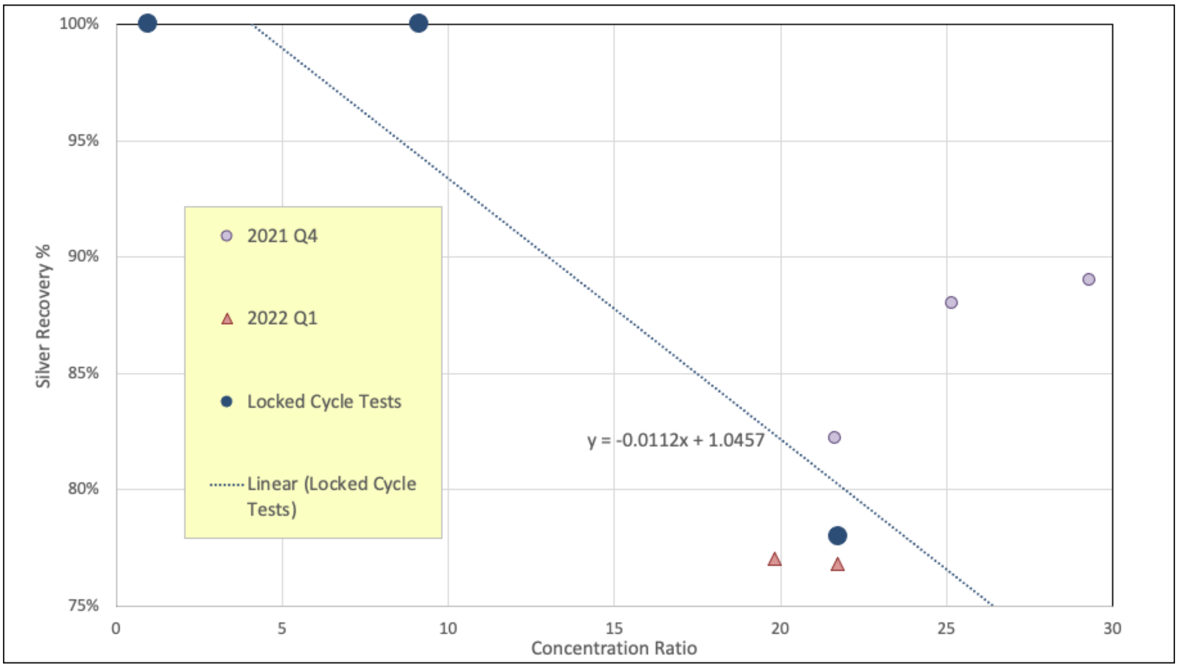
With the plant controlled to achieve a target gold value in concentrate (e.g. target 120 g/t), the predicted copper recovery will depend on the gold head grade of ore fed to the plant. Suitable calculations will translate the grade and tonnage of ore in the monthly mining schedule to copper contained in monthly concentrate production.

Taking the previous example; with an ore head grade of 6.7 g/t and a concentration ratio of 20, the gold recovery will be 92.9% and the concentrate grade with no gravity gold recovery will be 124.5 g/t. After allowing for 5% gold recovery by gravity, the effective head grade will be 6.365 g/t and the concentrate grade will be 118.3 g/t. From Equation 5, the copper recovery will be 95.11%. If the copper head grade was 0.75% Cu, then the concentrate grade would be 14.26% Cu.

Recommended Relationship Between Silver Recovery and Concentration Ratio

The silver results are very scattered and the relationship between concentration ratio and recovery is not clear. For financial evaluation, a silver recovery of 80% is recommended. This is a round number that is close to the average recovery for Q4 2021 and Q1 2022.

Figure 13.11.3 Silver Recovery versus Concentration Ratio for Plant Operations and the Locked Cycle Tests



13.12 Process Selection and Design Process Plant

13.12.1 Comminution Circuit Selection

Following review of the available comminution testwork, along with information about the existing crushing and grinding circuit, Orway Mineral Consultants (OMC) were engaged to undertake power modelling on four circuit options that were considered feasible.

The comminution circuit selected for the new processing plant was primary crushing with a SAB grinding circuit. Two stage grinding provides stable operation if the ore is consistent. This type of circuit is also relatively straightforward to operate, and control is well accepted and understood across the industry. A disadvantage of this option is that if the ore is variable, the power split between the SAG and ball mills can be challenging to manage. If the ore becomes harder, then the throughput can be limited by the capacity of the SAG mill. The preferred mill sizing by K92 was a smaller diameter for easier transportation, and an equal sized motor for both mills to increase maintenance flexibility. Therefore, the comminution circuit selection was a lower aspect SAG mill with common SAG and ball mill installed power. A pebble crusher is not required for two stage grinding circuit.

13.12.2 Gravity Circuit Selection

A gravity concentration circuit for recovering free gold has also been incorporated into the grinding circuit. The preferred arrangement by K92 is for the cyclone underflow to split into three streams between the gravity circuit, flash flotation and ball mill feed. To assist with maintaining an appropriate ball milling density, the gravity concentrator tailings stream is returned to the cyclone feed hopper. Since K92 is committed to not using cyanide, gravity shaking tables are employed to further upgrade the gravity concentrate before it is calcined and smelted on site to produce doré.

13.12.3 Flotation Circuit Selection

Conventional flotation follows crushing and grinding to produce a sulphide concentrate from the conventional flotation circuit. A flash flotation cell is also incorporated into the grinding circuit with the flash flotation concentrate combining with the conventional flotation circuit concentrate. This concentrate will be thickened, filtered and dried before being loaded into shipping containers and transported to smelter(s).

Based on the metallurgical testwork results, the conventional flotation circuit will consist of rougher and scavenger flotation stages, followed by cleaner, cleaner scavenger and recleaner flotation stages. The circuit includes recycling of the concentrate stream from the cleaner scavenger cells and the tailings stream from the recleaner cells back to the cleaner cells.

Flotation concentrate regrind was considered, but ultimately not included in the flotation circuit design. The testwork demonstrated an improvement in the flotation concentrate gold grade, but this was achieved at the expense of the gold recovery. Additionally, regrinding did show a benefit by reducing the fluorine concentration in the flotation concentrate. Fluorine is a penalty element for smelting terms and can result in a cost penalty if the fluorine concentration is above the limit. However, the fluorine concentration in the flotation concentrate can also be reduced by addition of depressant flotation reagent. Therefore, the flotation circuit does not include a regrind mill.

13.12.4 Process Plant Design

Design criteria, mass balance and process flow diagram(s) have been developed as a basis for preparing the capital and operational expenditure estimates. The process and plant description is detailed in Section 17.

The design criteria developed for a nominal throughput of 1.2 Mtpa (dry) with a head grade of 1.0% Cu and 10.0 g/t Au produces 56,000 tpa high grade gold-copper concentrate. The design criteria is based on Owners' information, current site operating experience and industry practice. Process flow diagrams have been developed covering:

- Crushing and grinding.
- Flotation, concentrate thickening and filtration.
- Reagent mixing and distribution.
- Plant air and water services.

Mass and water balances for the process streams were developed based on the design criteria and proposed process flow diagrams.

13.13 Paste Laboratory Testwork

13.13.1 Introduction

This section presents the results of all laboratory testwork undertaken on materials used in the investigation. Testwork presented includes material characterization, rheology testing, dewatering testwork and cemented strength testing. All testwork was undertaken in accordance with relevant Australian standards and, where relevant, specific Outotec Mine backfill testwork standards, which are available upon request.

13.13.2 Characterization

This section sets out characterization information relating to the tailings, water and cements used in the testwork.

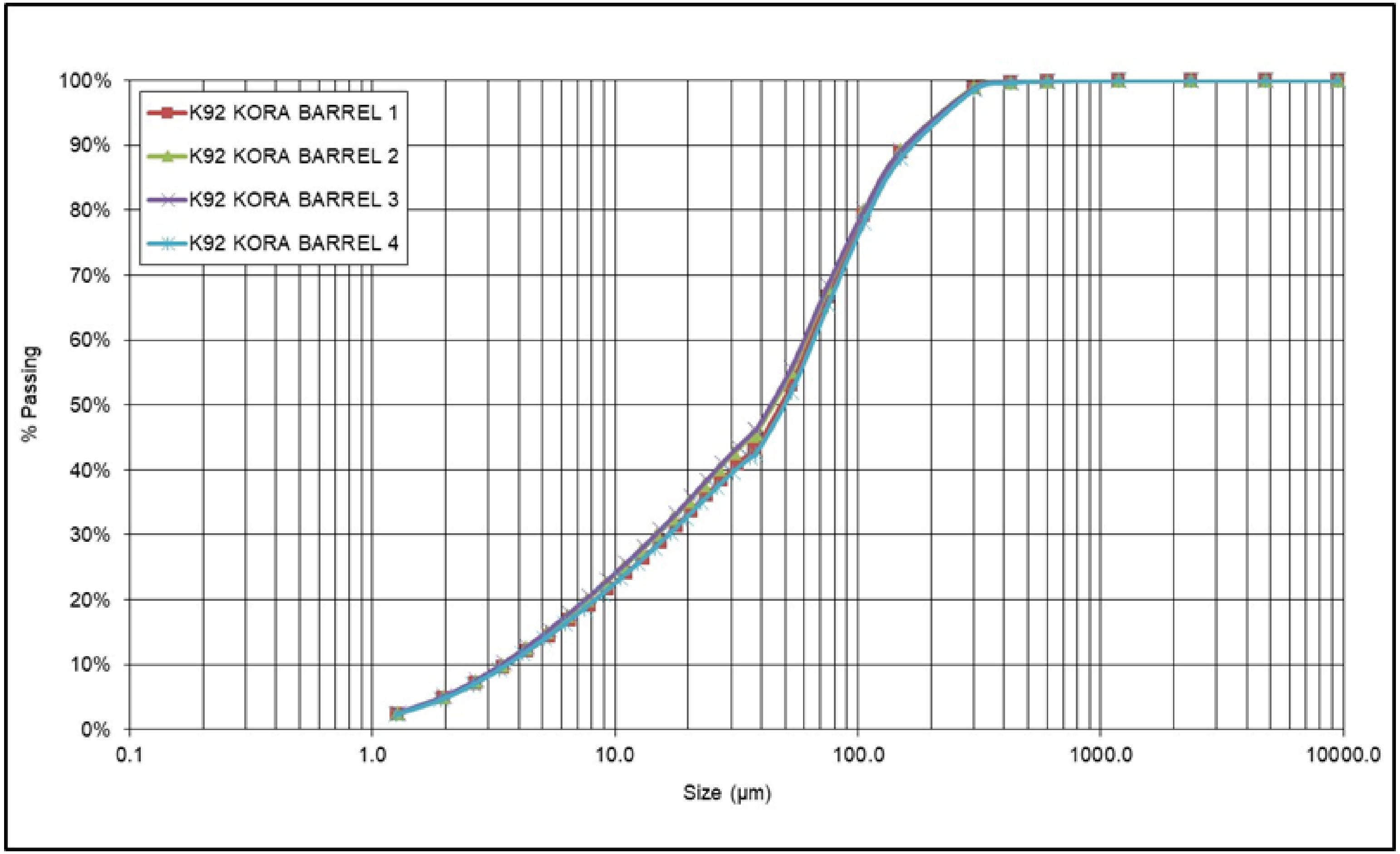
Tailings Characterization

Material used in this testwork campaign was supplied to MOG by K92. The full stream tailings sample arrived submersed in and IBC container. The water was decanted, and the tailings combined to form a bulk sample, that was used for all testwork. For handling purposes this bulk sample was divided into four identical sub-samples.

Tailings Particle Size Distribution (PSD)

To determine the particle size distribution of the tailings material used in this study, the tailings were first 'wet sieved' through a 38 µm sieve. The -38 µm material was laser sized, while the +38 µm material was dry sieved. The measured PSD of the full tailings samples collected from each of the sub-sample buckets is presented in Figure 13.13.1

Figure 13.13.1 PSD of Tailings Sample Used in this Testwork



Industry 'rule of thumb' to create paste is to have at least 15-20% passing 20 µm and the PSD of the full stream tailings for Kora shows 35% passing 20 µm. The significant portion of material finer than 20 µm is an indication that the Kora tailings are towards the finer end of material typically used for manufacturing paste. While this is not expected to pose any technical issues in relation to the paste, the fine material is likely contributing to the dewatering challenges described later in this Section.

It is acknowledged that the tailings adopted in this testwork represent a 'snapshot' of Kora tailings, and that the basis for design presented in this report is linked to these material properties. While some variation of this material may occur over time, over the range of variability historically experienced at Kora, we believe that the included risk management provisions (such as provision for high floc dosage rates) are suitable for managing this variability.

Tailings Mineralogy

To investigate the mineralogy of the Kora full stream tailings, the tailings were analysed using quantitative x-ray diffraction (QXRD). The results of this analysis are summarized in Table 13.13.1.

Table 13.13.1 Mineralogy of Tailings used in Testwork

Kora Tailings	
Phase	**Weight %**
Quartz	53.5
Muscovite	21.2
Pyrite	10.3
Chlorite	7.5
Biotite	3.2
Microcline	2.1
Bassanite	0.8
Actinolite	0.5
Chalcopyrite	0.3
Albite	0.3
Dolomite	0.2
Talc	0.1
Sphalerite	0.1

Table 13.13.1 shows the Kora tailings to consist primarily of Quartz and Muscovite. Typically, Quartz is a favourable mineral for paste backfill production. The presence of sulphides, such as Pyrite, has the potential to create longer term cemented strength deterioration issues, but of the problematic sulphides, Pyrite tends to be less active. Longer-term strength testing presented in Section 13.13.5 shows that long term strength degradation does not occur. While the Talc and Sphalerite portions presented in Table 13.13.1 do not present a problem. It is the opinion of the author of this section that these portions would have to increase significantly (above 2%) before issues associated with filtration and transportability would arise.

Tailings Specific Gravity (Gs)

The specific gravity of the Kora full stream tailings was measured in accordance with MF01.07 and the results are presented in Table 13.13.2.

Table 13.13.2 Specific Gravity of Tailings used in Testwork

Sample	Specific Gravity (t/m^3)
Full stream tailings	2.86

Mix Water Characterization

Chemical analysis was undertaken on the Kora process water and results are presented in Table 13.13.3.

Table 13.13.3 Mix Water Chemical Analysis Results

Analyte	Kora Process Water	Units
Ca	482	mg/L
Cl	6	mg/L
Electrical Conductivity	2,490	µS/cm
Fe	<0.05	mg/L
K	138	mg/L
Mg	14	mg/L
Na	65	mg/L
SO4	1,600	mg/L
Total Solids in Suspension	5	mg/L
TDS	2,360	mg/L
Density	1.0	t/m^3
Turbidity	0.7	NTU
pH	7.08	

The results presented in Table 13.13.3 indicate that the Kora process water appears very favourable for paste production. Nothing identified in the process water raises concerns around the influence on binder hydration.

Binder Products Characterization

Strength testwork carried out as part of this investigation was undertaken using various combinations of General Portland Cement, Ground granulated blast furnace slag, hydrated lime and quicklime. To benchmark this binder, a series of 'Standard Cement Prism' tests were undertaken in accordance with AS/NZS 2350.11, 2006. The test results for the binder are presented in Table 13.13.4.

Table 13.13.4 Cement Prism Testing on GP Cement used in this Testwork

Binder Type	Binder Prism Testing (AS/NZs 2350.11 2006) MPa		
	3 Day	7 Day	28 Day
GP Cement (from Drillcube)	31.6	43.8	64.1

13.13.3 Tailings Dewatering Testwork

Thickening

Thickener Test Method

To assess the performance of dewatering through thickening a series of thickener tests were undertaken. The purpose of the testing was to determine:

- flocculant type and dose
- overflow clarity
- underflow density
- underflow yield stress.

The thickener test rigs used in this test campaign are the 99 mm diameter high-rate thickener and the 190 mm diameter high-compression / paste thickener. Photographs showing these rigs are presented in Figure 13.13.2.

Figure 13.13.2 (a) 99 mm and (b) 190 mm Thickener Test Apparatus

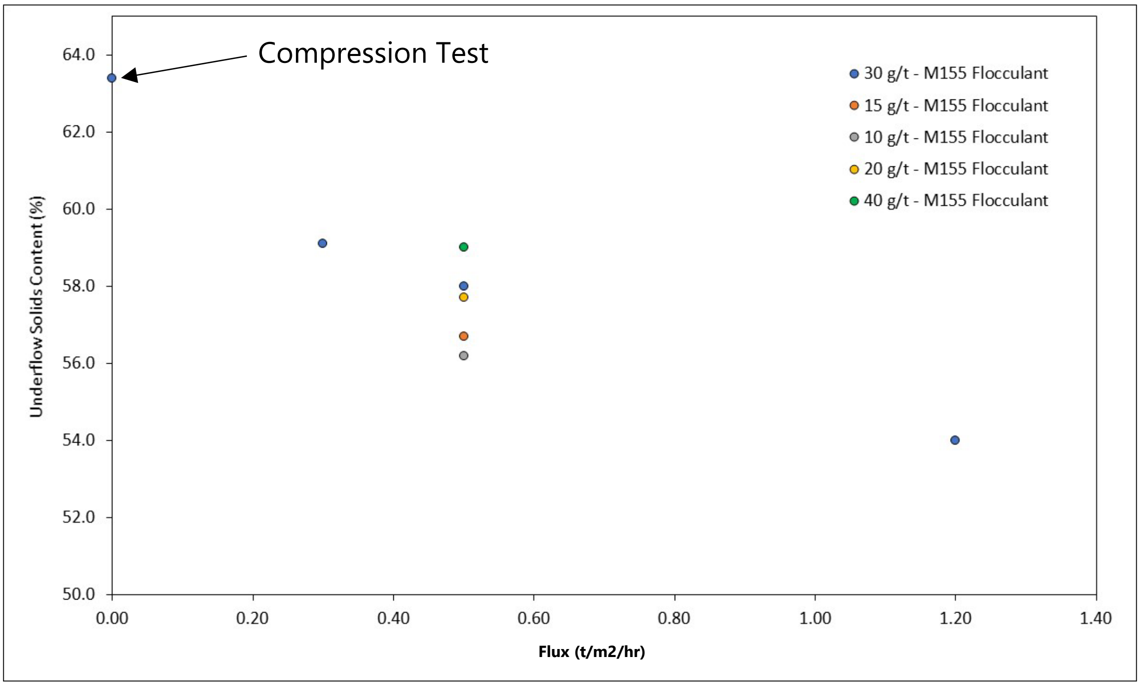
(a) **(b)**

Thickener Test Results

Initial Flocculant screening tests were undertaken, and this work showed the M10 and M155 flocculants to provide a similar performance. Based on this the M155 was adopted in the thickener testwork.

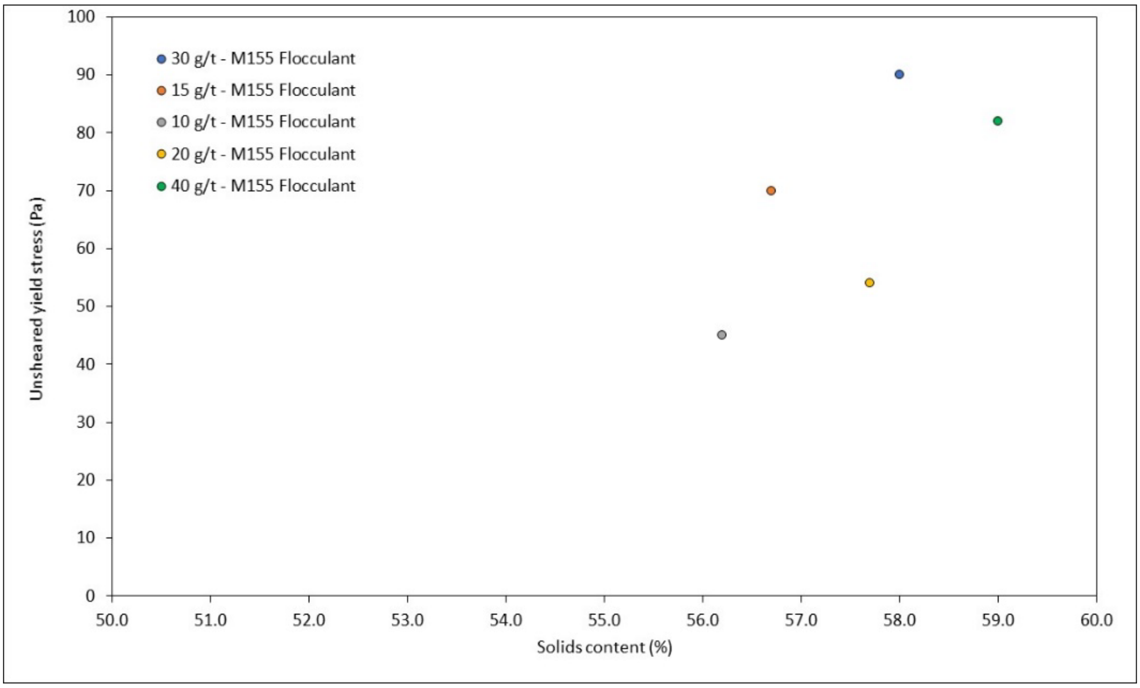
Testing was undertaken with flocculant dosage rates ranging from 10 to 50 g/t and with flux rates of 0.3-1.2 $t/m^2/hr$. The results of the thickener testwork are presented in Figure 13.13.3, which present the thickener underflow solids concentration versus flux rate. In addition to the conventional (High Rate Thickener) thickening testwork, a 'compression test' was also completed. This test was undertaken to determine the additional compression that could be achieved if thickener technology with a thicker bed depth (such as a High Compression or Paste thickener) was adopted. The compression test is plotted at a flux rate of 0 $t/m^2/hr$.

Figure 13.13.3 Thickener Underflow Solids Content Versus Flux



The results presented in Figure 13.13.3 show that a thickener underflow solids content of 56-59% solids (w/w) is achievable using a conventional high-rate thickening. The results of the 'compression' test (used to provide an indication of what might be achievable from a paste thickener) indicates that an underflow solids content in the order of 62-64% solids (w/w) may be achievable. The measured thickener underflow solids content is plotted against 'unsheared' yield stress in Figure 13.13.4. It should be noted that these yield stress values were measured on the material immediately after sampling from the test rig and as such include the influence of undamaged flocculants. Shearing through centrifugal pumping would destroy these flocculant bonds, reducing the yield stress considerably.

Figure 13.13.4 Yield Stress Versus Thickener UF Solids Content



As a paste thickener was being considered as a potential solution for Kora, it was necessary to undertake large scale thickener testing using the Outotec 190 mm dynamic thickener test rig. The 190 mm dynamic testwork was run continuously to achieve a flux rate of 0.5 t/m^2/hr while maintaining the target bed height of 3.0 m. Once 2 – 3 complete bed replacements had occurred and underflow sampling were able to produce consistent underflow density, the test run would be considered at steady state. Steady state samples collected from the underflow and the measured properties are summarized below:

- Underflow solids content – 59% solids (w/w)
- Unsheared yield stress – 20 Pa
- Fully sheared yield stress – 8 Pa.

This result indicates that, contrary to the original sighter 'compression test', little if any benefit is achieved through progressing from a High-Rate Thickener through to a High Compression or Paste Thickener. As such vacuum filtration is considered necessary for generating a paste with tangible yield stress.

Filtration

Filtration Test Method

To assess the dewatering equipment requirements for the Kora paste plant, filtration testwork was undertaken. Due to site space restrictions and expected low filtration capacity of the fine Kora tailing, a filtration method with small footprint was required. As such the only test method considered was disc vacuum filtration. The test method adopted for the disc filtration testing is the 'Scanmec leaf' test, and the associated test equipment is presented in Figure 13.13.5. The Scanmec leaf has a total filtration area of 0.01 m^2.

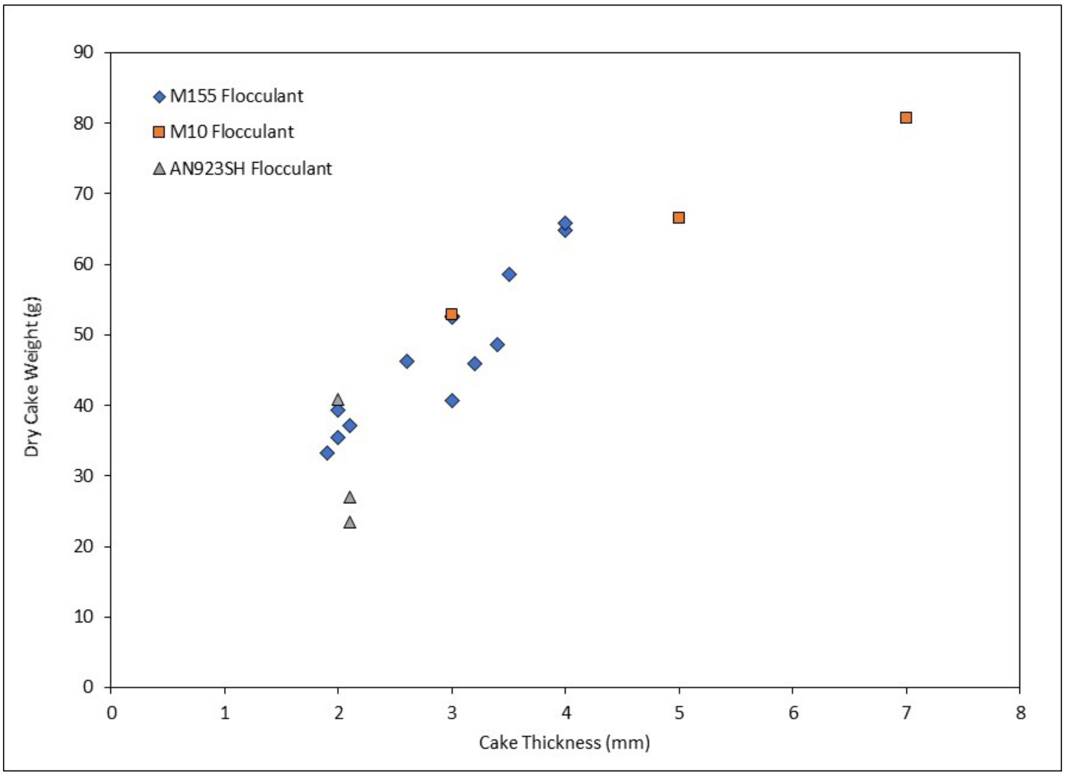
Figure 13.13.5 Photograph Showing the Scanmec Leaf Filter Test

These tests and the results from this work are summarized in the following section.

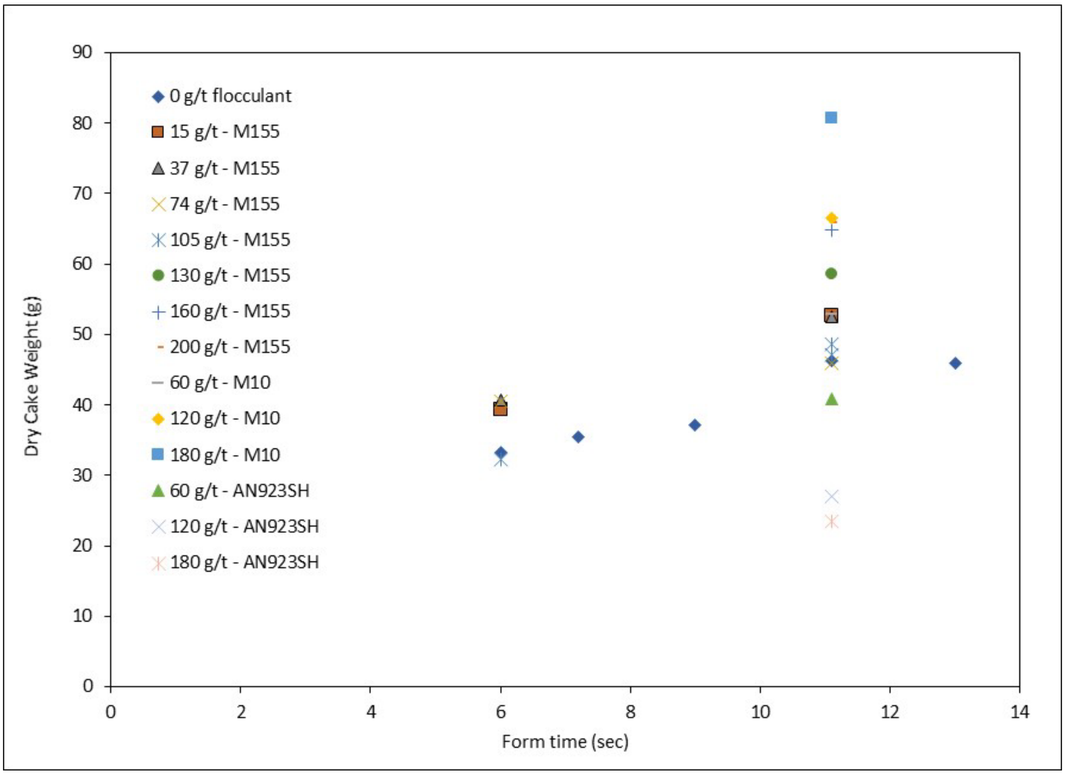
Scanmec Leaf (Disc Filtration) – Full Stream Tailings

Testing with the Scanmec filter leaf considered full stream tailings delivered to the filter over a range of different solids contents. This work considered the influence of different filter rotational speeds and flocculant addition on the performance. All testing was completed using 'needle felt' cloth because, relative to other available cloths, this product provides excellent filtration rates and high durability. The measured relationships are presented in Figure 13.13.6a, b and c. Figure 13.13.6a presents the relationship between cake thickness and dry filter cake mass. Figure 13.13.6b presents the relationship between form time and the dry filter cake mass. Figure 13.13.6c presents the dry time factor versus the cake moisture, where the dry time factor is defined as the dry time (in seconds) divided by the cake weight (g).

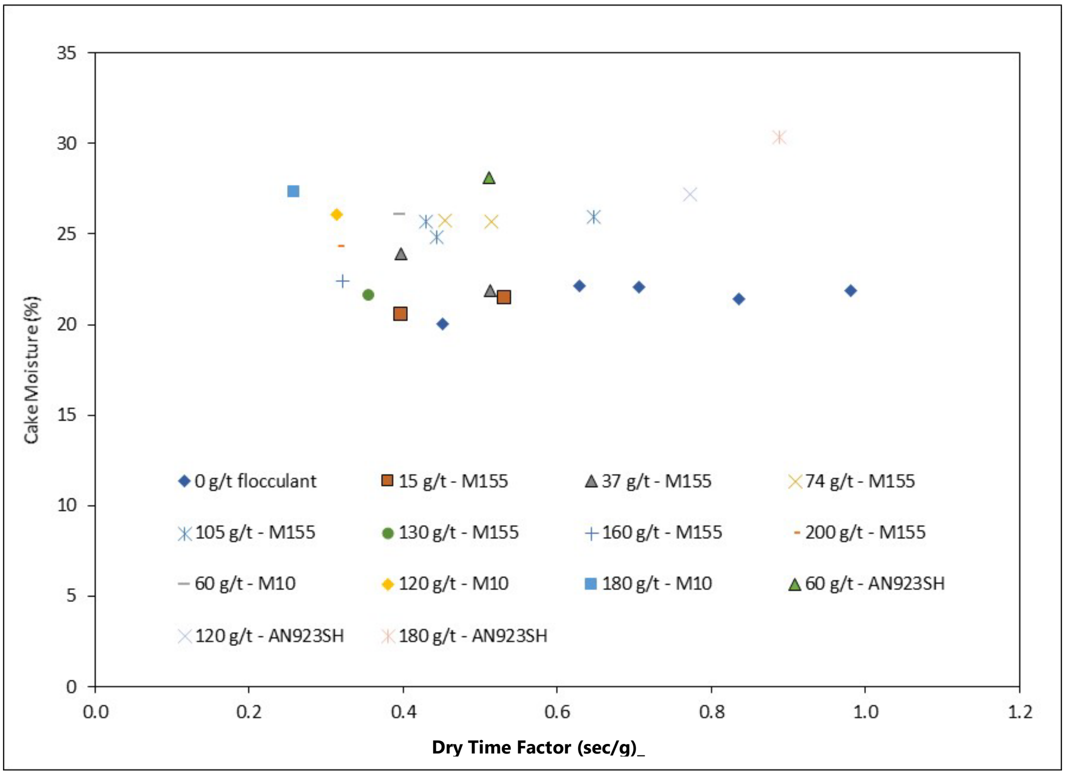
Figure 13.13.6 Dry Time Factor Versus Cake Moisture



(a)



(b)



(c)

Given the results presented in Figure 13.13.6 it is apparent that the addition of flocculant can provide a significant increase in filtration rates. This is further illustrated in Figure 13.13.7, which shows the cake weight versus flocculant dosage for form times of 6 and 11 seconds.

Figure 13.13.7 Cake Weight Versus Flocculant Dosage

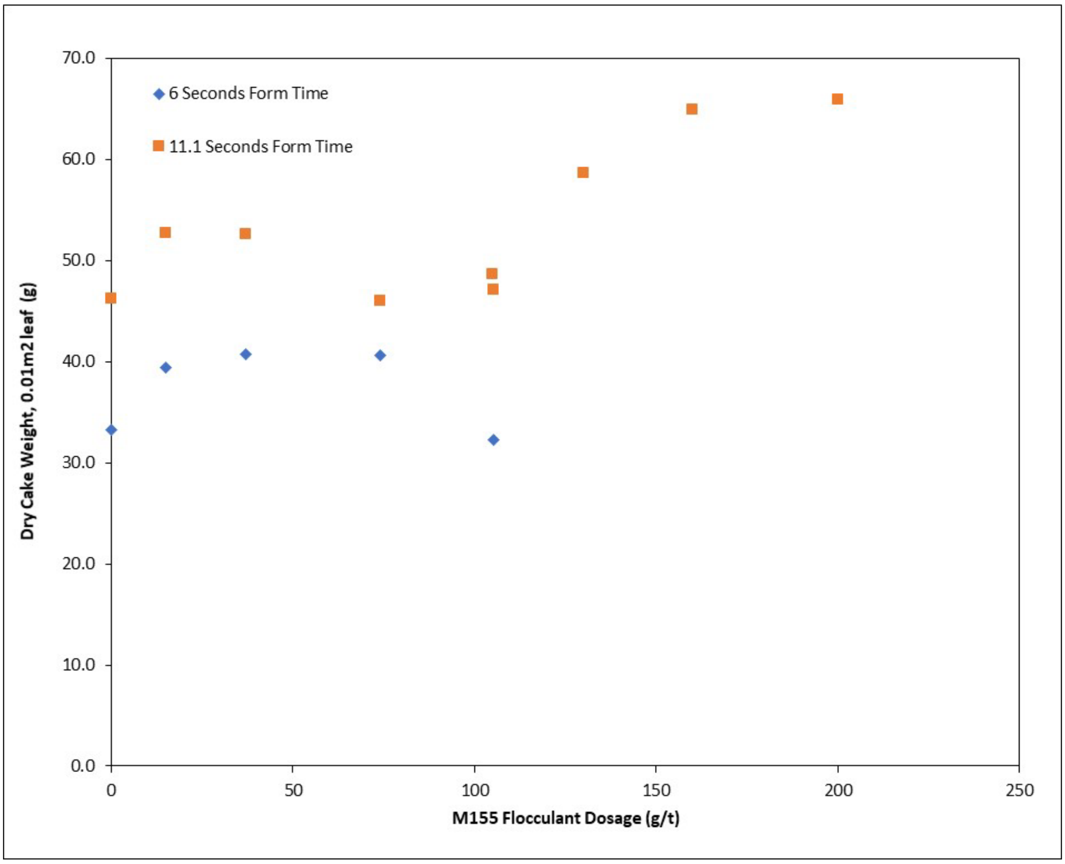


Figure 13.13.7 shows that, for the same form time, the dry cake weight can be increased 40% through flocculant aid. While this increase in filtration rate appears significant it must be recognized that flocculant costs can also be significant and should be assessed against the reduction in capital investment.

Given the results presented above, we believe that its appropriate to design for a moderate flocculant addition of 15-30 g/t.

To assess the influence of filter feed solids content on the resulting filter performance a series of tests were completed with 15 g/t of M155 flocculant and feed solids concentrations of 35, 45 and 55% solids (w/w). The measured filtration rate and cake moisture for each of these is presented in Figure 13.13.8.

Figure 13.13.8 Influence of Feed Solids Content on Filtration Rate and Moisture

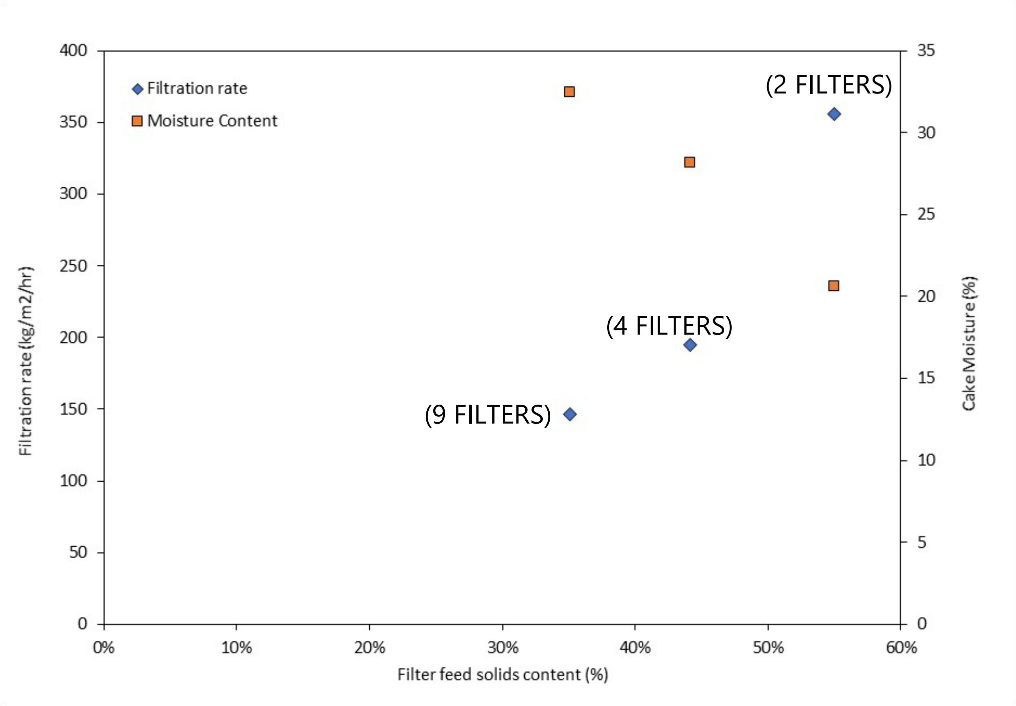


Figure 13.13.8 shows that any increase in filter feed solids content can make a significant influence on filtration rate significantly increases and the cake moisture content.

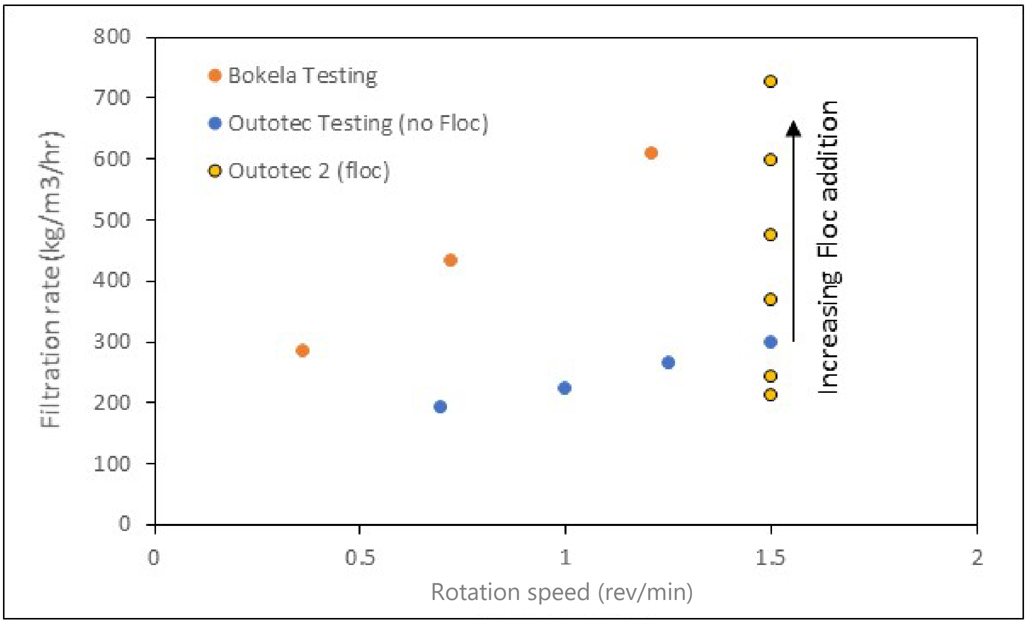
Filtration Test Method

Based on the results presented above it was identified that a large filter was required and the filter considered most appropriate to address this requirements is Bokela's 176L disc filter. To confirm the suitability of this equipment an additional sample was sent to Bokela for testing.

Testing was completed using the same methodology as the 'Scannmec leaf' testing described in the filtration test method above. The Bokela results are plotted as filtration rate versus disc rotation speed in Figure 13.13.9. Superimposed over this data is the filtration rates measured by the MOG, where results with and without flocculant are presented. Flocculant used in all tests is SNF AN-923-SH and this flocculant was diluted to a concentration of 0.025% using potable water. All tests were conducted with a filter feed solids concentration of 58% solids (w/w).

These results show that the addition of flocculant increases filtration rates considerably and that optimal rates measured by MOG correspond with rates measured by Bokela. This comparison provides confidence in the Bokela results and the proposed filter specification.

Figure 13.13.9 Bokela and Metso Outotec Filter Test Results on Kainantu (Kora) Tailings



13.13.4 Rheology Testing

Rheology Introduction

The Kora Project requires significant pumping duties and in order to properly understand the system performance a thorough understanding of the rheological performance is necessary.

To investigate the rheological behaviour a number of bench scale rheological tests were completed and the results from this work was compared with historic measurements on similar materials to develop rational relationships between paste solids content and pipe friction loss.

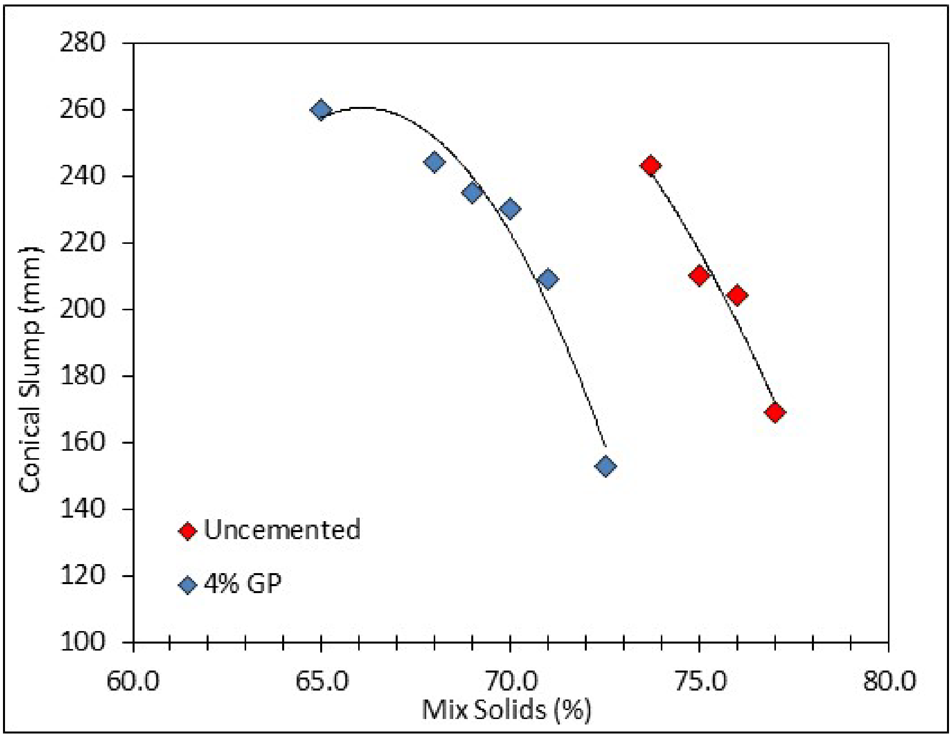
Bench scale testing undertaken on Kora paste included both cylindrical and conical slump testing as well as vane rheometer testing. In addition, both the vane rheometer and the cup and bob apparatus were used to measure the relationship between shear stress and shear rate.

Testing was completed on uncemented paste, paste containing 4% slag and paste containing 4% slag and 1.0% GP cement.

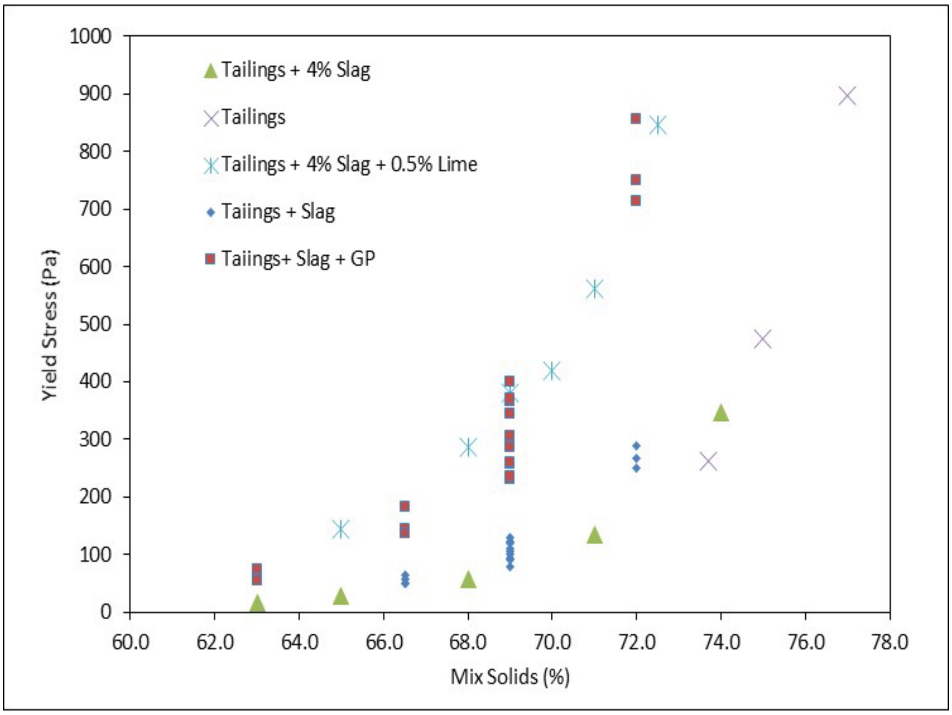
Kora Static Rheological Measurements

To determine the yield stress of paste manufactured from the full stream Kora tailings, rheology testing was carried out using the conical slump and vane rheometer. During this investigation a series of different paste mix solids contents were considered and the measured conical slump is presented in Figure 13.13.10.

Figure 13.13.10 (a) Conical Slump and (b) Yield Stress Versus Mix Solids Content for Full Stream Kora Paste



(a)



(b)

Kora Dynamic Rheological Measurements

In addition to the 'Static' rheology testing a series of dynamic rheology tests were also carried out. This testing was all completed on uncemented Kora tailings. These measurements are expected to be representative of both uncemented and tailings containing slag. The methods used are the 'Cup and Bob' and the 'Infinite Vane' (IQ vane) method (Sofra, 2001). The results are presented as shear stress versus shear rate in Figure 13.13.11.

Figure 13.13.11 Shear Stress Versus Shear Rate for Kora Tailings

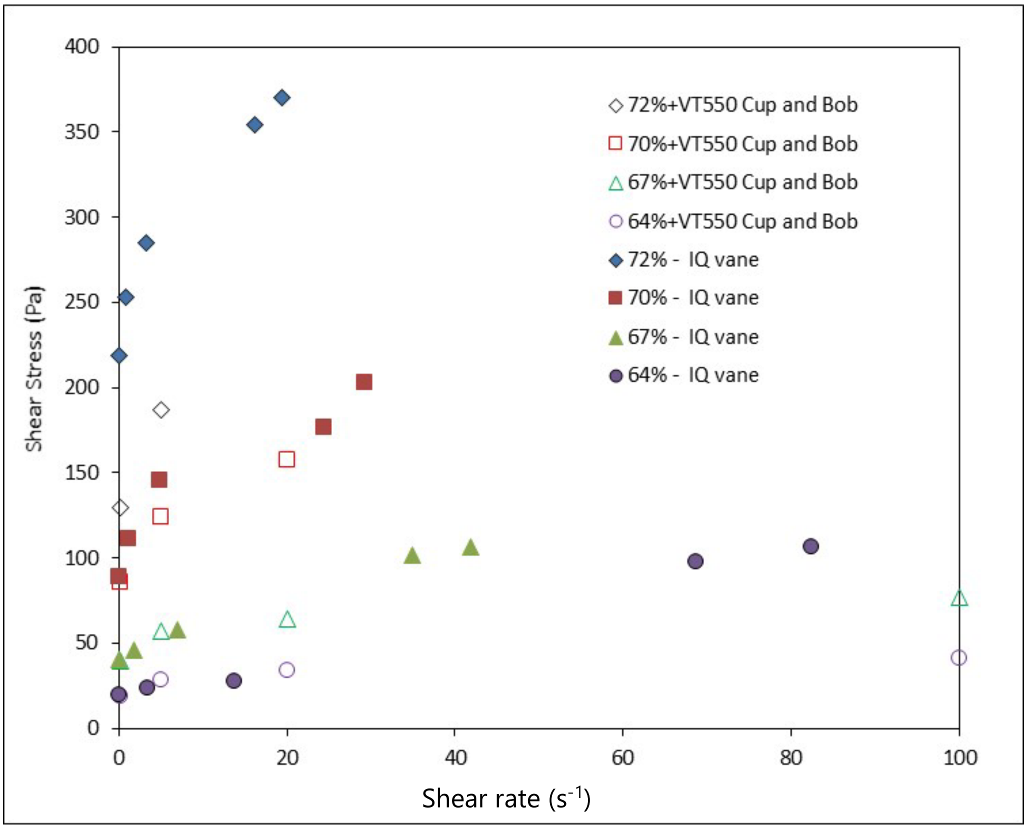
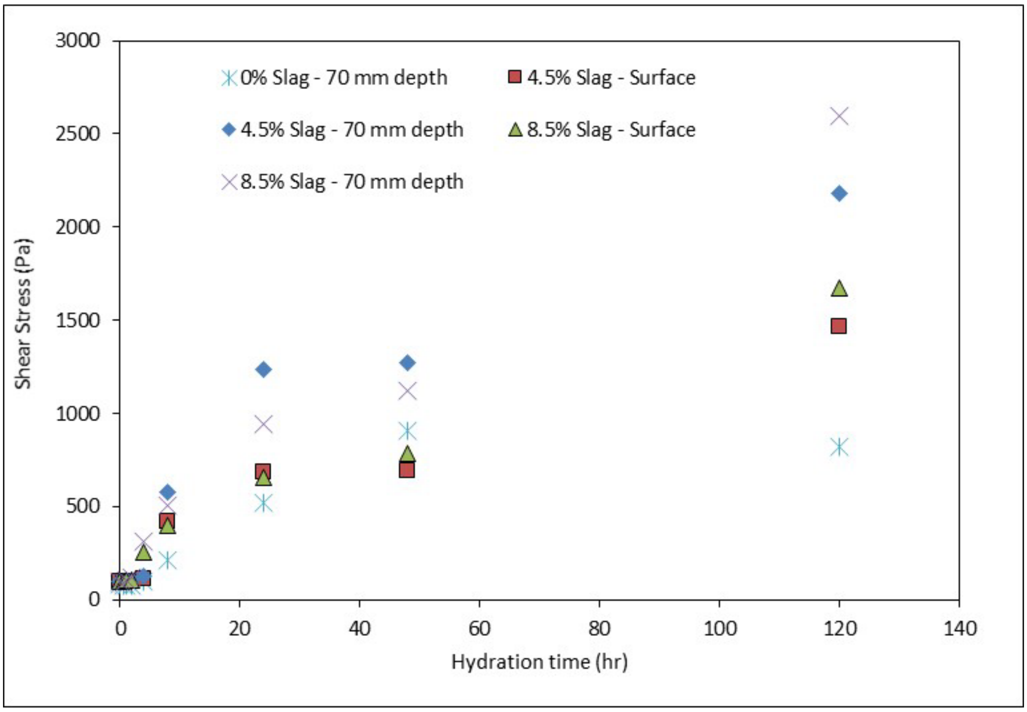


Figure 13.13.11 shows reasonable consistency between the shear stress versus shear rate curves for the Kora paste measured using both the 'Cup and Bob' and 'Infinite vane' methods. Superimposed over the data in Figure 13.13.11 is the expected shear rate within the Kora reticulation system, while shear rates for tank agitators are expected to be closer to 2-5 s^{-1}.

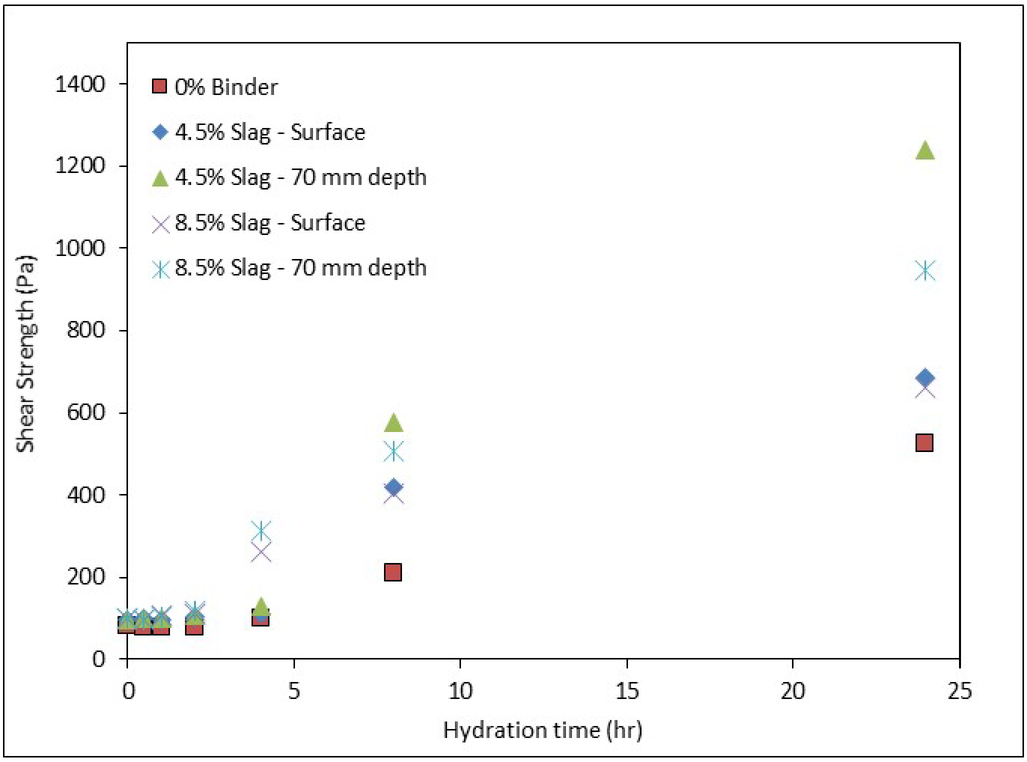
Rheology Over Time

In addition to measuring the paste rheology immediately after mixing a series of tests were also completed to understand the evolution of paste yield stress over time. For this test paste was cast into a large 130 mm thick container. After designated time periods the material was exposed to shear strength testing using the vane rheometer. After each time period the shear strength is measured 10 mm beneath the surface and at a depth 70 mm below the surface. Each test is conducted at a fresh site. The paste mixes tested as part of this experiment were all batched to 69% solids and mixes contained 0%, 4.5% and 8.5% slag. The measured shear strength is plotted against hydration time in Figure 13.13.12.

Figure 13.13.12 Shear Stress Versus Time for Kora Paste with and without Slag with (a) Large and (b) Small Scale



(a)



(b)

The results presented in Figure 13.13.12 show the material to have an initial yield stress of approximately 100 Pa. After the initial 4.0 hours this yield stress doesn't appear to increase for the mixes containing 0 and 4.5% slag, while that containing 8.5% slag increases to approximately 300 Pa. Beyond this point the yield stress levels progressively increase, with the uncemented material plateauing at approximately 700 Pa after 48 hours. Mixes containing slag increase to approximately 1,000 Pa after 48 hrs and 2,000 Pa after 120 hours.

Based on the results presented in Figure 13.13.12 it is expected to be reasonable to leave paste containing 8.5% slag in the lines for up to three hours, while paste containing less than or equal to 4.5% slag (which is expected to be the majority of cases) could remain stationary in the lines for a period in the order of 4-5 hours before being remobilized. In almost all occasions it is expected that Kora paste would contain less than 4.5% slag.

13.13.5 Cemented Strength Testing

Strength Testwork Results

To maintain stability during the proposed mining activities at Kora it is necessary to add binder to the backfill. As the cost associated with binder addition is expected to constitute a major component of the operating costs it is considered critical to properly define the expected binder requirements. This section presents the strength testwork programme undertaken to define the expected relationships between binder addition, hydration time, mix solids content and paste strength. The mine backfill mix combinations adopted in this section were selected based on:

- The rheological behaviour of the material and reticulation geometry.
- The expected system variability and sensitivity of the fill mix rheological properties to changes in solids concentration.
- The horizontal and vertical paste exposure geometries.

The majority of mixes were cast using a blend of Slag and GP cement. Importantly the mixes included a significant Slag addition and smaller portions of GP cement or Hydrated lime. In addition, a small number of mixes included quicklime. The idea behind this approach is based around:

- Past experience showing paste backfill tends to achieve better strengths with higher slag contents.
- A process strategy of adding the majority of the binder product outside of the Portal (a component not expected to result in high strength development) and then add a smaller, amount of 'activation' product (i.e. Lime or GP) within the mine.

The measured fill strengths for all mixes and paste properties are summarized in Table 13.13.5.

Table 13.13.5 Full Stream Tailings Paste Cemented Strength Test Results

Mix #	Binder					Yield Stress (approx.) (Pa)		Mix Solids Content %	UCS (kPa)						Dry Density t/m³	Moisture Content %
	Total Binder %	GP Cement %	Slag %	Hydrated Lime %	Mass per unit volume kg/m³	Before GP/Lime addition	After GP/Lime addition		3 days	7 days	14 days	28 days	56 days	112 days		
1	3.0	-	2.5	0.5	30	26	93	64	0	0	0	30	-	-	1.21	49.9
2	5.0	-	4.5	0.5	53	25	76	64	0	0	0	153	-	-	1.18	50.2
3	9.0	-	8.5	0.5	100	25	72	64	0	0	0	881	-	-	1.18	49.7
4	3.0	-	2.5	0.5	33	93	333	69	0	0	0	176	-	-	1.32	41.9
5	5.0	-	4.5	0.5	60	97	305	69	0	0	0	506	-	-	1.34	40.4
6	9.0	-	8.5	0.5	112	92	279	69	0	0	0	1662	-	-	1.31	40.7
7	3.0	-	2.5	0.5	36	347	1221	73	0	0	48	457	-	-	1.45	33.4
8	5.0	-	4.5	0.5	66	324	1132	73	0	0	609	1626	1626	-	1.46	31.7
9	9.0	-	8.5	0.5	123	348	1119	73	0	143	1957	3380	3380	-	1.45	33.4
13	5.0	-	4.5	1.0	59	119	344	69	0	0	568	1065	-	-	1.31	40.6
14	3.0	0.5	2.5	-	33	94	258	69	60	309	499	586	-	-	1.30	41.3
15	5.5	1.0	4.5	-	58	-	232	69	73	562	1083	1468	-	-	1.30	41.8
16	9.5	1.0	8.5	-	112	91	235	69	117	944	1999	2797	-	-	1.31	40.7
17	6.5	2.0	4.5	-	58	-	260	69	41	548	1137	1431	-	-	1.30	41.5
18	5.5	1.0	4.5	-	59	123	399	69	0	54	-	1499	-	-	1.32	41.7
19	5.0	0.5	4.5	-	60	111	297	69	0	175	684	975	1148	-	1.32	50.7
20	9.0	0.5	8.5	-	99	129	305	69	0	424	1758	3117	3709	-	1.16	41.2
21	3.5	1.0	2.5	-	30	49	140	66.5	17	213	-	347	347	-	1.22	50.7
22	5.5	4.5	1.0	-	13	49	184	66.5	52	158	-	269	-	-	1.26	41.2
22a	5.5	1.0	4.5	-	55	64	144	66.5	39	1301	-	1219	-	-	1.23	50.7
23	9.5	1.0	8.5	-	108	58	137	66.5	58	674	-	2705	-	-	1.27	44.9
24	3.0	1.0	2.0	-	28	267	856	72	36	319	-	401	-	-	1.41	35.6
25	5.0	1.0	4.0	-	56	250	713	72	68	942	-	1641	-	-	1.40	35.8
26	9.0	1.0	8.0	-	115	290	751	72	113	2021	-	4069	-	-	1.44	34.4
27	2.5	1.0	1.5	-	32.8	79	286	69	0	85	127	144	-	136	1.31	41.6
28	2.0	1.0	1.0	-	26.7	100	367	69	14	82	128	129	-	135	1.34	40.5
29	1.5	0.75	0.75	-	19.7	104	370	69	0	38	44	49	-	50	1.31	40.6
30	3.0	1.0	2.0	-	34.6	66	74	63	0	94	136	153	-	151	1.15	52.4
31	5.0	1.0	4.0	-	58.2	65	56	63	0	304	530	658	-	709	1.16	51.1
32	9.0	1.0	8.0		103.2	60	54	63	48	563	1186	1838	-	2691	1.15	51.5
33	5.0	-	3.0	2% (Quicklime)	57.5	-	708	70	20	23	35	49	-	-	1.36	39.9

Binder Optimization

Strength testwork carried out as part of this investigation was undertaken using various combinations of Slag, GP cement and Hydrated Lime. To provide a comparison between the binder constituents, the strength achieved by paste (batched to ≈69% solids) with slag, GP Cement (GP), Hydrated Lime and Quicklime are plotted against hydration time in Figure 13.13.13.

Figure 13.13.13 Strength Versus Hydration for Paste Mixes with Various Binders

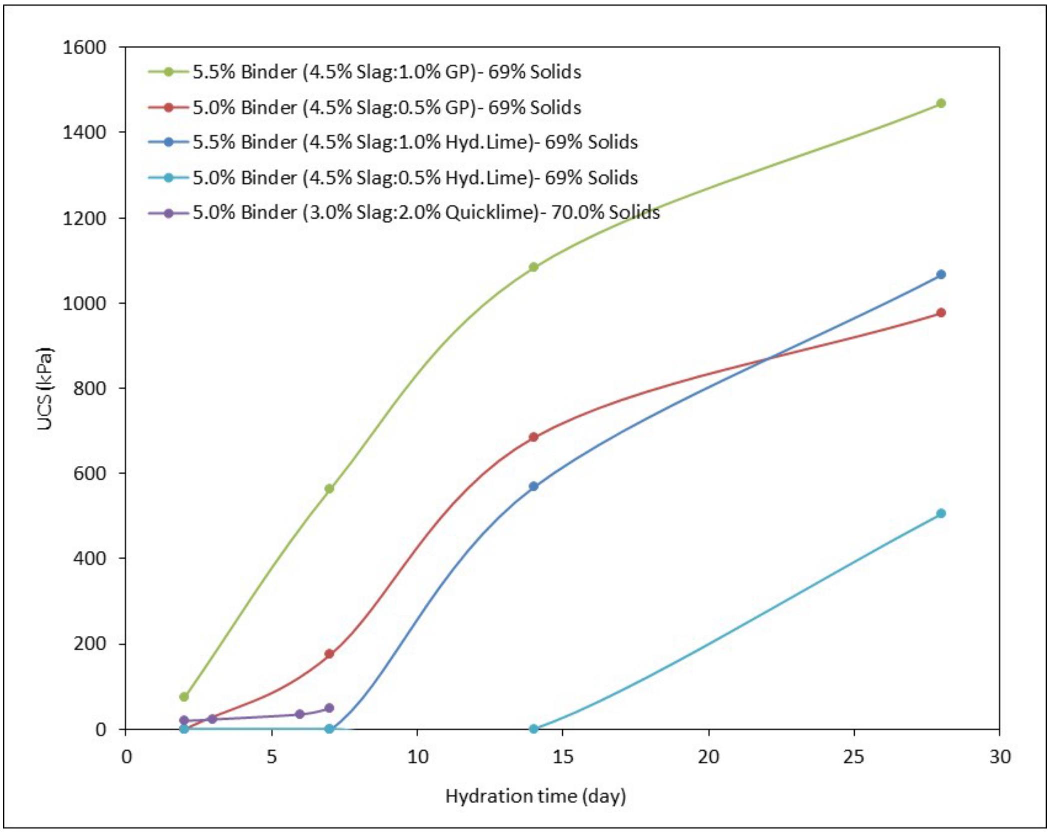
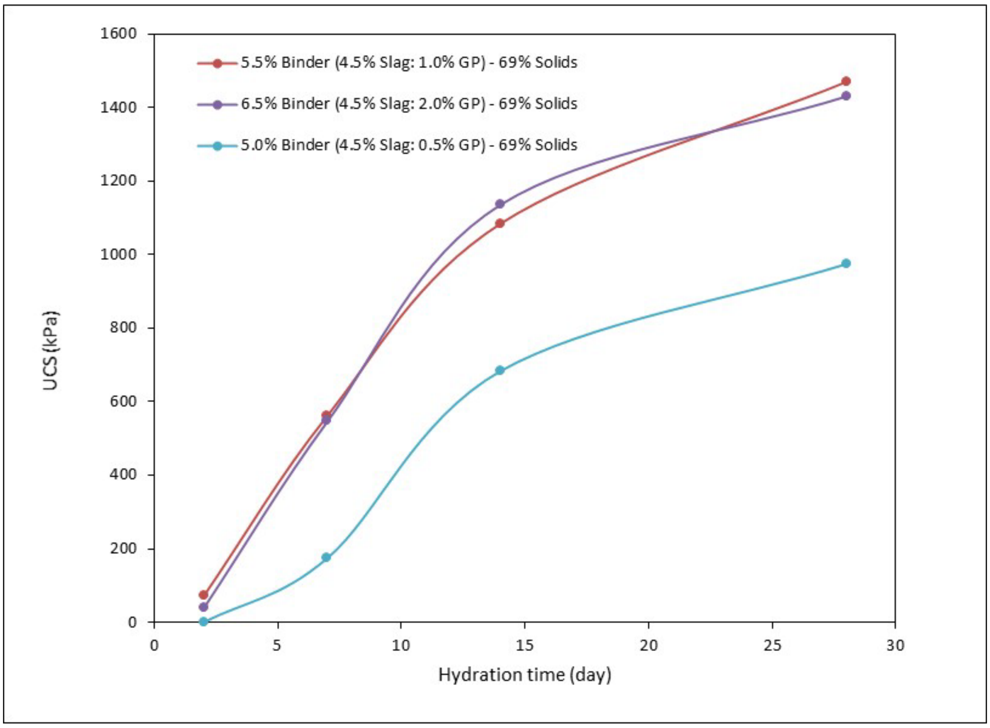


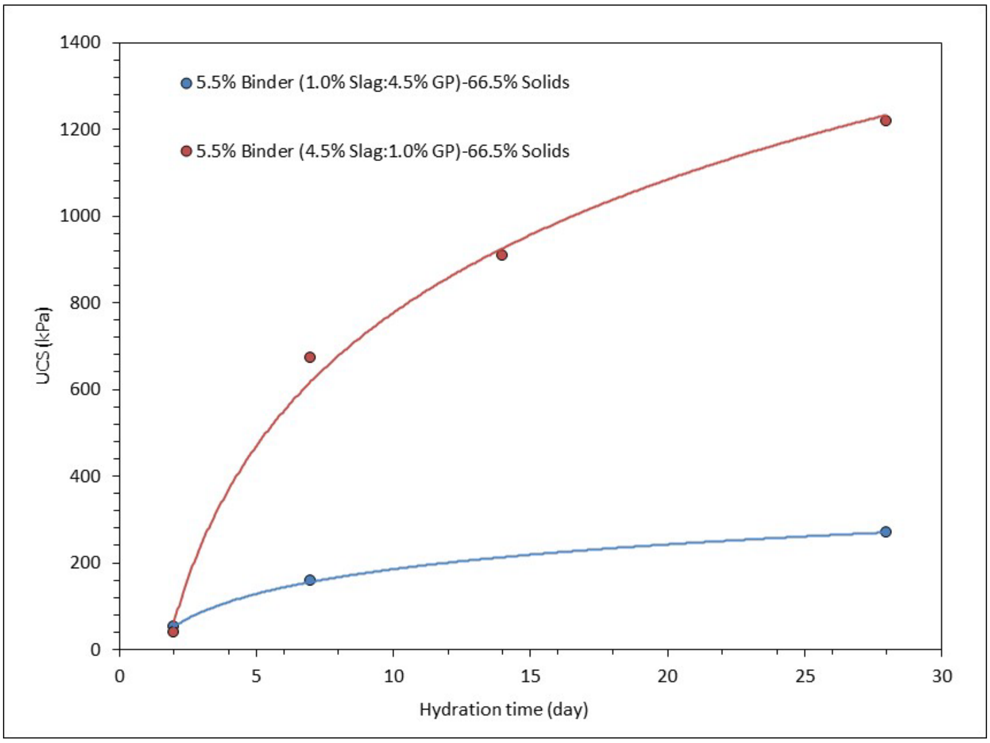
Figure 13.13.13 shows that, with equal binder additions, the 28 day strength achieved by the Slag / GP combination is significantly higher than that with both Slag / Hydrated Lime and Slag / Quicklime combinations. One particular aspect that is considered critical for the Kora mining plan is the early age strength performance. Figure 13.13.13 show that the GP blends develop some tangible strengths at 7 days, while the lime blends develop no tangible strength at this point. Table 13.13.5 shows that this trend is consistent across all mixes containing Lime rather than GP cement (with the slag).

To assess the optimal GP cement addition testing was completed with a various combinations of Slag and GP. Testing with a fixed Slag content of 4.5% binder and a paste solids content of 69% are presented in Figure 13.13.14a, while mixes containing different ratios of Slag and GP, batched to 66.5% solids, are compared in Figure 13.13.14b.

Figure 13.13.14 Strengths Versus Hydration Time for Different Slag / GP Combinations



(a)



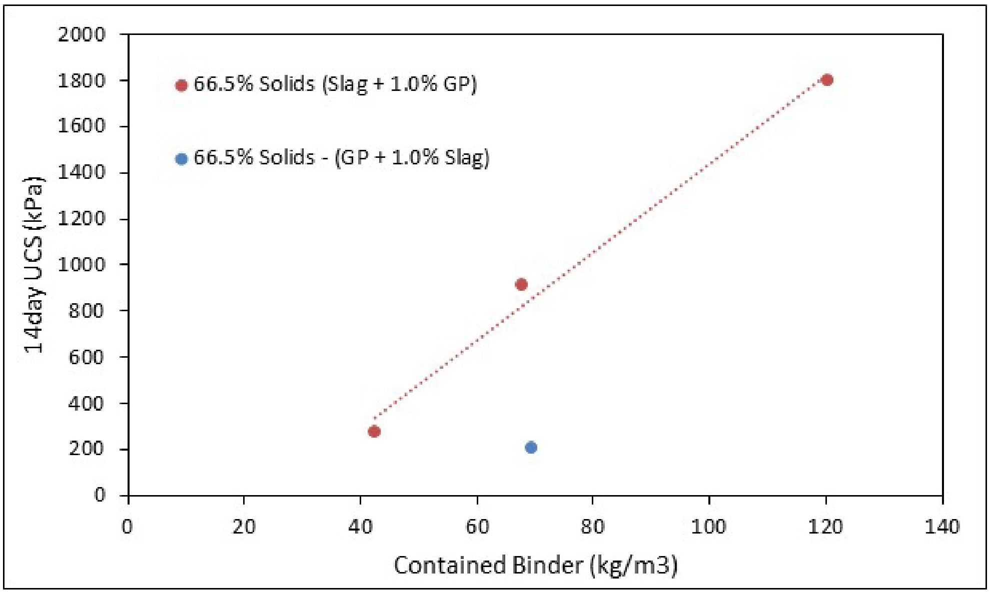
(b)

The results presented in Figure 13.13.14b illustrate that the mix containing the higher portion of Slag generates significantly higher strengths compared with the mix containing much higher GP content. Figure 13.13.14a shows that, in order to maximize the strength, it's necessary to add at least 1.0% GP, but GP addition above this portion appears to have no further benefit to the achieved strength.

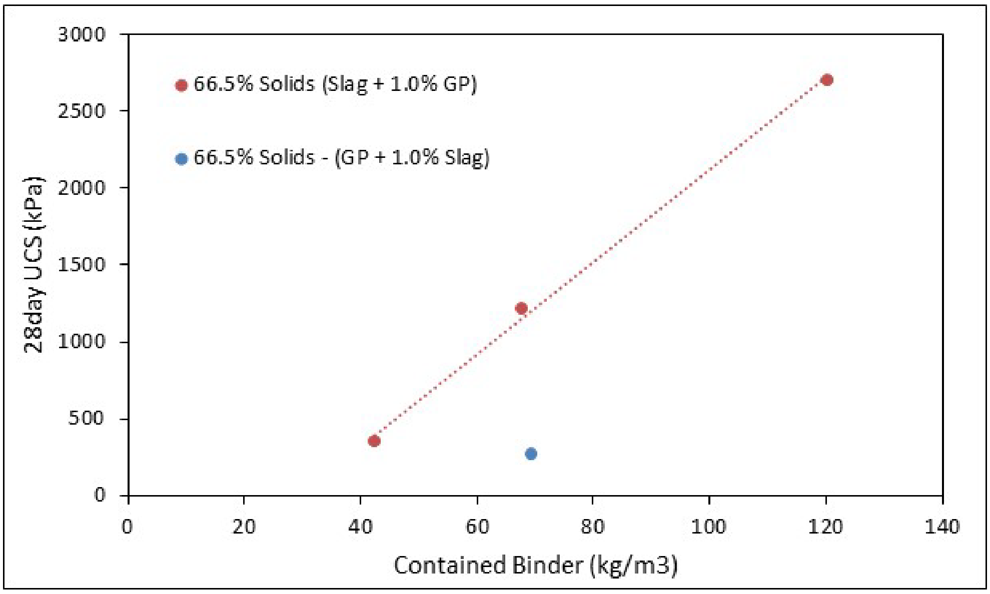
While further binder optimization should be considered in future studies, for the purpose of this investigation the remainder of testwork was completed with a constant GP addition of 1.0% and variable Slag content.

Given the high cost of Slag (at $316 USD/t), relative to GP cement (at $170 USD/t) consideration was given to the scenario where Slag is replaced by GP Cement. At the time of completing the testwork it was understood that the unit cost of Slag and GP cement were similar and as such no testwork was completed using 100% GP. However, Mix 22 (ref. Table 13.13.5) was completed with 4.5% GP cement and 1.0% Slag, to provide an indication of increasing GP cement contents. This mix is compared with equivalent paste mixes containing various portion of Slag and 1% GP cement (Mix 21, 22a and 23) in Figure 13.13.15. Figure 13.13.15 a and b present the 14 and 28 day strength versus binder contained binder for the different mixes, respectively.

Figure 13.13.15 (a) 14 and (b) 28 Day Strength for Paste Batched with Different Binder Types



(a)



(b)

For the weighted average target strength of 280 kPa (after both 14 and 28 days) the required 'Slag + 1.0% GP' binder blend content requires 40 kg/m^3 to achieve the target, while the 'GP + 1.0% Slag' binder blend (assumed to be similar to the 100% GP requirement) would require approximately 72 kg/m^3. Given the respective unit costs this equates to a weighted average Binder Component of the Operating Cost of:

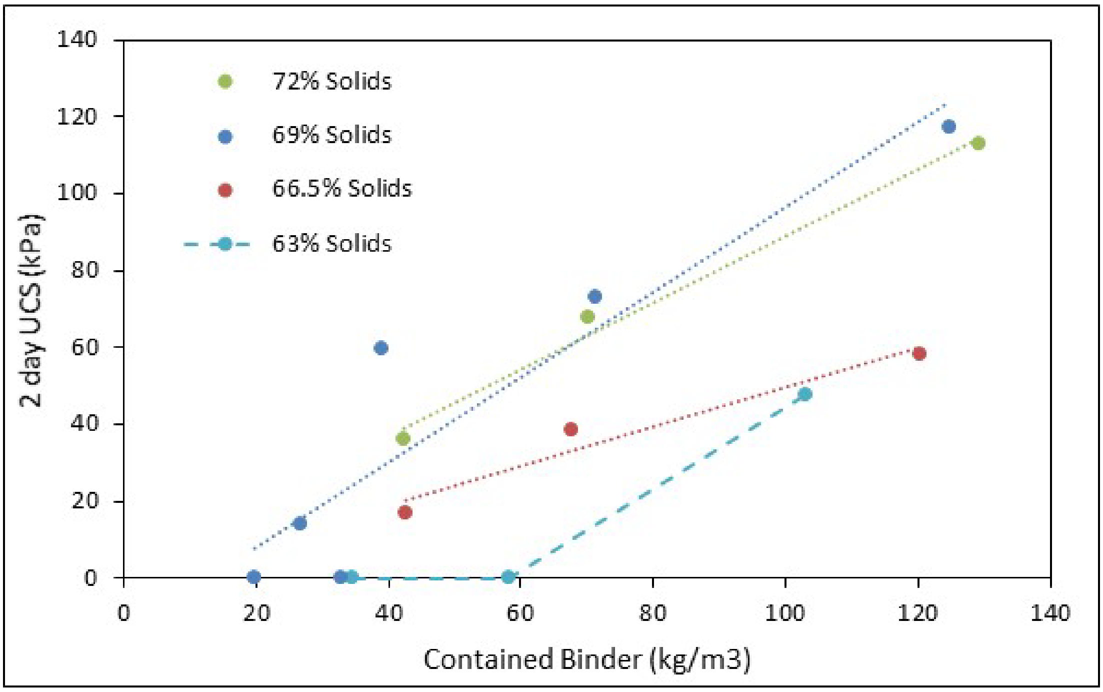
- $12.6 USD/m^3 for the 'Slag + 1.0% GP' binder blend (Binder component of OPEX).
- $11.9 USD/m^3 for the 'GP + 1.0% Slag' binder blend (Binder component of OPEX).

Costs presented above appear similar, however, as the preferred process requires the addition of GP cement at the 1205 underground mixing station, adopting 100% GP cement would (on average) require transport of approximately 150t of GP cement underground each day, compared with 26 t/day with the 'Slag + 1.0% GP' binder blend. This additional haulage requirement is expected to add significant additional costs and complexity to operation of any system that uses 100% GP cement.

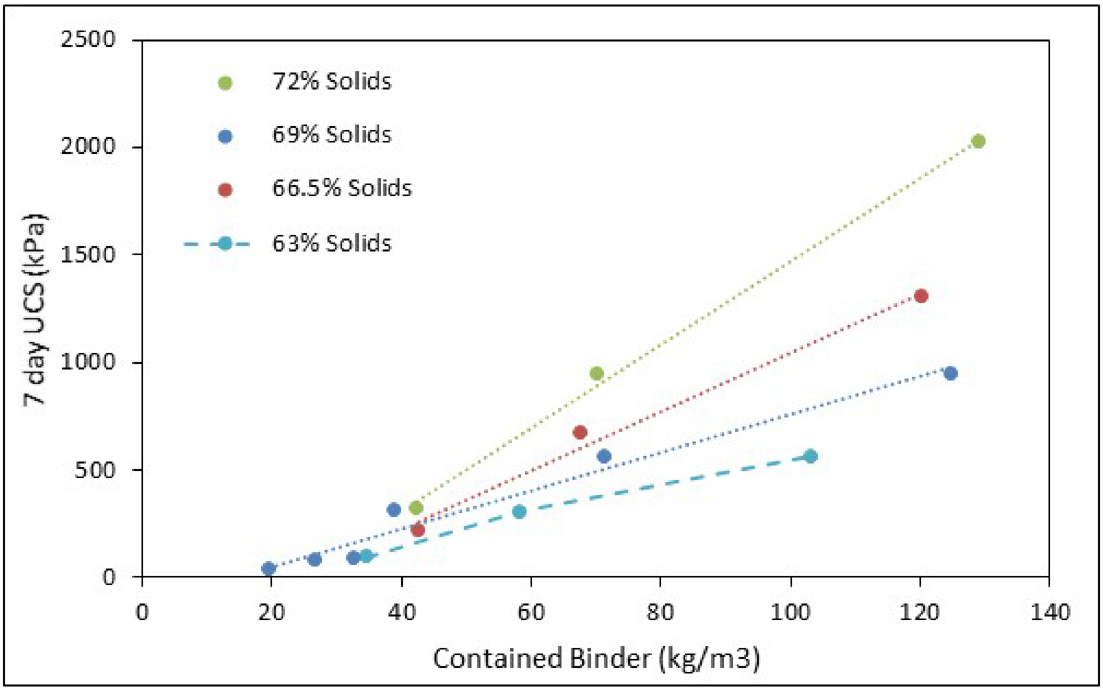
Influence of Paste Solids Content

To investigate the influence of paste solids content on paste binder requirements at Kora a series of tests were completed on paste batched to 63.0%, 66.5%, 69.0% and 72.0% solids (w/w). Paste mixes were batched to these solids concentrations and duplicate samples were tested for strength after hydration periods of 2,7 and 28 days, with some specimens tested after 14, 56 and 112 days. The measured strengths are plotted against contained binder in Figure 13.13.16. It should be noted that some mixes were not tested after 14 days and consequently strengths had to be estimated based on interpolation between 7 and 28 day measurements. These specimens are shown as open symbols. The contained binder is the binder mass contained in each in situ unit volume of paste.

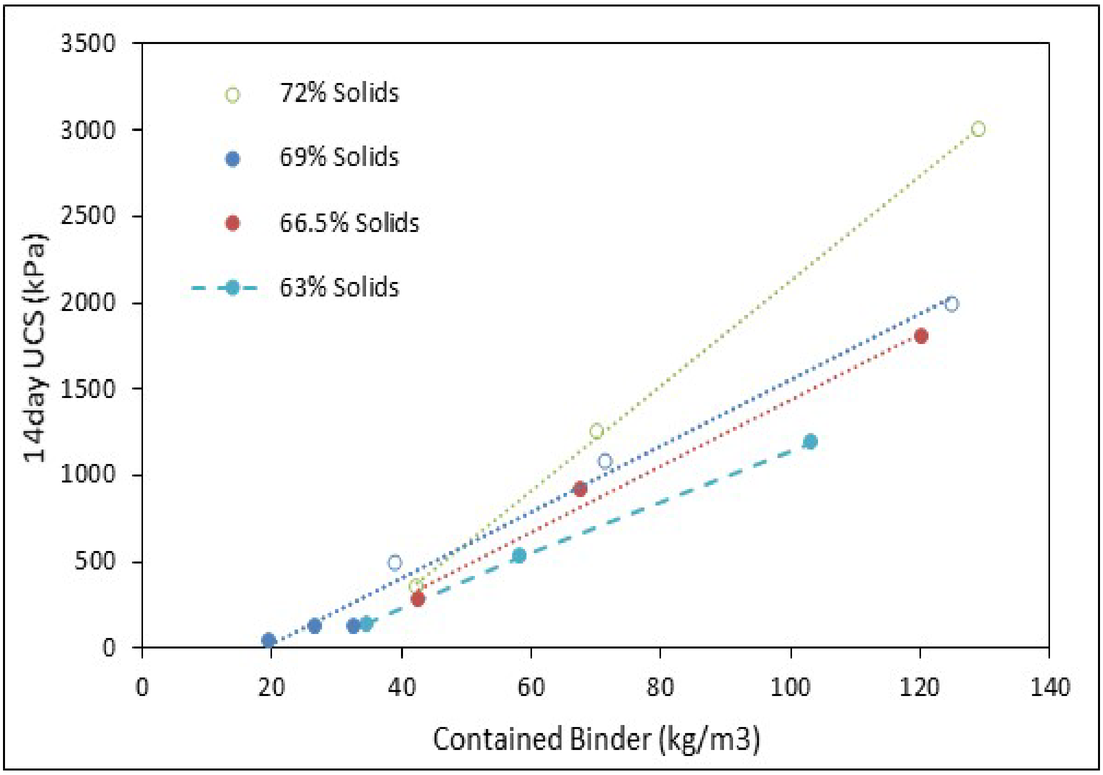
Figure 13.13.16 (a) 2, (b) 7, (c) 14 and (d) 28 Day UCS Versus Contained Binder



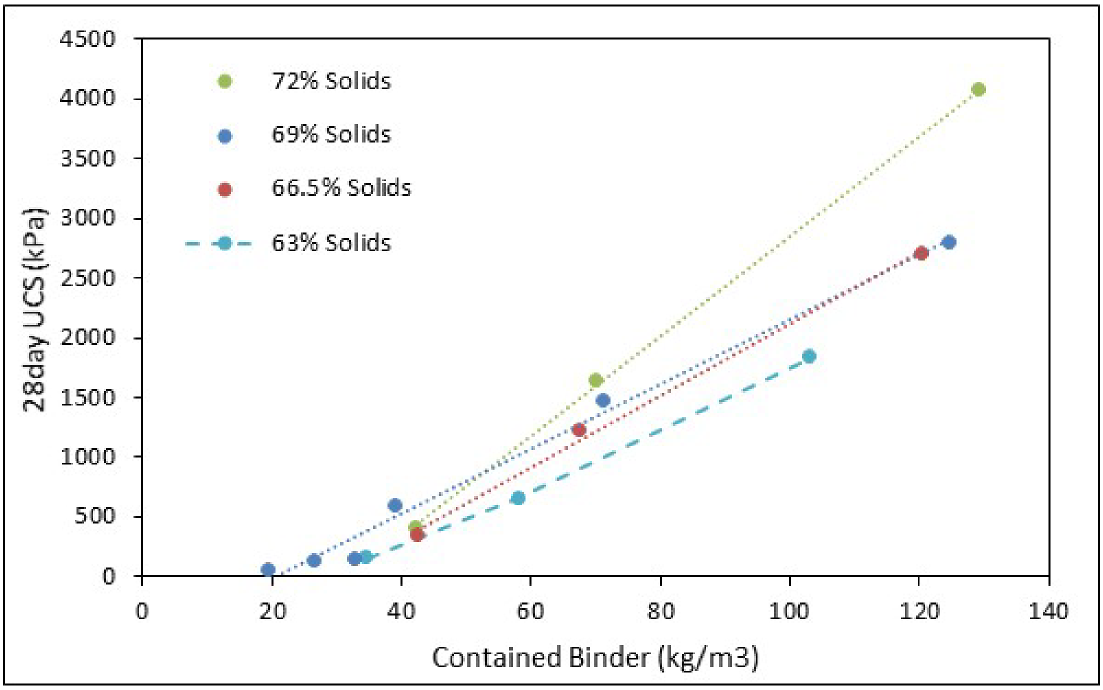
(a)



(b)


3500
3000
2500
2000
1500
1000
500
0
14day UCS (kPa)
72% Solids
69% Solids
66.5% Solids
63% Solids
0
20
40
60
80
100
120
140
Contained Binder (kg/m3)

(c)


4500
4000
3500
3000
2500
2000
1500
1000
500
0
28day UCS (kPa)
72% Solids
69% Solids
66.5% Solids
63% Solids
0
20
40
60
80
100
120
140
Contained Binder (kg/m3)

(d)

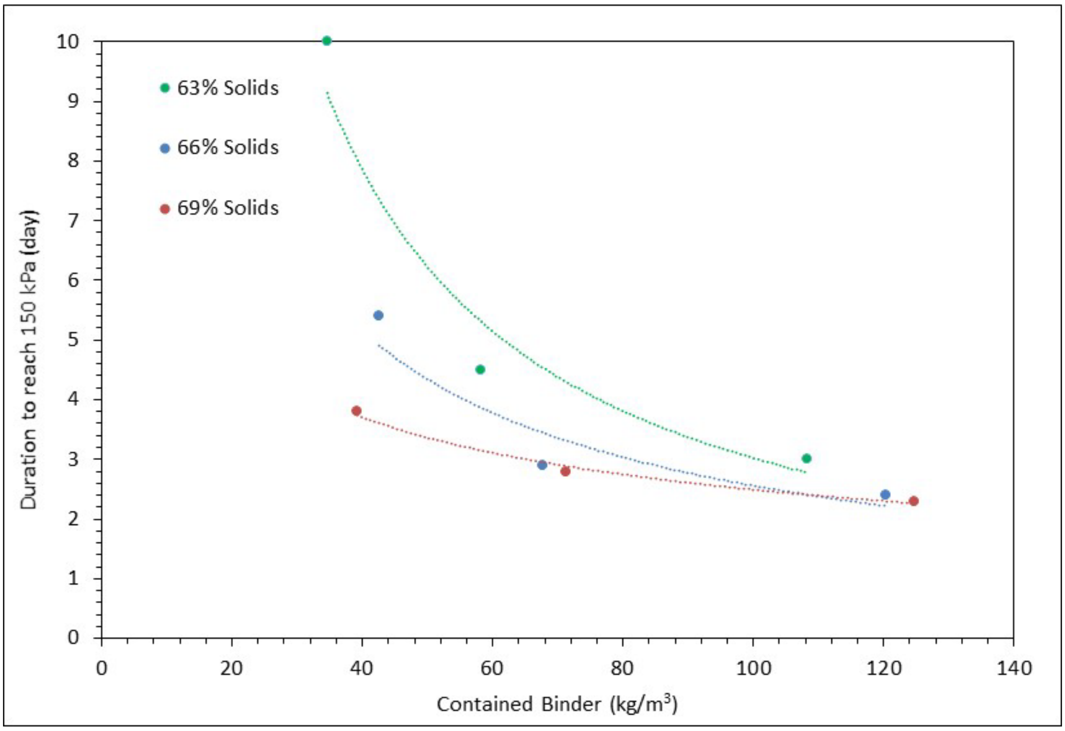
Figure 13.13.16 shows that:

- Paste batched to 69-72% solids appear to have a similar relationship between binder and strength.
- At typical strength targets paste batched to 66% solids is expected to require approximately 10% more binder compared with that batched at 69-72%.
- At typical strength targets paste batched to 63% solids is expected to require approximately 25% more binder compared with that batched at 69-72%.

In addition, Figure 13.13.16a shows that the rate of early age strength development for paste batched at 66-63% solids is very low. Figure 13.13.16a shows that at 72% solids a 2-day strength of 50 kPa can be achieved with approximately 50% less binder (60 kg/m^3 less) compared with paste placed at 66.5% and Figure 13.13.16b shows that at 7 days hydration 500 kPa can be achieved using approximately 50% less binder (40 kg/m^3 less).

To put this in context, its normal that re-entry to exclusion zones and re-commencement of mining activities in and around stopes would not occur until the paste has reached a nominal strength of around 150 kPa. Interpolating between the data presented in presented in Figure 13.13.16 the duration required to reach a strength of 150 kPa was determined and is plotted against contained binder for paste batched to 63%, 66% and 69% solids in Figure 13.13.17.

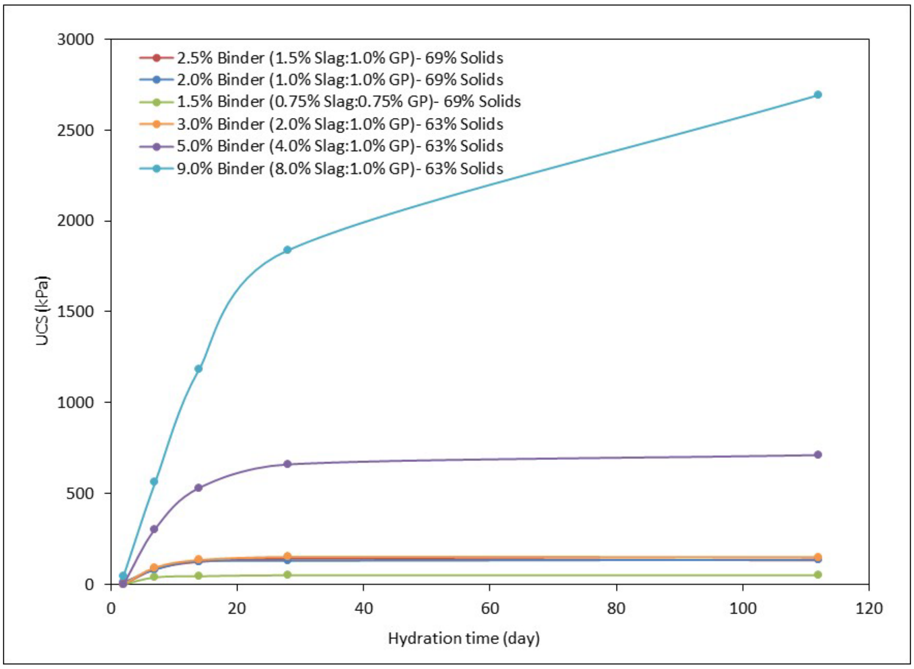
Figure 13.13.17 Duration to Reach 150 kPa Versus Contained Binder



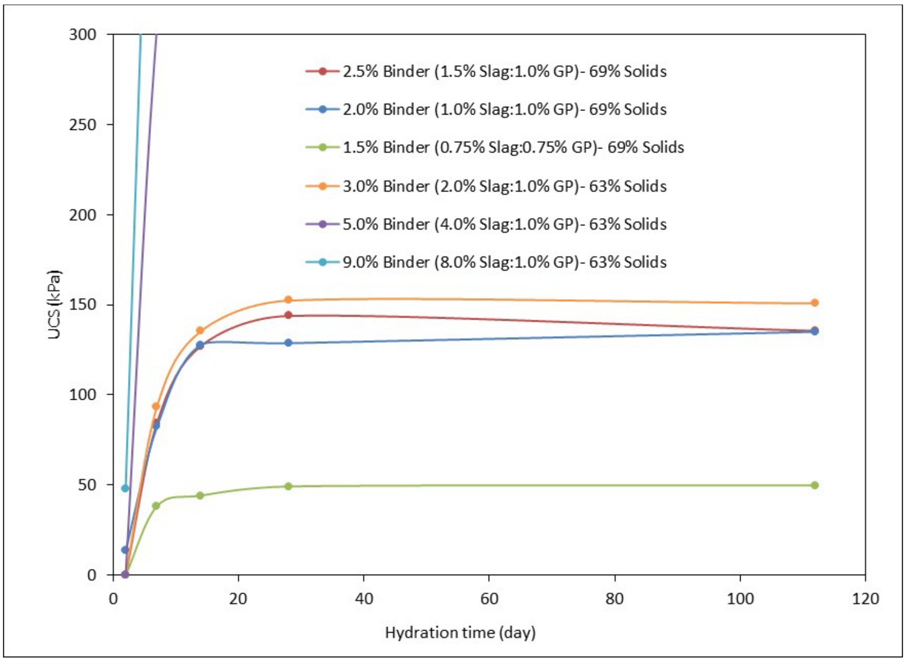
Strength Sustainability

Long term strength testing was completed on Kora paste mixes batched to 63% and 69% solids, with binder contents ranging from 1.5-9.0% equivalent. The measured strength for each mix are plotted against hydration time on two different vertical scales in Figure 13.13.18a and b.

Figure 13.13.18 Strength Versus Hydration for Kora Paste Mixes with Various Binders



(a)



(b)

Figure 13.13.13 shows that, at higher binder contents the continued increase in strength indicates that binder hydration is continuing. However, at lower binder contents the strength appears to plateau. Importantly, over the full range of binders and mix solids contents considered, Kora paste is shown to maintained sustainable strengths in the long term. This is a favorable outcome given the relatively high pyrite content (10%) in the Kainantu (Kora) tailings.

Based on the work by Clough (1989), the industry standard for a strength target that eliminates that likelihood of liquefaction of paste fill is 100 kPa. Based on the results presented in Figure 13.13.13, this can be achieved and sustained with paste containing 2% binder. This binder addition is expected to be targeted for paste deposited into remnant voids and the cost implication of this is discussed further in the Operating Cost Model, Section 21.6.

13.14 Process Selection and Design Paste Plant

13.14.1 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:

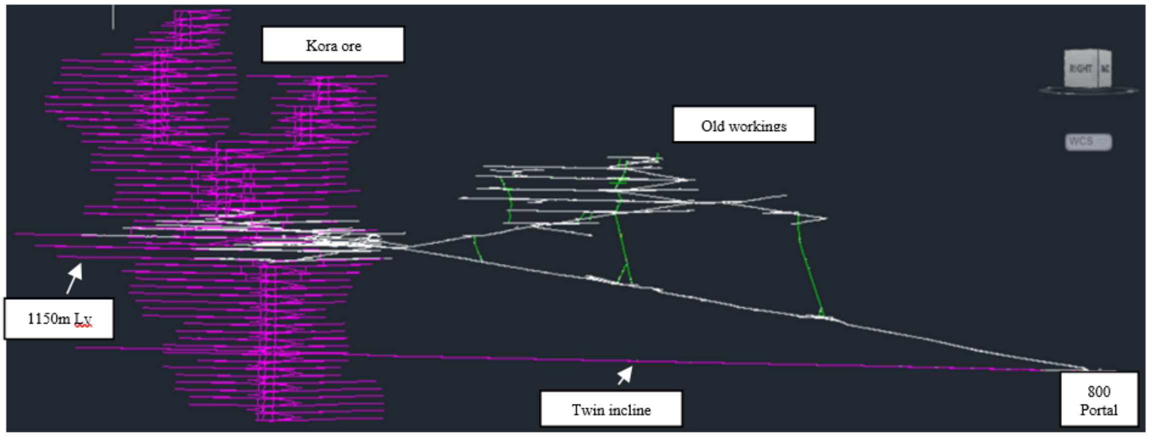
- Producing fill with the required mechanical properties.
- Delivered at rates that do not constrain mining activities.
- Have suitable infrastructure to permit a utilization capable of satisfying the mining schedule.

This section describes the specific backfill requirements in relation to the proposed K92 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 and it is understood that, from this point, a set of dual inclines are also being 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 13.14.1. The location of the Kora orebody is shown by the cluster of development on the LHS.

Figure 13.14.1 Long Section Showing Proposed Kora Underground Mine



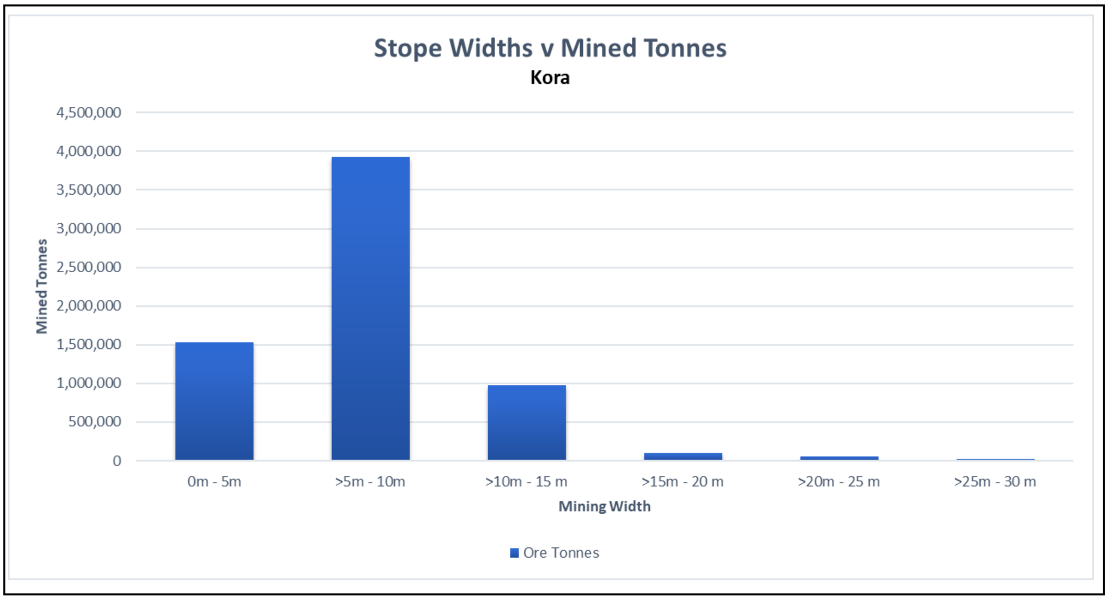
The Kora orebody is understood to exist in a series of closely spaced narrow veins and it is proposed to mine these veins independently with relatively narrow stopes, leaving a slender pillar between ore veins. The proposed stope extraction sequence is to involve both a 'bottom-up' and 'top-down' extraction 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 13.14.2. This figure shows that 83% of Kora stopes (and therefore fill exposed) are expected to have an across strike span less than 10 m, 15% of stope range in width from 10 to 15 m, while only 3% of stope spans are greater than 15 m. The floor-to-floor sub-level interval throughout the mine is 20 m.

Figure 13.14.2 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 13.13 indicates that this minimal strength requirement can be achieved using a combination of 1% GP cement and 1% Slag.

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 combination of analytical and numerical analysis was undertaken. This section provides a summary of the work that was completed to define the Kora paste strength requirements.

Geometry

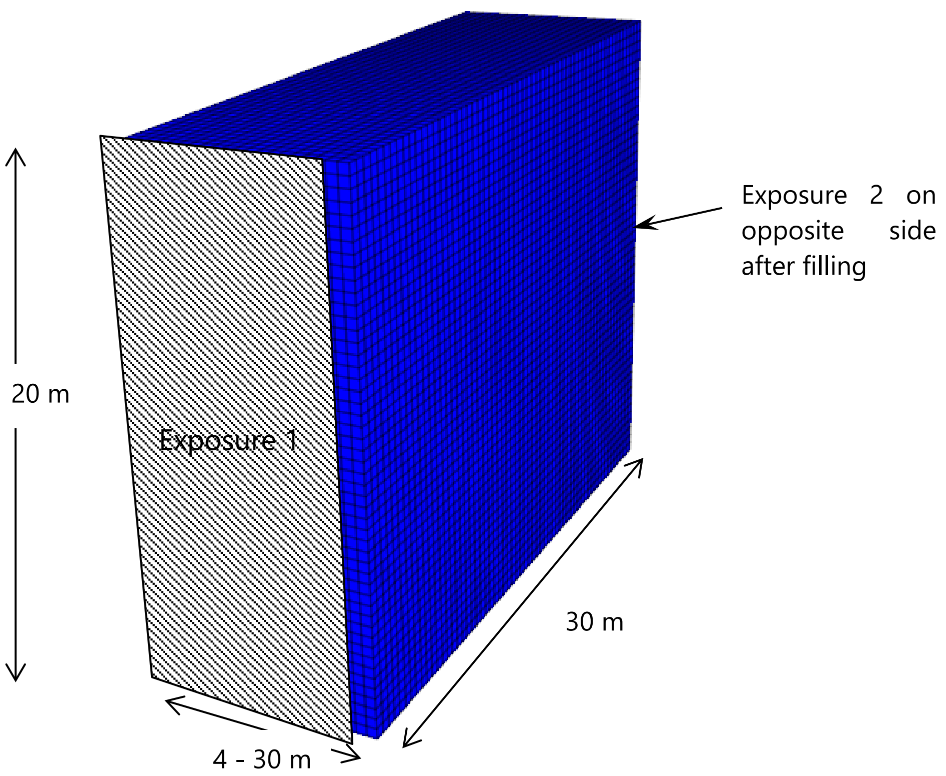
Mining at Kora involves extraction of ore immediately adjacent to, above and beneath previously deposited fill masses. During this process, fill is exposed both vertically and horizontally. This analysis considers the strength requirements for both of these exposure events.

Vertical Exposure Analysis

Stoping at Kora is understood to be undertaken using primarily a continuous retreat sequence. Based on discussions with K92, stopes are understood to be sub-vertical and to represent this the fill exposures were assumed to occur in vertically oriented stopes.

As part of this analysis the range of vertical exposures were represented by the model geometry shown in Figure 13.14.3. The region indicated by the white hatching shows the face to be exposed vertically during mining of the first adjacent stope. While it is understood that the along strike stope length is 20 m, modelling was completed using a 30 m fill mass length to account for the boundary condition at the fill-fill boundary (i.e. in the case of multiple stopes along strike).

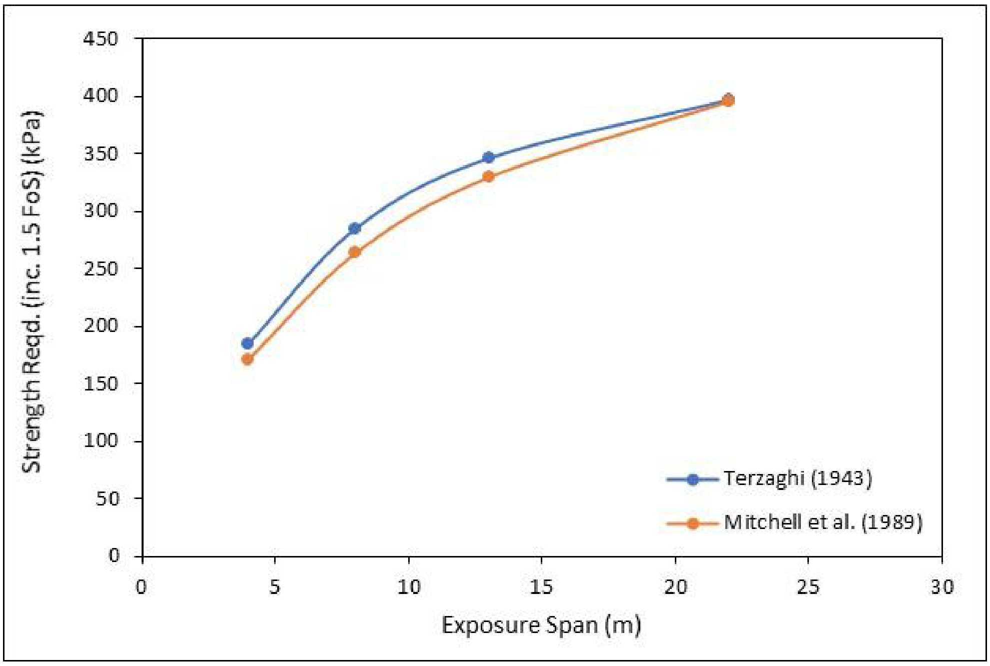
Figure 13.14.3 Isometric View Showing Geometry Adopted for Kore Fill Stopes



To determine the paste strength requirements for vertical exposures a combination of analytical methods were applied. The analytical solutions adopted are those by Mitchell et al. (1981) and the method of horizontal slices by Terzaghi (1943), and later applied to mine backfill in Winch (1999). These methods have been successfully applied many times previously for determining mine backfill vertical exposure strength requirements.

Applying these two methods the calculated strength requirements for 20 m tall vertical exposures is plotted against exposure width in Figure 13.14.4. The presented strengths include a FoS of 1.5 and should therefore be taken as the design target strength.

Figure 13.14.4 Paste Strengths Versus Exposure Span for 20 m Tall Vertical Exposures



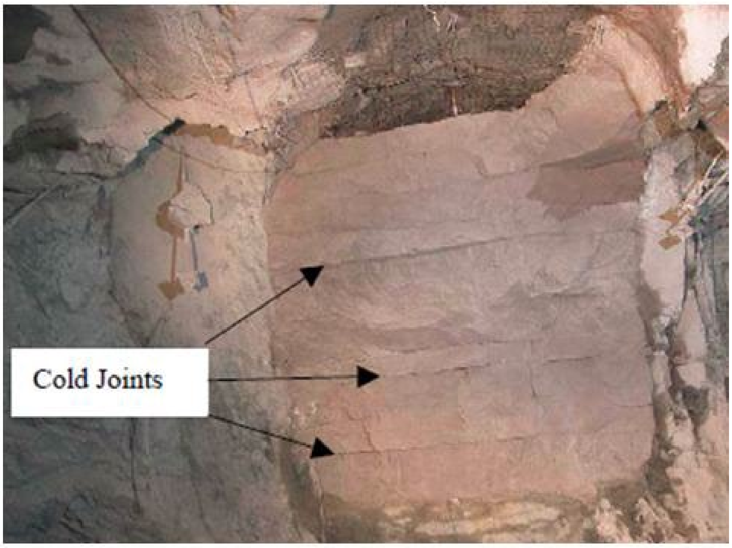
Horizontal Exposure Geometry

While it is understood that horizontal fill exposure spans may range from 2 to 30 m, the majority of horizontal exposures would be on narrow stopes. For the purpose of developing horizontal exposure strength relationships four analysis geometries were considered. These geometries include:

- 4 m exposure of a 20 m long and 20 m tall fill mass
- 8 m exposure of a 20 m long and 20 m tall fill mass
- 13 m exposure of a 20 m long and 20 m tall fill mass
- 22 m exposure of a 20 m long and 20 m tall fill mass.

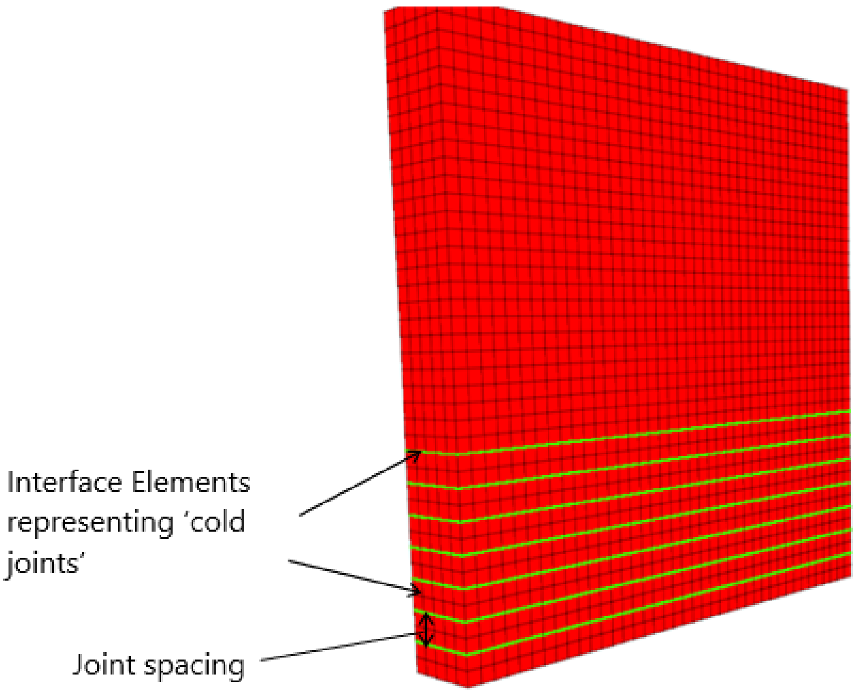
To determine the horizontal paste exposure strength requirements numerical analysis was undertaken. While analytical solutions exist for horizontal exposure of paste masses, due to the interbedding of 'cold joints' observed in all paste masses in situ, the analytical solutions (which assume a homogenous fill mass) fail to capture the failure mechanism. An example of this 'cold jointing' is presented in Figure 13.14.5.

Figure 13.14.5 Photograph Showing 'Cold Joints' in a Typical Paste Fill Mass



To capture this mechanism, our approach is to undertake exposure analysis using a 'dis-continuum' numerical model, where horizontal friction only interface elements are inserted at regular intervals to represent cold jointing. An example of this model is presented in Figure 13.14.6.

Figure 13.14.6 Example of Numerical Mesh with Interface Elements to Represent Cold Joints



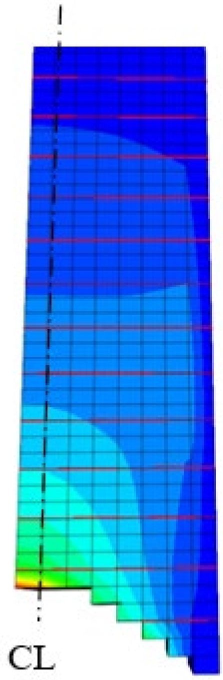
To simplify the analysis both the 4 m and 8 m wide geometries were represented using Plane-strain analysis. This was considered appropriate given the significant length relative to the across strike span. For the 13 m × 20 m and 22 m × 20 m horizontal exposure cases, these were analysed using a 'quarter-space'.

For all horizontal exposure analysis it was assumed that a high strength layer with thickness equal to the minimum horizontal span is included at the base of the fill mass. Above this layer the model was assigned strengths that correspond to paste a UCS of 160 kPa.

Based on past experience the frequency of cold joints tends to be approximately 0.5 m. This is illustrated in the photo presented in Figure 13.14.5, which shows approximately 10 cold joints across a 5 m drive. A 0.5 m cold joint spacing was adopted in the analysis.

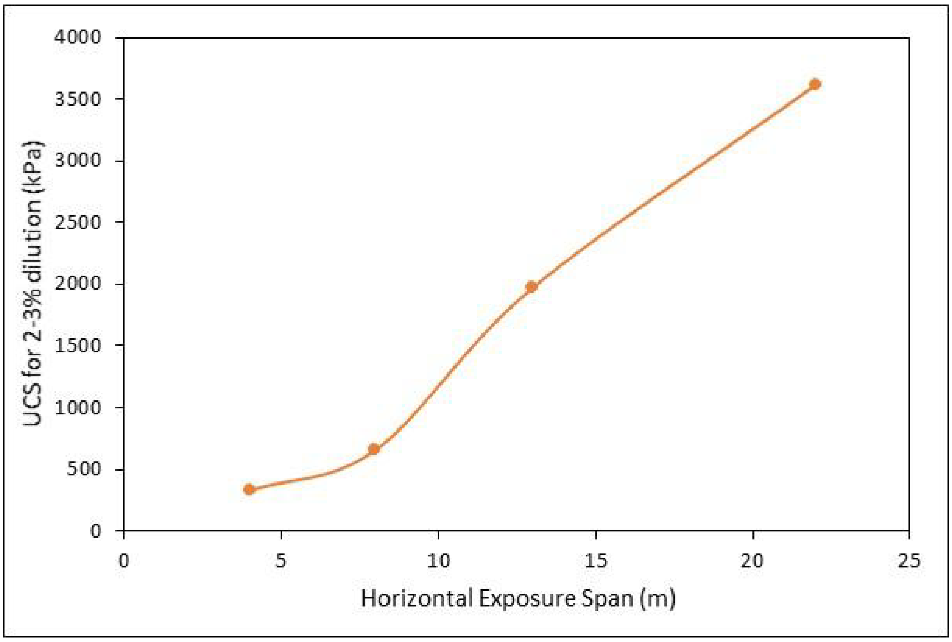
Based on discussions with (mine design engineers) Entech it is understood that a provision for 2% dilution has been allowed. To account for this in the analysis an arched profile that corresponds to an allowed dilution mass of 2-3% was established for each geometry. Figure 13.14.7 shows a 'half-space' model used to represent this geometry.

Figure 13.14.7 Horizontal Exposure Arched Shape Used in Horizontal Exposure



Using this model geometry, analysis was carried out to determine the paste strength requirements necessary to restriction dilution levels to the assumed quantity. The calculated strength requirements necessary to limit the paste dilution to a level of 2-3% is plotted against span in Figure 13.14.8. This figure incorporates a FoS of 1.3, which is considered appropriate given that the consequence of lower strength is only a marginal increase in dilution.

Figure 13.14.8 Estimated Kora Paste Strength Versus Horizontal Exposure Span



Strength Summary

Given the results of fill exposure analysis described above the strength requirements for each of the fill exposure geometries is presented in Table 13.14.1. All exposure geometries assume a fill mass length (along strike) of 20 m and a vertical height of 20 m.

Table 13.14.1 Kora Paste Strength Requirements

Fill Type	Exposure width m	Portion of total fill place %	Hydration Period days	Design Strength kPa
Sill for Horizontal Exposure	0-5	1.0	28	275
Sill for Horizontal Exposure	5-10	7.1	28	550
Sill for Horizontal Exposure	10-15	1.9	28	1097
Sill for Horizontal Exposure	15-20	0.3	28	2800
Sill for Horizontal Exposure	20-25	-	28	-
Sill for Horizontal Exposure	25-30	-	28	-
Layer for bogging horizon	All	7.0	14	200
Vertical Exposure	0-5	20.6	14	185
Vertical Exposure	5-10	48.2	14	285
Vertical Exposure	10-15	11.7	14	345
Vertical Exposure	15-20	1.2	14	370
Vertical Exposure	20-25	0.8	14	400
Vertical Exposure	25-30	0.3	14	430

Binder Requirements

Proposed Sequence

To determine the expected paste binder requirements the binder versus strength relationships presented in Section 13.13 were combined with the strength targets from Table 13.14.1. The binder requirements for paste placed at 66.5% solids is presented in Table 13.14.2. Also presented in Table 13.14.2 is the portion of fill to be placed into stopes of the relevant dimension as well as the duration required for each mix to achieve a strength of 150 kPa. As detailed further in Section 'Bulkhead Exclusion Zone', 150 kPa is the nominal strength where mining activities can recommence in close proximity to the stope being filled. This strength is representative of typical strengths used for re-entry to the 'exclusion zone' around bulkheads and the point that paste bulkhead can be removed for the purpose of excavating the drawpoint fill (i.e. the first step in establishing the next stope along strike).

Table 13.14.2 Estimated Binder Requirements at Kora Paste

Fill Type	Exp width m	Portion of total fill place %	Target strength kPa	Paste @66.5% solids	
				Binder reqd. kg/m³	Duration until 150 kPa strength days
Sill for Horizontal Exp	0-5	1.0	275	39	5.3
Sill for Horizontal Exp	5-10	7.1	550	48	4.5
Sill for Horizontal Exp	10-15	1.9	1097	66	3.5
Sill for Horizontal Exp	15-20	0.3	2800	123	2.2
Sill for Horizontal Exp	20-25	-	-	-	-
Sill for Horizontal Exp	25-30	-	-	-	-
Bogging horizon	All	7.0	200	35	5.7
Vertical Exposure	0-5	20.6	185	35	5.7
Vertical Exposure	5-10	48.2	285	40	5.2
Vertical Exposure	10-15	11.7	345	43	4.9
Vertical Exposure	15-20	1.2	370	44	4.8
Vertical Exposure	20-25	0.8	400	46	4.6
Vertical Exposure	25-30	0.3	430	47	4.5
Weighted average binder/re-entry time (kg/m³. day)				40	5.2

The results presented in Table 13.14.2 indicate that the binder requirements for Kora are quite low (at approximately 3.5%), which is expected to be a result of the small exposures and the provision for a conservative cure period (of 14 days), which is relatively long for the proposed stope sizes.

Another notable thing about the results presented in Table 13.14.2 is the relatively long duration (5.2 days) required for the paste to achieve a strength suitable for removing exclusion zones and commencing 'bulkhead dig-outs'. This must be accounted for in the mining schedule. Alternately this duration could be reduced through increasing the binder content, which would adversely impact the presented paste operating costs.

Sequence Sensitivity

To develop an understanding of the sensitivity of the Kora backfill system binder requirements to the stoping sequence the analysis presented in Section 'Binder Requirements' was repeated, assuming that all stopes are to be mined underhand. Under this scenario all stopes would include a high strength sill, but no provision would be included to account for a bogging platform. Under these circumstances the average binder requirement is expected to increase from 40 to 46 kg/m^3. Based on operating costs presented in Section 21.7, this change in binder requirement would be expected to increase paste operating costs by $2 USD/m^3 (of fill).

Mass Balance and Fill Production Rate

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 13.14.9. This figure shows the monthly fill requirements plotted against sequential mining month.

Figure 13.14.9 Kora Fill Schedule

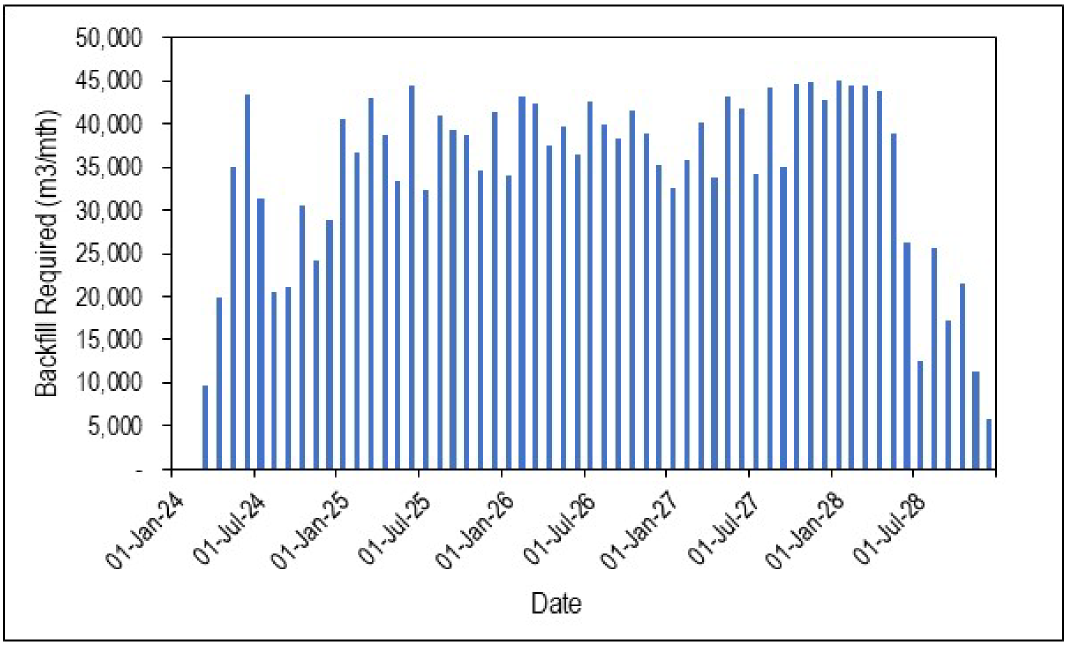


Figure 13.14.9 shows that a life of mine fill quantity of approximately 2.0M m^3 is required with a maximum monthly fill production rate of 45,000 m^3 and maximum fill demand (over any 12-month period of 508,000 m^3) is necessary to ensure mine production is not constrained.

Material Availability

Fresh Tailings

The Kora mining and processing operations are focused on a total ore processing rate of 1,200,000 tpa. Based on information provided by K92, it is understood that the process plant would remove 5% of the total mass with the remaining 95% reporting to tailings. Assuming an in situ dry backfill density of 1.23 t/m^3, the monthly available mass and required backfill mass are compared in Figure 13.14.10. Also plotted on the RHS axis is the portion of available tailings that must be utilized for filling, on a 3 month rolling average basis.

Figure 13.14.10 Mass of Available Tailings and Tailings Mass Required for Filling Against Production Month

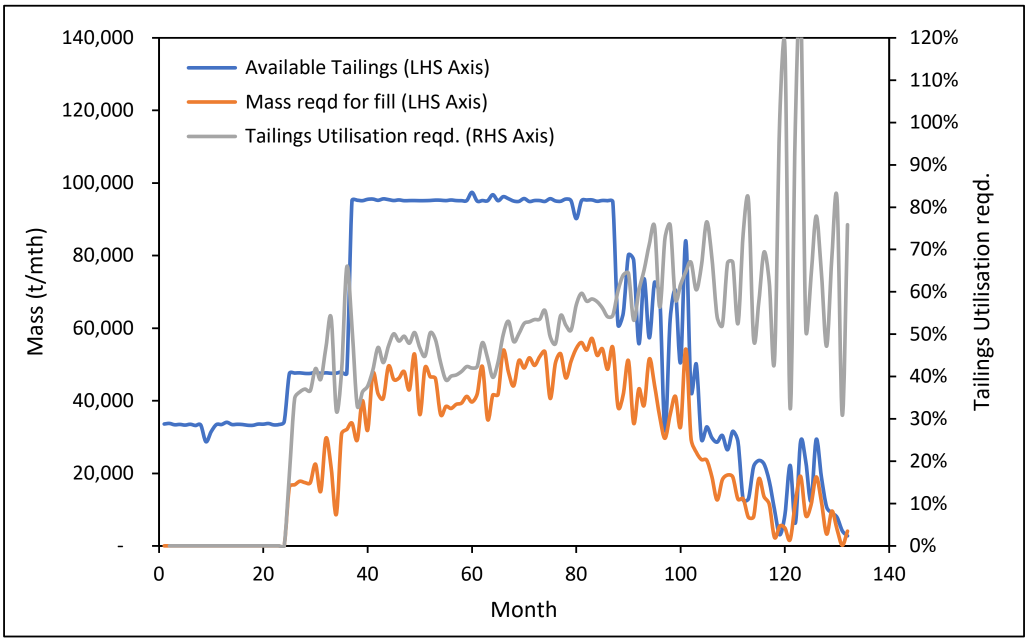


Figure 13.14.10 shows that, up to month 85, the required tailings utilization is approximately 50%, with this increasing to approximately 60% at 100 months. Beyond 100 months the ore (and tailings) production rate decreases and required tailings utilization increases to 70-80%. This utilization is not desirable. However, given that this is towards the later stages of the LOM and that this is largely driven by the drop-off in available tails we believe that is expected to be manageable using strategies such as campaign milling if necessary.

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 utilized 50% and 60% of the time. Typically fill systems would be sized to satisfy the mining requirements at around 65% utilization, however due to the increased complexity of the Kora system and the relatively small stope sizes it is recommended that a design utilization closer to 55% is more appropriate. The instantaneous fill demand for these utilization rates is plotted against mining month in Figure 13.14.11

Figure 13.14.11 Mass of Available Tailings and Tailings Mass Required for Filling Against Production Month

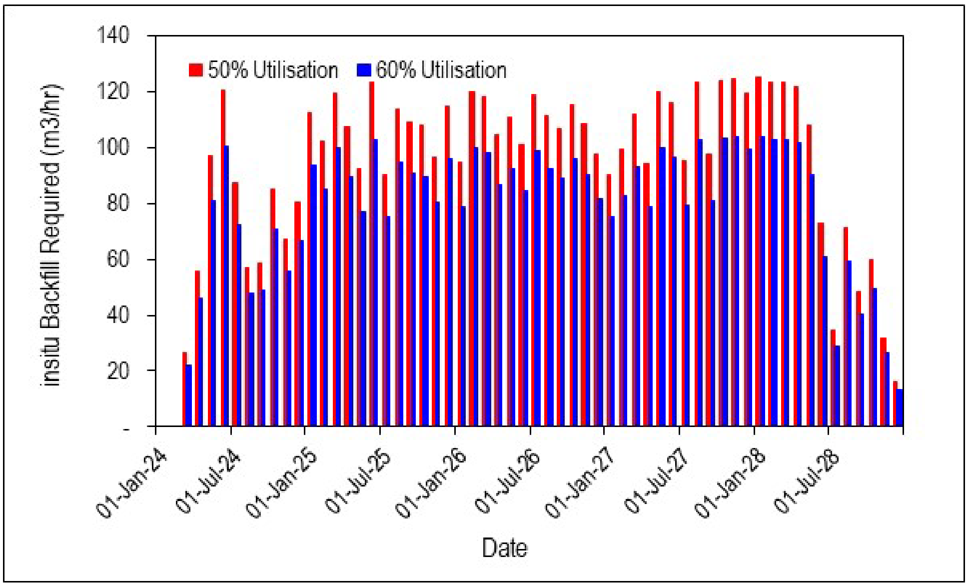


Figure 13.14.11 shows an instantaneous fill demand of just over 100 m^3/hr (at 60% utilization) and 120 m^3/hr (at 50% utilization). At the in situ dry density of 1.23 t/m^3 this equates to a fill tailings consumption of 123 and 148 t/hr for 60% and 50% utilization, respectively.

Assuming a process plant utilization of 92% the 'instantaneous' tailings production rate is expected to be approximately 141 tph. Given the close match between the fill requirements and the available tailings mass, it is recommended that Kora backfill system be designed to utilize the entire tailings stream (i.e. 141 t/hr of dry solids). Assuming a binder content of 3%, if all tailings are utilized voids would be expected to be filled at an 'instantaneous' rate of 118 m^3/hr, which would be expected to fill the maximum monthly fill demand, of 45,000 m^3/mth, with a system utilization of 53%. Given the continuous fill cycle described in Section 'Stope Fill Cycle' we believe that the system utilization required by the proposed mining cycle can be comfortably satisfied.

Kora paste is expected to be transported at an equivalent dry density of 1.15 t/m^3 before shrinking to an in situ dry density of 1.23 t/m^3, meaning that every cubic meter filled underground would require 1.07 m^3 of paste. On this basis, the 141 t/hr of available tailings would generate 122 m^3/hr of wet paste that must be accounted for in the process design.

Stope Fill Cycle

Bulkhead Design and Fill Sequence

Given the 20 m sub-level intervals proposed for Kora, the maximum pressure to be exerted onto a bulkhead from the paste (i.e. assuming no consolidation or cementation) would be given by the full geostatic stress. This pressure is given by:

$$\sigma_{bulkhead} = H_{stope} . \rho_{paste} . g = 20. 1.77 . 9.8 = 350\ kPa$$

Assuming a bulkhead uncertainty FoS of 1.5 (i.e. no bulkhead load FoS is required as the worst-case scenario is assumed) this results in a required ultimate bulkhead capacity of 520 kPa.

Fibrecrete with a 28 day characteristic strength of 32 MPa is expected to have a 24-hour characteristics strength of 15 MPa. With 15 MPa fibrecrete previous analysis, on 5-7 m bulkhead spans, has shown a 400 mm thick arched fibrecrete bulkhead to have an ultimate capacity of 550 kPa, which is considered appropriate for the proposed continuous filling strategy. On this basis it is expected that Kora stopes would be filled using a single continuous pour with:

- 400 mm thick arched fibrecrete bulkheads.
- Bulkheads should not be loaded until the fibrecrete strength has achieved a characteristic strength of 15 MPa, which is expected to require a 24-hour cure period.

For a typical 7.5 x 20 x 20 m tall stope, the expected fill schedule is outlined below:

- 12 hours – Bulkhead formwork installation.
- 12 hours – Bulkhead spraying.
- 24 hours – Bulkhead curing.
- 33 hours filling (3,000 m^3 @ 102 m^3/hr).

Given the schedule provided above, it's expected that a minimum individual stope filling time (prior to commencement of the cure period) of 71 hours would be necessary, which is well within the mining schedule requirements for Kora.

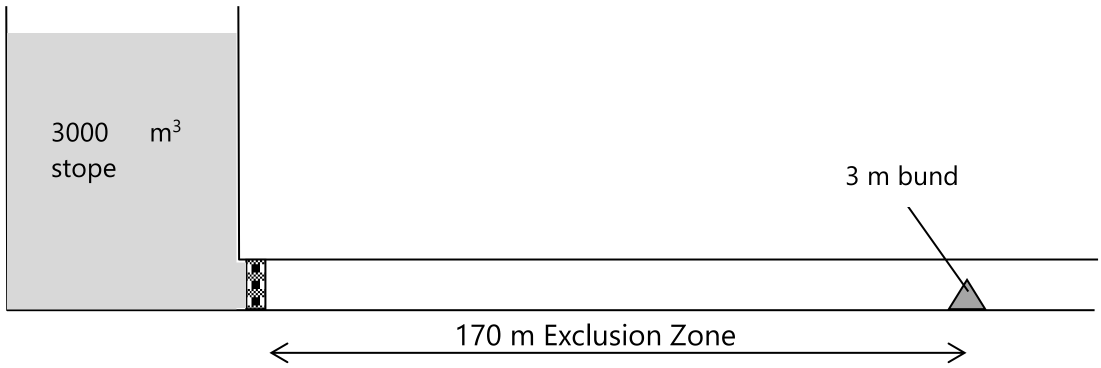
Bulkhead Exclusion Zone

Due to the catastrophic consequence of a bulkhead failure it is common to provide two forms of barrier between underground personnel and potentially liquifiable paste. To achieve this, it is typical to establish a bunded exclusion zone in front of every paste bulkhead, while containing liquid paste.

In the Kora case, given the low target paste strengths, and the relatively low rate of strength development, continuous filling of stopes would likely result in the entire stope being full of potentially liquified paste (i.e. paste that must be contained in the exclusion zone).

Assuming that a 3.0 m tall bund is constructed at the end of the containment bund and that the exclusion zone is in a 5 m wide drive, the total excluded drive length is expected to be 170 m for a 3,000 m^3 stope. This scenario is illustrated in Figure 13.14.12.

Figure 13.14.12 Typical Stope Exclusion Zone



As a minimum it is expected that the exclusion zone shall be maintained until the fill in the lower 8 m has achieved a paste strength of 150 kPa. Given the paste strength development test results presented in Section 13.13, the exclusion zone is expected to remain in place for a period of approximately 5-6 days. It is understood that the mining schedule undertaken as part of this study has made provision for the required exclusion zone.

13.14.2 Paste Fill – Process Description

Paste System Process Design Criteria

Given the background described in Section 13.14.1, the only backfill solution considered technically feasible for the Kora mine is cemented paste backfill manufactured from full stream Kainantu tailings. The key process design criteria for the Kora paste system is set out in Table 13.14.3.

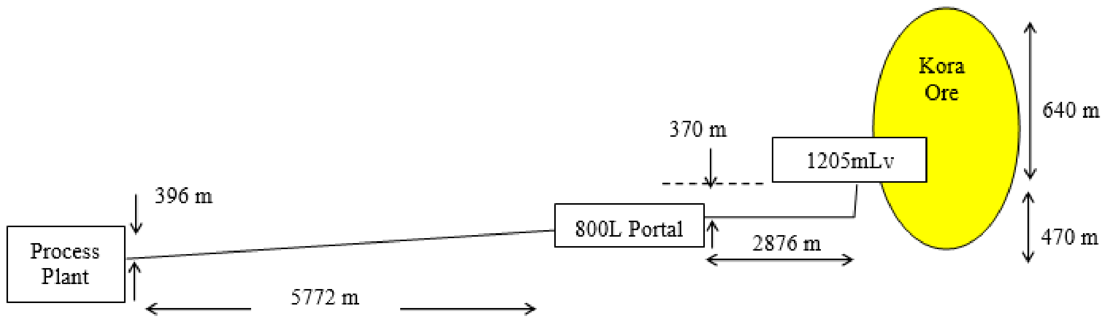
Table 13.14.3 Kora Paste System General Design Criteria

General		
Description	**Units**	**Quantity**
Site Conditions		
Altitude (Process Plant)	mAMSL	507
Altitude (800 Portal)	mAMSL	904
Climate Type		
Min Temperature	deg C	13
Max Temperature	deg C	25
Min Montly Average Rainfall	mm/mth	102
Min Montly Average Rainfall	mm/mth	317
Yearly Average Rainfall	mm/yr	2,410
Relative Humidity		
Average Relative Humidity	%	87%
Design Relative Humidity	%	90%
Tailings Characteristics		
Tailings SG	t/m^3	2.86
Target Grind P80	micron	110
Tailings % solids (by mass)	%	35
Tailings pH		7
Tailings Temperature	deg C	25
Raw Water Characteristics		
pH Range		6.5-8.0
Temperature Range	deg C	15-25
Specific Gravity	t/m^3	1.000
Total Suspended Solids	ppm	5
Chloride	mg/L	6
Sulphate	mg/L	1600
Binder Material Properties		
Binder Product #1		Granulated Ground Blaste Furnace Slag
Material SG	t/m^3	3.0
Bulk Density (nominal)	t/m^3	1.1
Binder Product #2		GP Cement Blend
Material SG	t/m^3	3.2
Bulk Density (nominal)	t/m^3	1.2
Backfill Plant Operating Duty		
Operating hours per day	h	24
Operating days per year	days	230
Availability	%	85%
Operating hours per year	h	4300
Paste Fill Plant Dry Tails Feed	dry t/a	606000
Paste Fill Plant Dry Tails Feed	dry t/hr	141
Paste Production		
Mass	tph	217
Flow	m^3/hr	123
Paste Density (excluding binder)	t/m^3	1.76
Tailings Feed		Final Tailings
Tailings Source		Final Tailings Feed Hopper
Sub Level interval	m	20
Length (along strike)	m	20
Stope width		
Average	m	7
Minimum	m	3
Maximum	m	30
Stope Fill Time		
Average	hrs	23
Minimum	hrs	10
Maximum	hrs	75
Stength Requirements		
Average	kPa	550
Minimum	kPa	275
Maximum	kPa	2800
Slag Requirements		
Weighted Average	t/hr	3.2
Minimum	t/hr	1.4
Maximum	t/hr	14.4
GP Requirements		
Weighted Average	t/hr	1.4
Minimum	t/hr	0.7
Maximum	t/hr	2.2

Process Overview

The Kainantu site layout is heavily constrained in terms of both terrain and real estate and to address this the proposed paste system includes major infrastructure located in three discrete locations. Paste is transferred between these stations hydraulically, through high pressure pipelines. These locations include the Process Plant, 800L Portal and 1205mLv (underground chamber). A schematic showing the relative location of these sites is presented in Figure 13.14.13.

Figure 13.14.13 Schematic Long Section Showing Kainantu Layout and Kora Paste System Profile



The process adopted for paste manufacturing is structured such that:

- All tailings dewatering occurs at the process plant to generate a relatively low, but tangible, yield stress uncemented paste. The major benefit of dewatering at the process plant is that and engineered slurry can be created and controlled to simplify the hydraulic transport duty throughout the remainder of the system. The design adopted is to manufacture paste with a yield stress of approximately 50 Pa, that can be transported in a laminar flow regime with low friction losses, which will not settle in overland pipelines at low, and no, velocity. Dewatering at this location was also favoured because of the availability of real estate, the proximity of dewatering infrastructure to the process area (for servicing) and because it eliminates the need to include a return filtrate water line from the stations further downstream.

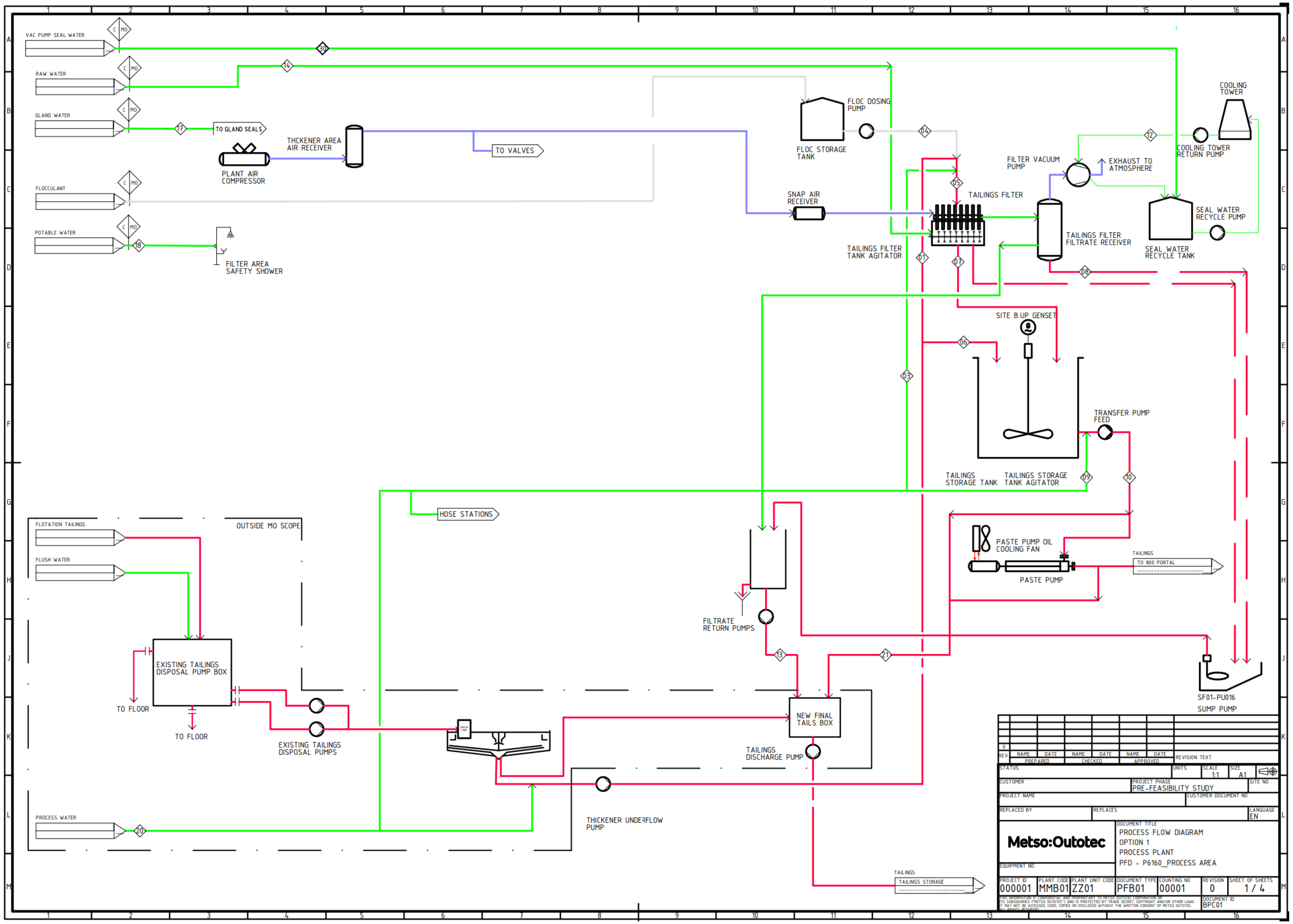
- Ground granulated blast furnace slag (slag) is added to the paste tailings at the 800 Portal and the slag-paste is transferred underground using a second pump stage. Slag is added in this location because this constitutes the most significant binder component, and therefore minimize the quantity of binder to be transported underground. This is also convenient because addition of slag to the paste tailings doesn't not change the rheological behaviour (making for a convenient pumping duty) and because slag alone would not hydrate and block pipelines in the case of a shutdown. It should be noted that slag is not added at the process plant because significant quantities of this valuable product would be lost when flushing the system. It should also be noted that conventional paste pumping systems are not capable of transporting paste between the Process plant and the 1205 level therefore, a transfer pumping station is required at or near the 800 Portal area.

- GP cement is added at the 1205 level mixing station before gravity feeding the paste into hydraulic piston pumps (on the 1170 level) for distribution throughout the mine. GP cement is to be added at low addition ratios (expected to be 1% by weight), meaning that haulage of this material underground is manageable. Addition of GP cement increases the paste yield stress considerably. It should be noted that the 1205/1170 level location was selected because pump transfer station at this site permits paste to be transported to all extremities of the Kora orebody using conventional hydraulic piston pumps, with no additional transfer stations.

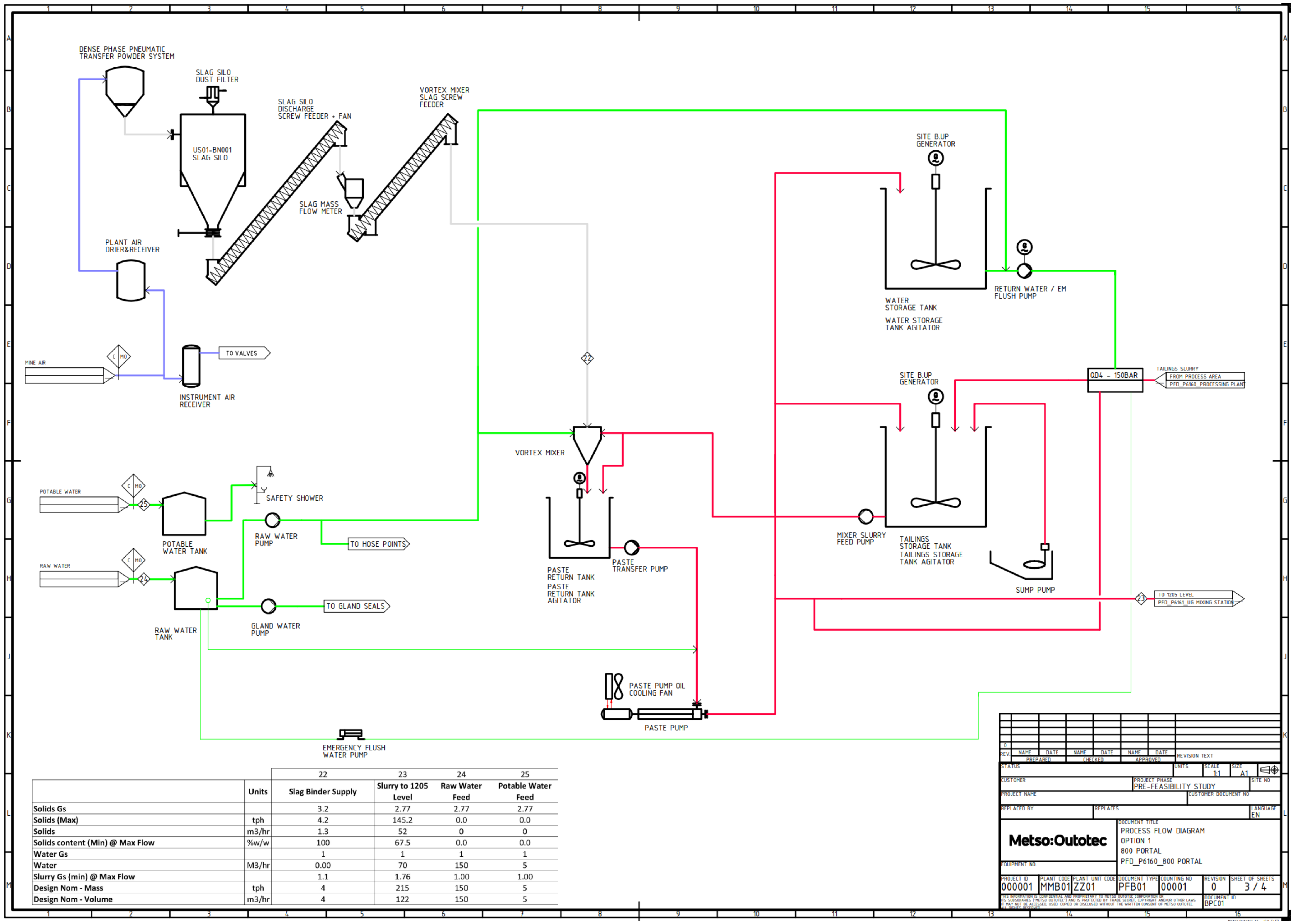
Process Flows

The process flow diagrams at the Process Area, 800 Portal Area and 1205/1170 Area are presented in Figure 13.14.14a, b and c, respectively.

Figure 13.14.14: Kora Paste Fill System Process Flow Diagram at the (a) Process Area, (b) 800 Portal Area and (c) 1205/1170 Level Area

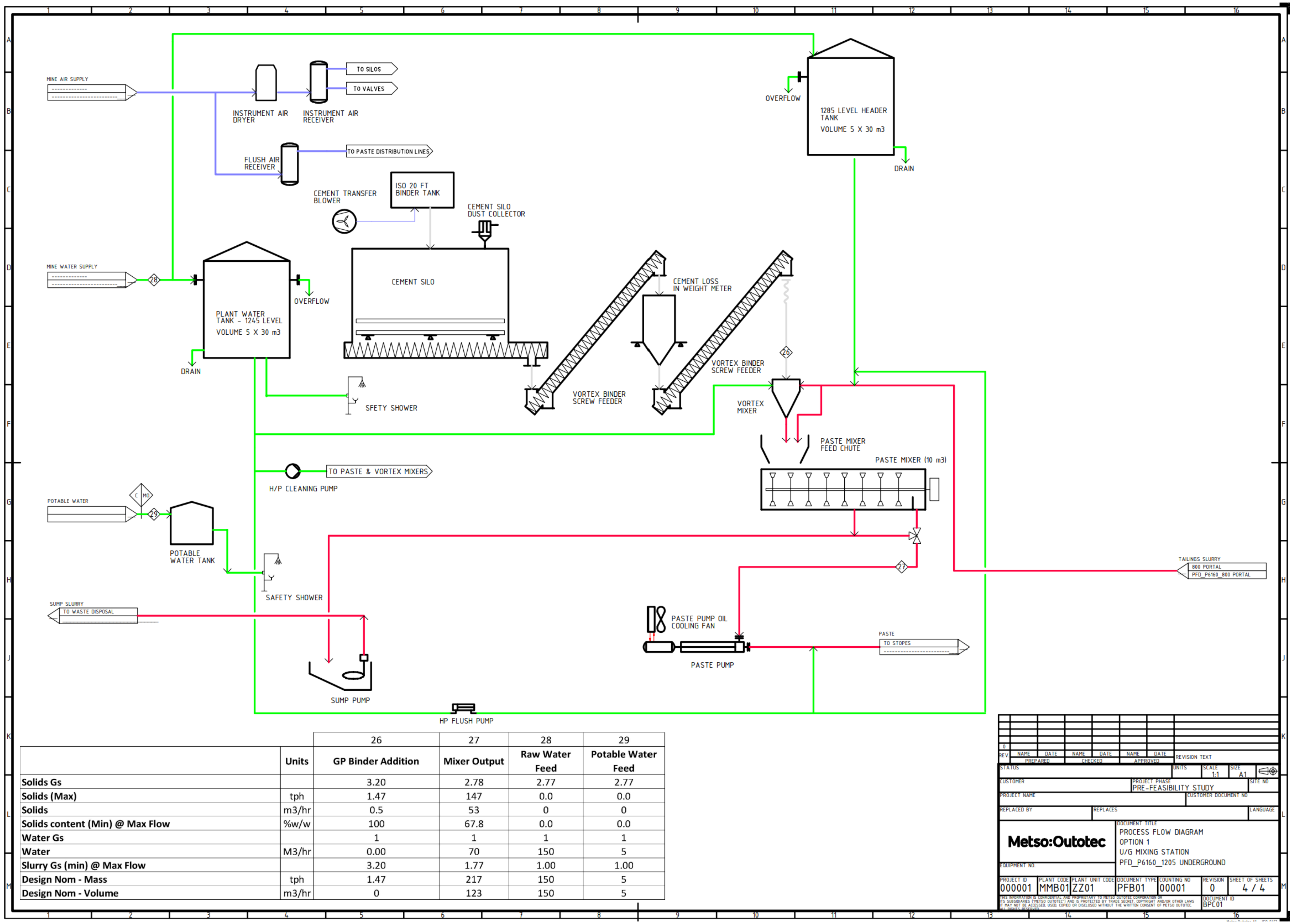


(a)



	Units	22 Slag Binder Supply	23 Slurry to 1205 Level	24 Raw Water Feed	25 Potable Water Feed
Solids Gs		3.2	2.77	2.77	2.77
Solids (Max)	tph	4.2	145.2	0.0	0.0
Solids	m3/hr	1.3	52	0	0
Solids content (Min) @ Max Flow	%w/w	100	67.5	0.0	0.0
Water Gs		1	1	1	1
Water	M3/hr	0.00	70	150	5
Slurry Gs (min) @ Max Flow		1.1	1.76	1.00	1.00
Design Nom - Mass	tph	4	215	150	5
Design Nom - Volume	m3/hr	4	122	150	5

(b)



	Units	26 GP Binder Addition	27 Mixer Output	28 Raw Water Feed	29 Potable Water Feed
Solids Gs		3.20	2.78	2.77	2.77
Solids (Max)	tph	1.47	147	0.0	0.0
Solids	m3/hr	0.5	53	0	0
Solids content (Min) @ Max Flow	%w/w	100	67.8	0.0	0.0
Water Gs		1	1	1	1
Water	M3/hr	0.00	70	150	5
Slurry Gs (min) @ Max Flow		3.20	1.77	1.00	1.00
Design Nom - Mass	tph	1.47	217	150	5
Design Nom - Volume	m3/hr	0	123	150	5

(c)

Detailed Process Description

The paste system is to include the following:

Process Area

- Extract tailings from the boot of a new tailings thickener (thickener by others) at ≥59% solids and a flowrate of 150 m^3/hr.

- From the thickener, one stream (of nominally 80 m^3/hr) is drawn and fed (through mass flow metering) to a 176 m^2 vacuum disc filter that is mounted immediately above a 200 m^3 agitated mixing tank. The stream fed to the tailings filter is combined with flocculant at nominally 40 g/t. Capacity exists to increase floc dosage to 100 g/t, if required. A photograph showing the filter cake is presented in Figure 13.14.15a.

- A second thickener underflow stream (of nominally 70 m^3/hr) is fed directly into the agitated mixing/storage tank (i.e. bypassing the filter).

- Filtrate water is gravity fed to a return water pump where it is pumped back into the proposed plant process water system.

- The agitator/mixing tank is fitted with a high-power mixing propeller the break down filter cake and blend this with the direct feed slurry, to form paste tailings. The design mixing tank outflow is 67% solids content, and is expected to have a yield stress greater than 50 Pa. A photograph showing the consistency of this material is presented in Figure 13.14.15b.

- Outflow from the agitated storage tank is fed through large entry opening froth pump via. a series of pressure transducers, a density gauge and flowmeter to a Piston diaphragm pump. Between the mixing tank isolation valve and the froth pump a water injection line is included. This water injection line is included to regulate the tank outflow density, ensuring that the rheological characteristics of the paste consistently match the design targets for transfer to the 800 Portal.

- The pipe route from the centrifugal (forth) pumps is configured to permit the mixing tank outflow to bypass the piston diaphragm pump and return out of spec paste directly to the final tailings hopper, for disposal at the TSF.

- Initially the froth pumps are configured to transfer paste to the final tailings box. This line is fitted with a series of pressure transducers and a density gauge to confirm the pipe friction loss. After the target flowrates and solids contents and line friction loss are satisfied paste is to be fed to the piston diaphragm pump to commence transfer to the 800 Portal, via the overland pipeline.

Figure 13.14.15 Kainantu Tailings as (a) Filter Cake and (b) Thickener Underflow at 66% Solids

(a) (b)

- From the Process area, paste is to be transferred approximately 6000 m laterally and 355 m vertically to the 800 Portal using a single stage piston diaphragm pump. This material is to be transported through a predominantly welded 200NB Schedule 120 pipeline. At 67% solids and a flowrate of nominally 120 m^3/hr is expected to have a line velocity of approximately 1.3-1.4 m/s. The operating pump discharge pressure is expected to be 17 MPa, with a line and pump design capacity of approximately 19.2 MPa.

- Immediately downstream of the paste pump would be a T-piece (and high-pressure knife gate valve) that would permit backflush of the paste line from the 800 Portal back to the final tailings hopper at the process plant.

- The overland pipeline is fitted with a series of 2 outlet valves and scour pits to assist with cleaning of the system in the case of a blockage. At each scour pit location an expansion loop is included to accommodate thermal pipeline expansion.

- At the 800 Portal the paste tailings are deposited into a 200 m^3 agitated storage tank. While the operating level for this tank is expected to remain low, the relatively large volume is required to provide surge capacity for flushing activities.

- From the storage tank, paste tailings are combined with slag (at addition rates of 3% - 8%) in a vortex mixer / agitator arrangement. As the paste would be at a yield stress of around 50 Pa, vortex blending followed by mixing in an agitator tank is considered an adequate mixing strategy. To ensure satisfactory operation of the vortex mixer a bypass line (with flow control valve) would be included. The vortex mixer feed line is to be fitted with a pinch valve, and vortex mixer fitted with a level sensor, to ensure feed flows can be regulated to achieve optimal vortex mixer performance. In addition to the tailings slurry injection point, the vortex mixer is also fitted with a water injection point that can be used to assist in maintaining satisfactory vortex flows and also be used for paste dilution control.

- Slag is to be delivered in 20-foot shipping containers and the 800 Portal infrastructure is inclusive of a 20-ft container tipper and dense phase unloading / transfer system. This infrastructure allows containers to be delivered on a conventional flat-bed truck. This system is designed to permit 20-ft containers to be unloaded into a 300 m^3 storage silo over a period of approximately 1.5 hours.

- Slag is fed out of the 300 m^3 silo using a variable speed screw feeder that feeds through a continuous Coriolis style mass flow meter, before a second screw feeder that transfers the binder to the vortex mixer where it is combined with the paste.

- The slag/paste is blended in the agitated mixing tank and the tank underflow passes into a large opening froth style centrifugal pump that feeds a second Piston diaphragm pump.

- This pump transports paste approximately 3,400 m laterally and 400 m vertically to the 1205 level. This pump has an expected operating discharge pressure of 12.0 MPa, but the pump and pipeline are sized to permit a discharge pressure of 20 MPa, to provide operational flexibility.

- At the 1205 level mixing station paste is fed directly into a vortex mixer, where it is combined with nominally 1% (w/w) of GP cement, before flowing into a 10 m^3 paste mixer. A bypass line with flow control valve would be included to regulate vortex mixer flows ensuring optimal performance. The bypass line is to be fitted with a Coriolis style mass flow meter to determine the mixer feed density. The coriolis style density metering system is adopted for this application to eliminate the need for an underground nucleonic source.

- While the required GP cement addition rate is relatively small, and the paste yield stress is relatively low, as quality control sampling is to be undertaken at the 1205 m mixing station, the additional cost associated with a large robust mixer is justified based on the increase in quality control consistency. As such a 10 m^3 (4 m^3 active volume) paste mixer, with expected operational resonance time of 1.8 min has been selected for the duty.

- GP cement is to be delivered directly to site in 20 t ISO containers. These ISO containers are to be transported directly (from the supplier's Lae distribution centre) underground where the content is blown into a 50 t horizontal binder silo. From the silo, binder is fed through a constant speed transfer screw feed auger, which loads a 'Loss in Weight' (LIW) hopper. Binder is metered out of the LIW hopper into the vortex mixer using a variable speed screw feeder, where the discharge speed is regulated to achieve the target weight loss.

- The vortex mixer is to be fitted with a mine water injection nozzle to permit water addition for vortex flow control and paste density control.

- The outflow from the mixer is at a solids content of nominally 67% solids (w/w). Due to the increase in pH, which comes about as a result of the GP addition, the paste yield stress is expected to increase to 200-300 Pa.

- Outflow from the mixer passes through a diverter valve, from where flows could be diverted to either the hydraulic piston pumps or into a sump for disposal. The pumps are to be located on the 1170 level, with a 35 m^3 and 60 m^3 dump sump located on the 1205 and 1170 levels, respectively.

- Due to the higher yield stress, cemented paste pumping is to be undertaken using hydraulic piston pumps. This pump is configured to distribute paste to the extremities of the Kora orebody. For stopes beneath the 1720 level, flow modelling indicates that paste could be distributed at 66-67% solids. However, due to pump limitations, transportation to stopes above the 1720 level would require the paste to be further diluted and it is expected that these stopes would be filled with paste at 64-65% solids. To compensate for the lower density paste a slight increase in binder addition would be required. In cases where no pumping is required (all stopes beneath the 1060 Level and some centrally located stopes below the 1170 m Level) paste can be transported under gravity (i.e. no pumping).

- Compressed air is to be taken from the mine air system and stored in a 14 m^3 flush air receiver and a 1 m^3 instrument air receiver. Flush air would feed directly from the mine air system, while that fed to the instrument receiver (used for both instruments binder aeration) would pass through a dryer before entering the receiver.

- Process water is to be stored in a series of 5 x 30,000 L tanks on each of the 1245 and 1285 levels. Process water is located at these locations to ensure adequate plant water supply and suitable gravity head for reticulation flushes.

- Normal 'back flushing' of the paste delivery line from the 800 Portal would be completed using the 1285 level header tanks, but the high-pressure flush pump is also configured to flush this line when required.

Start-up / Shut-down Considerations

During operation the rheological properties and density are measured in real time, between the Process area mixing tank and the piston diaphragm pump, and to protect the overland pipelines any paste that is measured to be out of specification is diverted to the final tailings box for disposal. This is particularly important during start up.

When executing a "long term" the Process area – 800 Portal pipeline is to be backflushed to the final tailings hopper, while paste in the line between the 800 Portal and the 1205 is to be backflushed to the waste paste and tanks at the 800 Portal. Backflushing enables the assistance of gravity to clear these lines and also enables the wasted paste material to be disposed of at the tailings storage facility.

Cemented paste, downstream of the 1170, is to be flushed into the stope area for disposal using high pressure pumps and / or a combination of water and an air flush system.

Due to the fine nature of the Kainantu tailings, testing showed this material to remain in a stable state for up to 4 hours when uncemented and when containing small portions of slag. To facilitate fast stope changeover, in cases where the system is shut down for less than 4 hours, only partial backflushing of the Process plant – 800 Portal and 800 Portal – 1205 pipelines is required. In this case backflow valves are opened to enable gravitational backflow and dissipation of line pressures, but after the gravitational backflush, the remaining paste is to remain static in the lines. Pumps have been sized to enable this static paste to be remobilized upon the resumption of filling.

An emergency shutdown situation may come about as a result of either a power outage or failure of a paste pump. In this situation:

- Provision has been made for a high-pressure water pump, and dedicated backup generator, at the 1170 level for flushing for the cemented paste backfill line between the 1170 and the stope.

- A series of header tanks located on the 1285 level provide gravity head for clearing the line between the 800 Portal and the 1205 level. If additional pressure is required the high pressure water pump, located on the 1170 level can also be used for this, duty after clearing the cemented paste from the system.

- A high pressure - high flow water pump, powered by site wide backup power, is to be used to backflush the 800 Portal – Process plant line.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The effective date of the MRE for the Kora lodes is the 31 October 2021, which was the date that the latest database was received by H&SC. The effective date of the MRE for the Judd lode is the 31 December 2021, which was the date that the latest database was received by H&SC.

The MRE for the Kora Consolidated 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 Consolidated deposits is relatively straightforward. A 3D interpretation of geological 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 data composites 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 seven 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 production.

From the 2020 / 2021 drilling the geological interpretation of mineral domains has been confirmed with only very minor modifications to the previous 2020 Kora Consolidated domain designs, although locally there has been 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 has been offset by 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/stringer systems. These veins appeared at times to be parallel to the bounding K1 and K2 structures, and at other times they appeared slightly oblique, transecting from the K2 footwall to the K1 hangingwall. The lode has been subject to further geological reassessment following a substantial amount of infill drilling and some face sampling such that H&SC now consider 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.

The Judd lode is a parallel structure to the main Kora Consolidated mineral system, approximately 150m to the east. The vein system is slightly narrower than the K1 or K2 lode but has a very similar dip and strike. A 3D geological interpretation of the lode was developed by H&SC based on a supplied interpretation by K92ML and is consistent with the geological interpretation for the main Kora K1 & K2 lodes. Geological continuity of the lode is regarded as good for the bulk of the drilling / sampling except for the southern end where the lode breaks down into a series of much narrower and lower grade lodes that seemingly are non-aligned with the main lode (further drilling is required to resolve this).

The MRE are reported at a 1.75g/t Au cut off, up from 1g/t Au in 2020, which is based on mining studies completed by K92ML. The global grade tonnage curves indicate a consistent shape to the curves compared to the 2020 estimation results. The MRE are in line with H&SC's expectations based on a combination of K92ML drilling strategy and the 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.

The MRE reported in this section have been classified under the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves.

14.1.1 Resource Estimates

The updated Global Mineral Resource estimate (using a 1.75g/t gold cut-off) for the Kora Consolidated deposit effective 31 October 2021 is tabulated below:

Table 14.1.1 2022 Kora Consolidated Resource Estimate

Kora									
Category	**Mt**	**Au g/t**	**Au Moz**	**Ag g/t**	**Ag Moz**	**Cu %**	**Cu Kt**	**Au_Eq g/t**	**Au_Eq Moz**
Measured	2.8	9.07	0.8	15.7	1.4	0.85	24.1	10.51	1.0
Indicated	4.4	6.68	0.9	20.2	2.8	0.97	42.4	8.35	1.2
Total M & I	**7.2**	**7.62**	**1.8**	**18.4**	**4.3**	**0.92**	**66.4**	**9.20**	**2.1**
Inferred	8.1	7.12	1.8	27.3	7.1	1.38	111.1	9.48	2.5

The Global Mineral Resource estimate (using a 1.75 g/t gold cut-off) for the Judd deposit effective 31 December 2021 is tabulated below:

Table 14.1.2 2022 Judd Resource Estimate

Judd									
Category	**Mt**	**Au g/t**	**Au Moz**	**Ag g/t**	**Ag Moz**	**Cu %**	**Cu Kt**	**Au_Eq g/t**	**Au_Eq Moz**
Measured	0.22	11.26	0.08	19.9	0.14	0.72	1.59	12.56	0.09
Indicated	0.15	7.46	0.04	13.9	0.07	0.77	1.20	8.76	0.04
Total M & I	**0.38**	**9.70**	**0.12**	**17.5**	**0.21**	**0.74**	**2.79**	**11.00**	**0.13**
Inferred	1.01	4.24	0.14	11.0	0.36	0.87	8.82	5.66	0.18

Table 14.1.3 2022 Combined Kora Consolidated and Judd Resource

Kora and Judd									
Category	**Mt**	**Au g/t**	**Au Moz**	**Ag g/t**	**Ag Moz**	**Cu %**	**Cu Kt**	**Au_Eq g/t**	**Au_Eq Moz**
Measured	3.1	9.23	0.9	16.0	1.6	0.84	25.7	10.66	1.0
Indicated	4.5	6.70	1.0	20.0	2.9	0.97	43.6	8.36	1.2
Total M & I	**7.6**	**7.72**	**1.9**	**18.3**	**4.5**	**0.91**	**69.2**	**9.29**	**2.3**
Inferred	9.1	6.80	2.0	25.5	7.4	1.32	119.9	9.05	2.6

- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
- Resources were compiled at 1.75,2.5,3,4,5,6,7,8,9 and 10 g/t gold cut-off grades for Kora and 1.75,2.5,3,4,5 for Judd.
- Density (t/m ^{3}) is on a per zone basis, K1, K2: 2.84 t/m ^{3}; Kora Link: 2.74 t/m ^{3}; Judd: 2.71 t/m ^{3}; Waste: 2.67 t/m ^{3}.
- Minimum mining width for wireframes: Kora: 5.2 m; Judd: 5.2 m.
- Reported tonnage and grade figures are rounded from raw estimates to reflect the order of accuracy of the estimate.
- Minor variations may occur during the addition of rounded numbers.
- Estimations used metric units (metres, tonnes, and g/t).
- Gold equivalents are calculated as AuEq = Au g/t + Cu%*1.607*92.8% + Ag g/t*0.0125*89%. Gold price US$1,600/oz; Silver US$20/oz; Copper US$3.75/lb. Metal payabilities and recoveries are incorporated into the AuEq formula. Recoveries of 92.8% for copper and 89% for silver.

The key to the confidence of the resource estimates is the apparent good reconciliation of the block model with the mill production 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.

Details of the MRE for Kora and Judd deposits are in Section 14 of the 'Independent Technical Report, Mineral Resource Estimate Update, Kora and Judd Gold Deposits, Kainantu project, Papua New Guinea', effective date 20 January 2022 which is filed on SEDAR.

15.0 MINERALS RESERVE ESTIMATES

The Mineral Reserve estimate was prepared by Patrick McCann (PEng), of Entech Mining Ltd and Shane McLeay (FAusIMM) of Entech Pty Ltd, 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. Patrick McCann and Shane McLeay are independent consultants of the Company and are Qualified Persons as defined by NI 43-101.

The total Mineral Reserve for the Kainantu Project is shown in Table 15.1. The Mineral Reserve estimate is based on the Judd Mineral Resource Estimate (effective date 31 December 2021) and the Kora Mineral Resource Estimate (effective date 31 October 2021), net of post-resource mining depletion from 1 November 2021 to 31 December 2021 of 36,765 tonnes at 12.94 g/t Au, 0.59 % Cu and 9.29 g/t Ag.

Table 15.1 Kainantu Mineral Reserve Statement (Effective Date 1 January 2022)

	Tonnes	Gold		Silver		Copper		AuEq	
	Mt	g/t	moz	g/t	moz	%	kt	g/t	moz
Kora									
Proven	2.26	7.58	0.55	14.96	1.09	0.82	18.52	9.17	0.67
Probable	3.55	5.88	0.67	19.46	2.22	0.95	33.90	7.76	0.89
Total P&P	**5.81**	**6.54**	**1.22**	**17.71**	**3.31**	**0.90**	**52.41**	**8.31**	**1.55**
Judd									
Proven	0.21	9.99	0.07	16.88	0.11	0.57	1.17	11.18	0.07
Probable	0.14	6.50	0.03	10.65	0.05	0.59	0.81	7.65	0.03
Total P&P	**0.34**	**8.60**	**0.09**	**14.40**	**0.16**	**0.58**	**1.98**	**9.77**	**0.11**
Kora and Judd									
Proven	2.46	7.78	0.62	15.12	1.20	0.80	19.69	9.34	0.74
Probable	3.69	5.90	0.70	19.13	2.27	0.94	34.70	7.75	0.92
Total P&P	**6.15**	**6.65**	**1.32**	**17.53**	**3.47**	**0.88**	**54.39**	**8.39**	**1.66**

- 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 US$1,600/oz gold, US$4.00/lb Copper, US$20/oz Silver.

16.0 MINING METHODS

16.1 Introduction

Entech was engaged by K92 Mining Inc. (K92) to complete a Feasibility Study (FS) 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 K92 since October 2016 and has a current processing throughput of 400,000 tpa. The FS targets an expansion to a peak processing rate of 1,200,000 tpa with a 7-year mine life. The FS assumes a start date of the life of mine schedule of 1 January 2022.

16.2 Geotechnical

A total of 3,662 m of drill core was geotechnically logged from core photographs in detail for the Kora underground mining assessment during Q1 and Q3 2021. The drillholes were selected from existing exploration holes. From this total, 1,946 m was logged as part of an initial campaign, with a second campaign of 1716 m completed in October 2021. In addition to the 3662 m, 472.3 m of core was validated and logged associated with the Judd orebody.

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 K92 staff and provided to Entech.

Rock property testing specific to the orebodies was unable to be sampled and sent for testing again due covid restrictions, resourcing availability in country, and available drill holes for sampling. 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 still remain a significant information gap for the project and should be addressed as soon as practical. The drill core logging data was analysed by Entech and forms the basis of this study. This information has allowed for characterization of the rock mass, assessment of stable stoping span predictions and estimates of dilution factors.

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 utilized for visualization 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 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.

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, and 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 both the Kora and Judd orebodies can be classed as good ground. Table 16.2.1 outlines the recommended stope spans and dilution estimates for the Kora and Judd orebody.

Table 16.2.1 Summary of Assessed Stoping Parameters for Kora and Judd

Orebody	Parameters	Hangingwall	Footwall
K1	Allowable Strike Length	16 m	20 m
	Dilution	0 - 0.5 m	0 - 0.5 m
K2	Allowable Strike Length	31 m	19 m
	Dilution	0 - 0.5 m	0 - 0.5 m
Judd	Allowable Strike Length	35 m	35 m
	Dilution	0 - 0.5 m	0 - 0.5 m

The presence of the FGZ and pervasive structure throughout the rock units will dominate stope wall behaviour with the possibility of slabbing, sliding and unravelling failure types occurring 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 will be the key to minimising stoping performance issues.

16.3 Current Site Mining Methods

Currently Avoca and modified Avoca stoping methods (as described below) are used on site.

16.4 Mining Method Selection Process

The 2020 PEA, in combination with current site mining practices, are 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 mechanized 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 two 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 optimization parameters for each of the potential mining methods.
- Completed stope optimizations over a range of cut-off grades (COG).
- Estimated mining physicals and mining costs.
- Produced excel based schedules with stope optimization 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 16.4.1 shows the four key criteria applied against the considered mining methods, these criteria being technical feasibility, mining cost, resource recovery, and production rate.

Table 16.4.1 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 mechanized mining methods with low production rates such as cut and fill were discounted. The mining method long list is shown in Figure 16.4.1.

Figure 16.4.1 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 two years of mining all three LHS methods remained for consideration in the short list. For the remaining LOM, the key consideration was that the mining method must result in a reduction in surface tailings storage facility requirements due to limited capacity. 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.

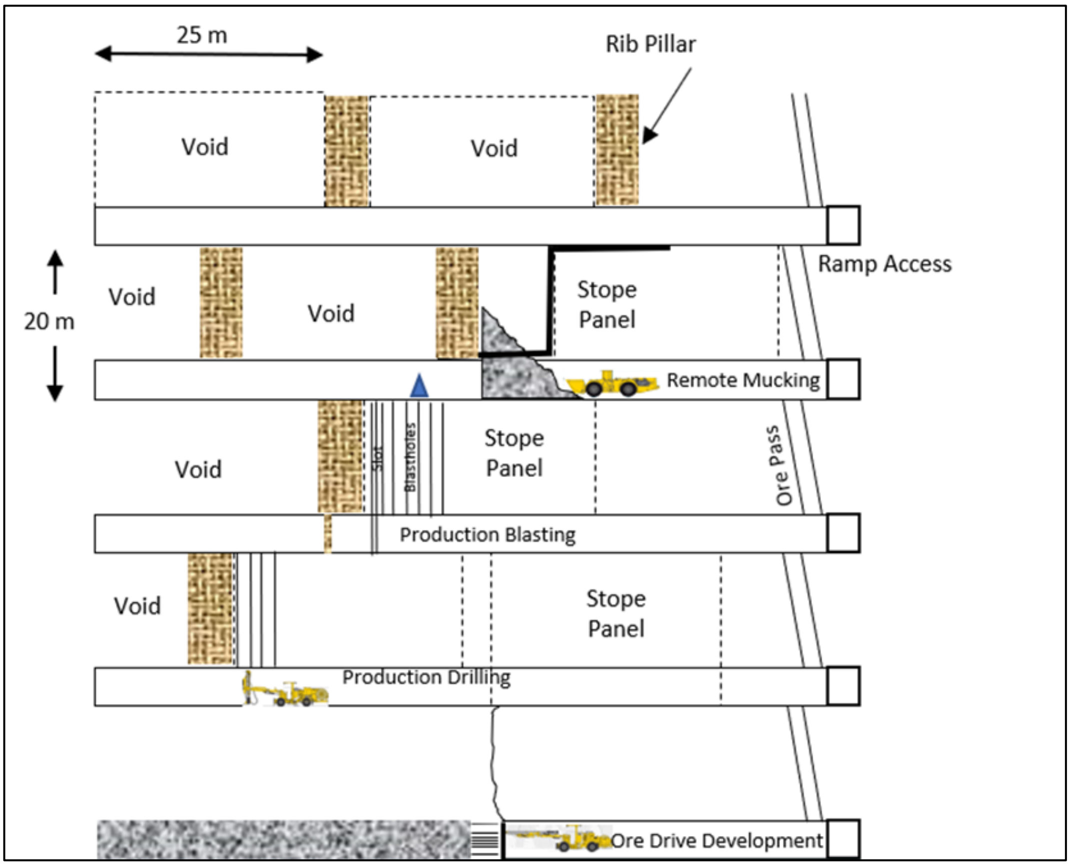
16.4.2 Short Listed Mining Methods

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 mechanized 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 16.4.2.

Figure 16.4.2 Example Long Section of Longitudinal LHOS Extraction



Avoca

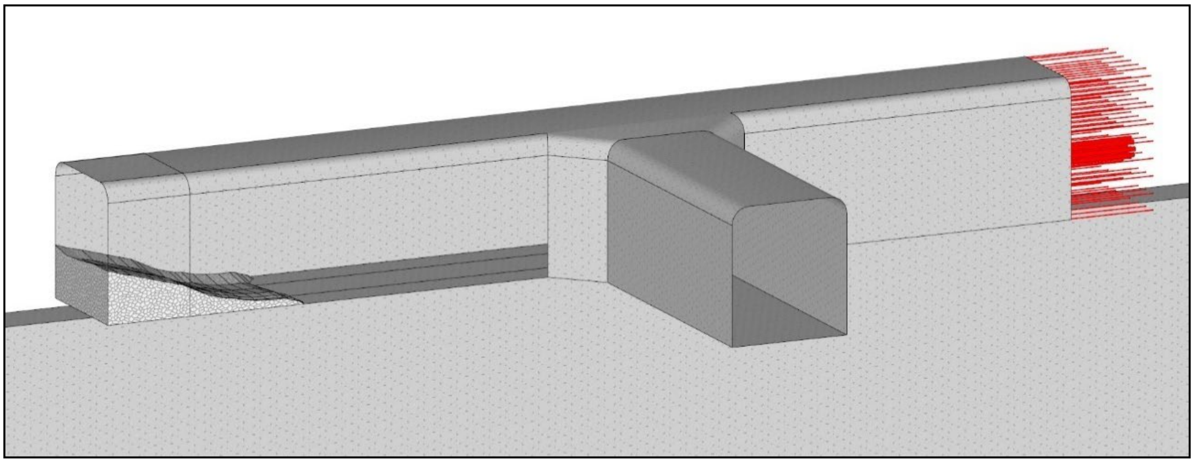
Avoca is a mechanized mining method which is widely used in global underground mining. The proposed Avoca stoping design assumptions included a 20m vertical level spacing, a bottom-up extraction sequence, and 76 mm or 89 mm diameter down-hole production drilling. The stoping panels are subdivided into 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 is repeated. In Avoca, there is access to the stoping horizon at both ends on 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 16.4.3.

Figure 16.4.3 Step 1 – Ore Development



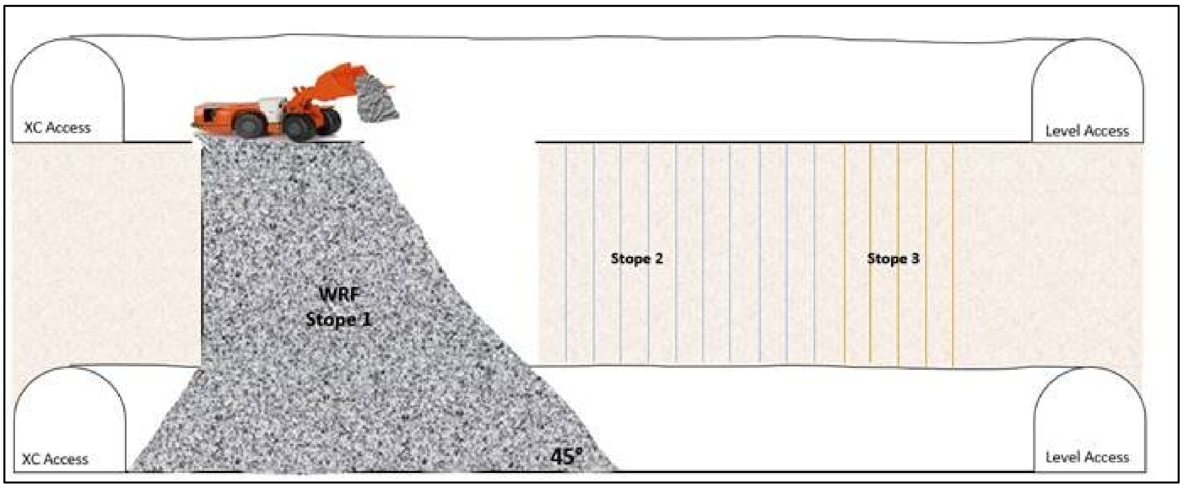
2. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block / panel. A strike length of approximately 6.0 m for each firing is typical.

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 16.4.4.

Figure 16.4.4 Avoca Overall Mining Method Schematic



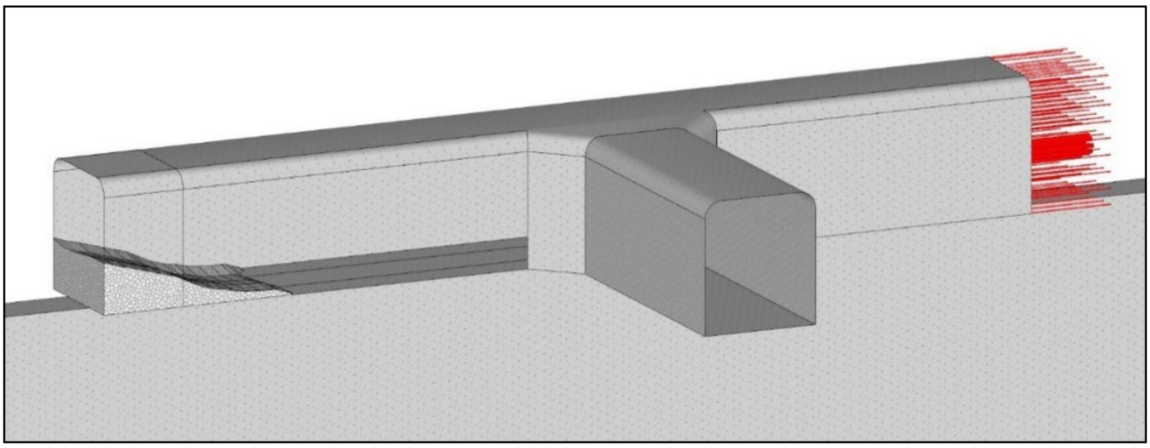
Modified Avoca

Modified Avoca is a mechanized 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:

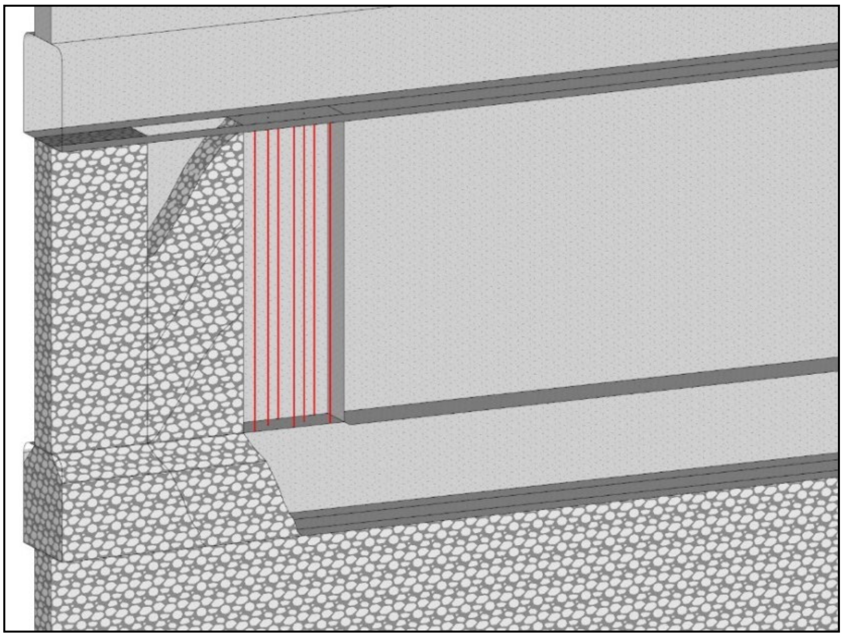
Ore drives are developed along strike at the top and bottom of the stoping block or panel using jumbo drills as per Figure 16.4.5.

Figure 16.4.5 Step 1 – Ore Development



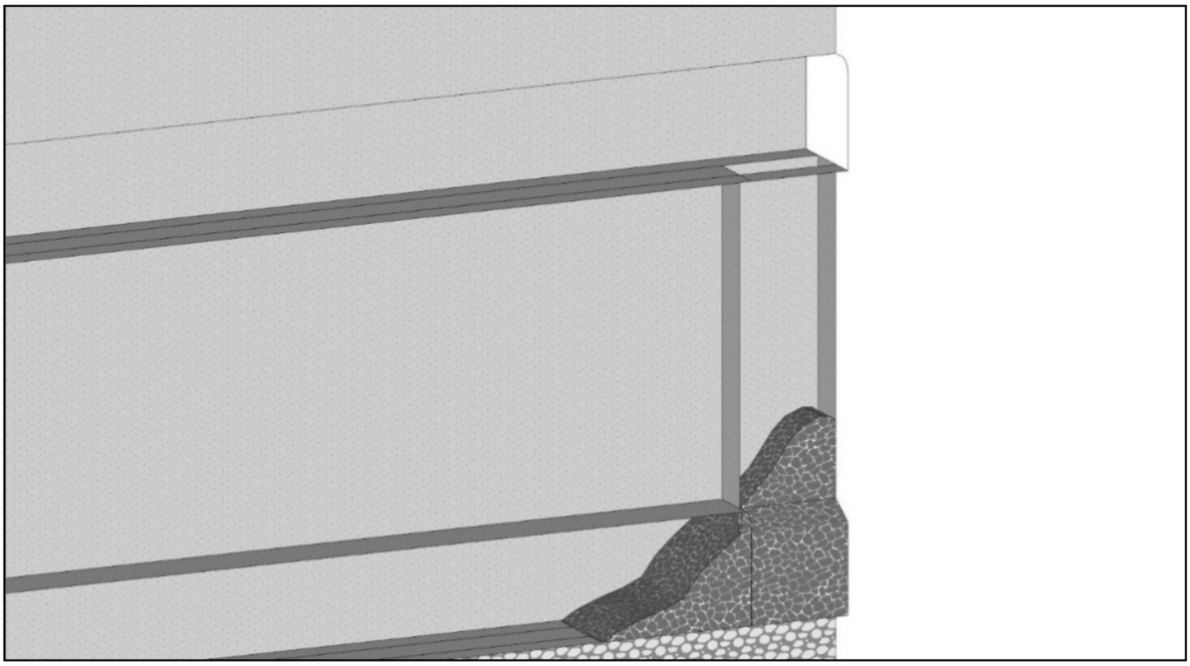
1. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block. A strike length of 5 m for each firing is shown in this example as per Figure 16.4.6.

Figure 16.4.6 Step 2 – Stope Drill & Blast



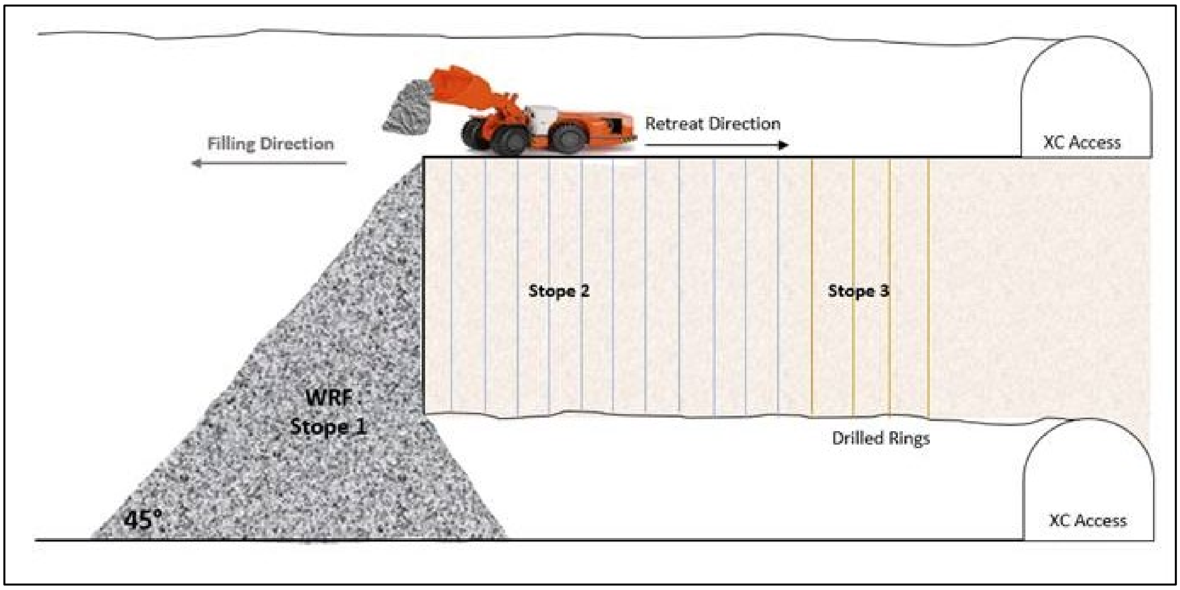
2. Blasted ore is bogged to level stockpiles for truck loading using conventional diesel-powered underground loaders, utilising both conventional (i.e., manual) and tele-remote techniques as in Figure 16.4.7**.**

Figure 16.4.7 Step 3 – Stope Loading



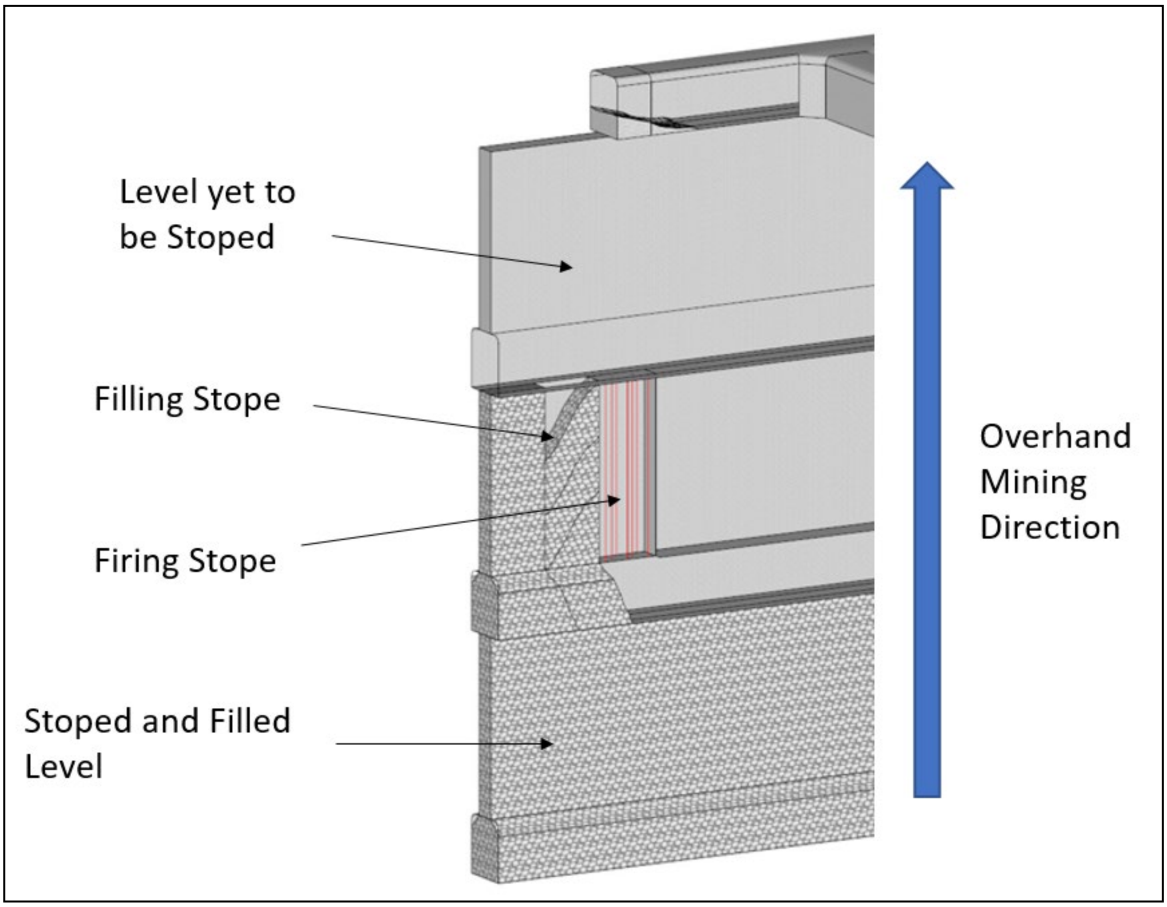
3. Waste material is dumped into the stope void from the level above as per Figure 16.4.8.

Figure 16.4.8 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 16.4.9.

Figure 16.4.9 Modified Avoca Overall Mining Method Schematic

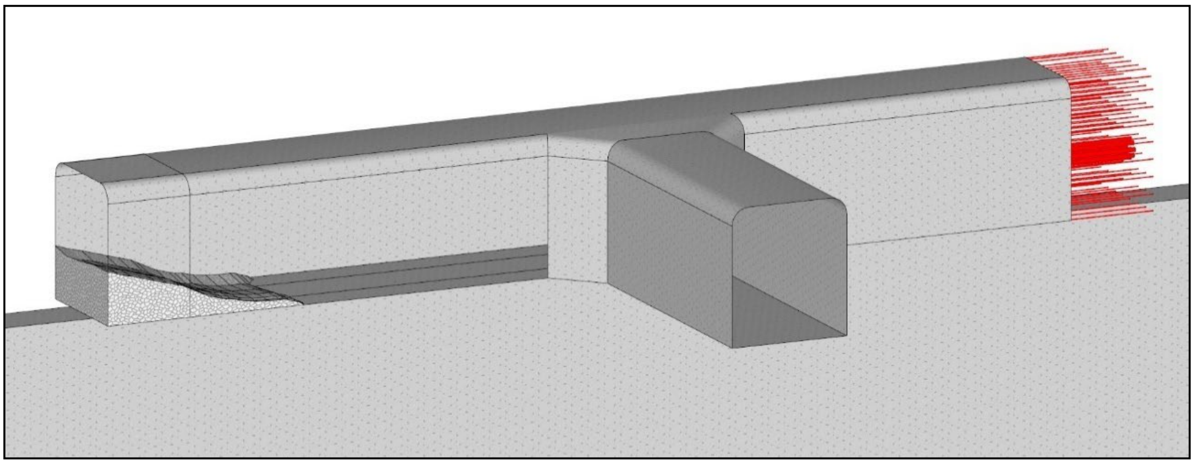


Longhole Stoping with Cemented Rock Fill

The LHS with CRF assumes a continuous fill mining method involving the following steps:

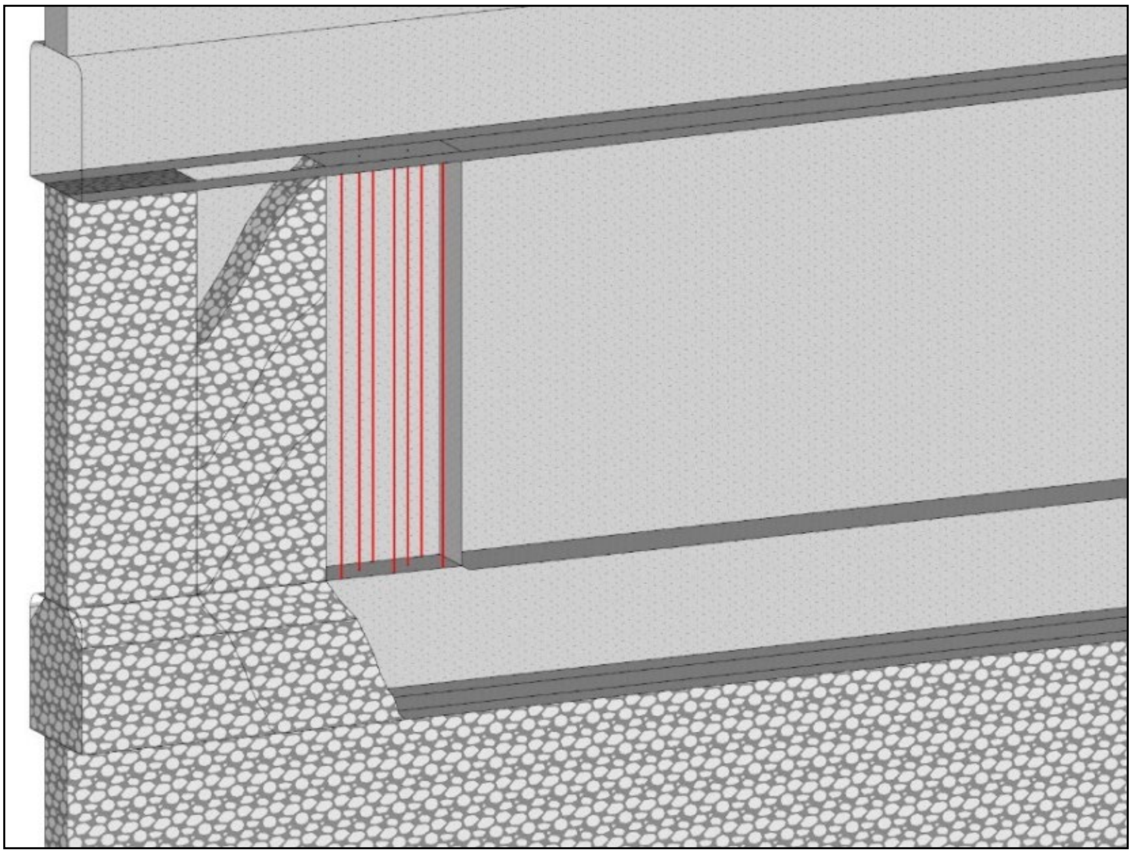
1. Ore drives are developed along strike at the top and bottom of the stoping block or panel using jumbo drills as per Figure 16.4.10.

Figure 16.4.10 Step 1 – Ore Development



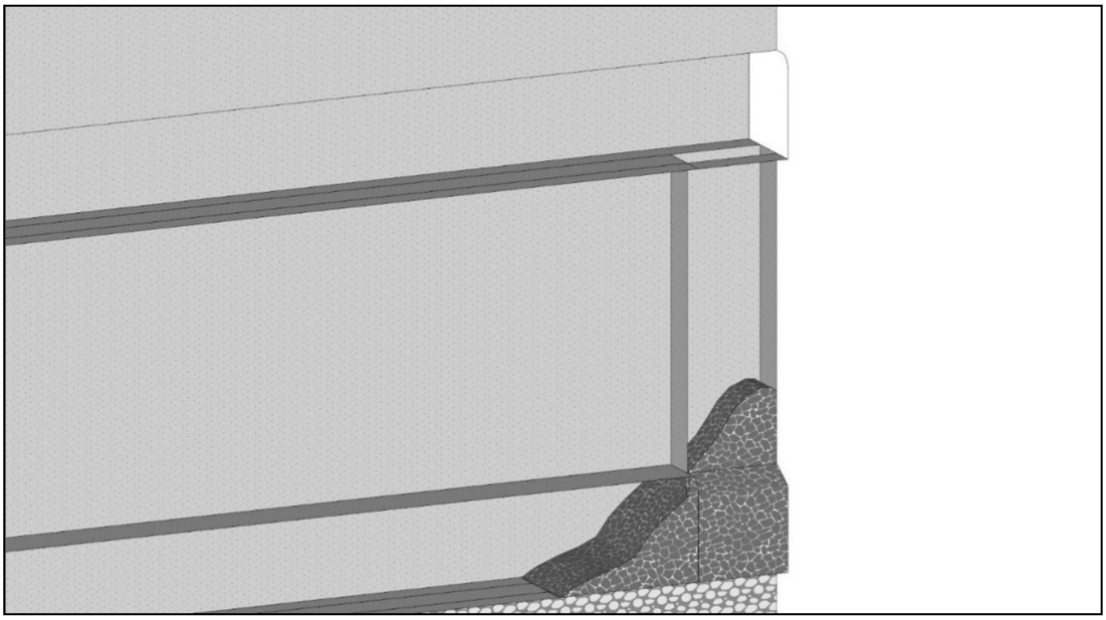
2. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block. A strike length of 5 m for each firing has been assumed in this example as per Figure 16.4.11.

Figure 16.4.11 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 16.4.12.

Figure 16.4.12 Step 3 – Stope Loading



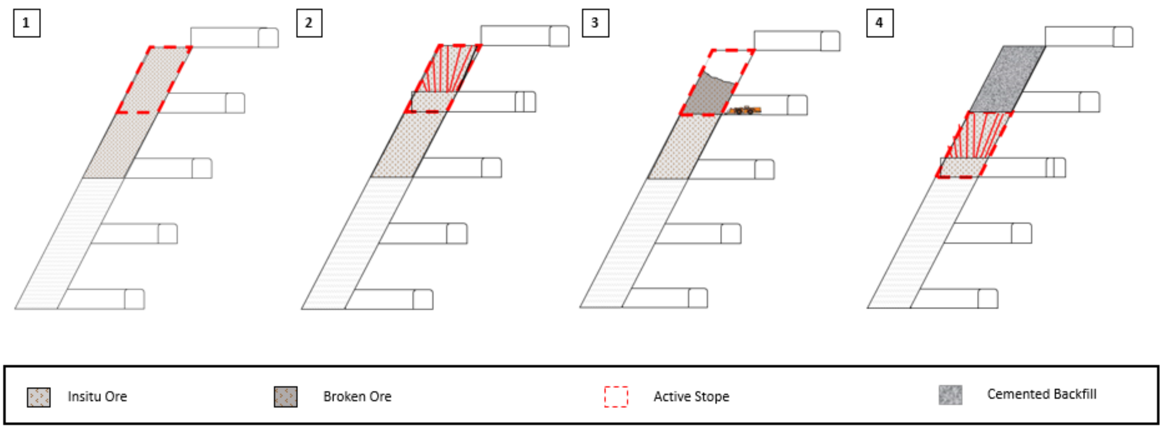
4. Waste material mixed with a cement slurry is dumped into the stope void from the level above.
5. Once filled, the next stope along is fired against the fill typically within 12-36 hours which allows for firing against the fill without requiring another slot to be established. Alternatively larger 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.

Longhole Stoping with Paste Fill

LHS with paste fill is proposed to be utilized 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 utilize transverse extraction where required, most likely 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 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) utilized in a top down / underhand mining sequence is shown in Figure 16.4.13.

Figure 16.4.13 Example Cross-section of Stoping Extraction with Paste Fill Mining Cycle



16.4.3 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 Q2 2024, and longhole stoping with pastefill for the remainder of the mine life. An economic comparison was done 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, Entech were provided with the FS geological model. Stope optimizations were completed to compare to the trade-off studies and validate the selected Avoca, and LHS with paste fill mining methods. The stope optimization results aligned with the findings of the trade-off studies and K92 confirmed selection of the Avoca/Modified Avoca and LHS with paste fill mining methods for design and scheduling of the FS. The FS mine plan only 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 the 2020 PEA and updated inputs provided by K92. The cut-off grade is typically generated as a base input for stope optimization to generate a fully costed, operating only, and incremental stoping cut-off grade. The initial cut-off grade ranges are shown in Table 16.5.1 below.

Table 16.5.1 Initial Cut-off Grade Estimation

Mining Cut-off Calculation	Units	Fully Costed	Operating Cut-off	Incremental Stope Cut-off
Sustaining Capital Cost	USD / t ore	34.80		
Mining Operating Cost	USD / t ore	41.40	41.40	31.05
Processing and G&A Cost	USD / t ore	52.21	52.21	52.21
Total Cost	**USD / t ore**	**128.41**	**93.61**	**83.26**
Gold Price	USD / ounce	1500	1500	1500
Metallurgical Recovery	%	95.0%	95.0%	95.0%
Royalties	%	2.5%	2.5%	2.5%
Revenue	$/g AuEq	44.67	44.67	44.67
Economic Stope Cut-off Grade	AuEq g/t	2.87		
Operating Cut-off Grade	AuEq g/t		2.10	
Incremental Stope Cut-off Grade	AuEq g/t			1.86t

This information was then used in the trade-off study to generate the lower bound of 3.0 g/t Au Eq. for the FS stope optimizations. To assess the appropriate final cut-off grade to use in design and scheduling, it was agreed to produce stope optimizations on a range of cut-off grades to conduct further analysis and compare results.

16.5.2 Stope Optimization

The stope optimization parameters for the FS are shown in Table 16.5.2 and Table 16.5.3 below, based on the proposed mining method. The results of the previous trade-off studies meant that a smaller range of cut-off grades was run for the FS.

Table 16.5.2 LHOS with Pillars / LHS with Paste Fill

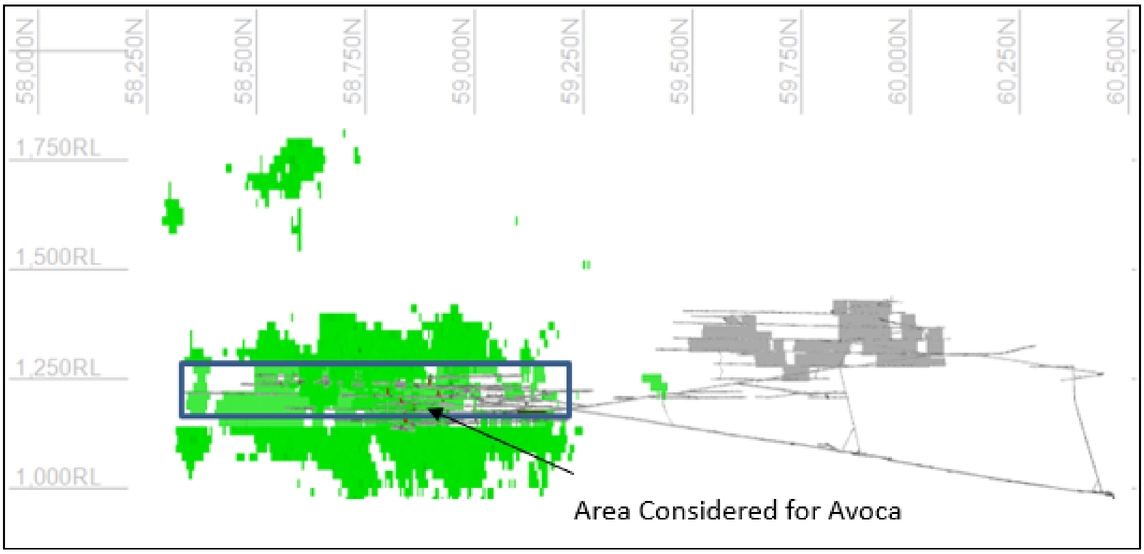
Stoping Parameter	Value
Stoping COG (g/t Au Eq.)	3,4,4.5,5,5.5
Min. Mining Width (m)	2.0
Max. Mining Width (m)	Not Limited
Vertical Level Interval (m)	20
Section Length (m)	5
HW Dilution (m)	0.5
FW Dilution (m)	0.5
Min. Parallel Waste Pillar Width (m)	7.5

Table 16.5.3 Avoca

Stoping Parameter	Value
Stoping COG (g/t Au Eq.)	3,4,4.5,5,5.5
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 (the initial two years of the mine plan as advised by K92), and therefore planned to be mined with the Avoca method, are shown in Figure 16.5.1 below. All areas outside of this rectangle are planned to be mined using LHS with paste fill.

Figure 16.5.1 Kainantu As-builts with Preliminary MSO Shapes (Green)



An MSO metal inventory to Mineral Resource metal conversion was completed to ensure that the stope optimizations 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 16.5.2, Figure 16.5.3, and Figure 16.5.4 below.

Figure 16.5.2 Kora Only MSO vs Resource - Moz AuEq (y-axis) vs Cut-off Grade (x-axis)

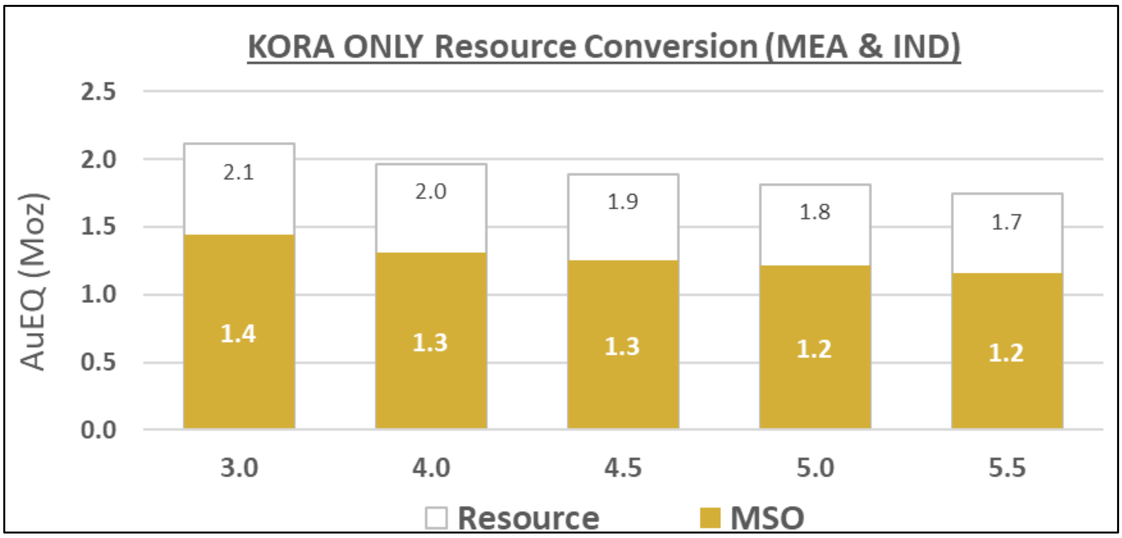


Figure 16.5.3 Judd Only MSO vs Resource - Moz AuEq (y-axis) vs Cut-off Grade (x-axis)

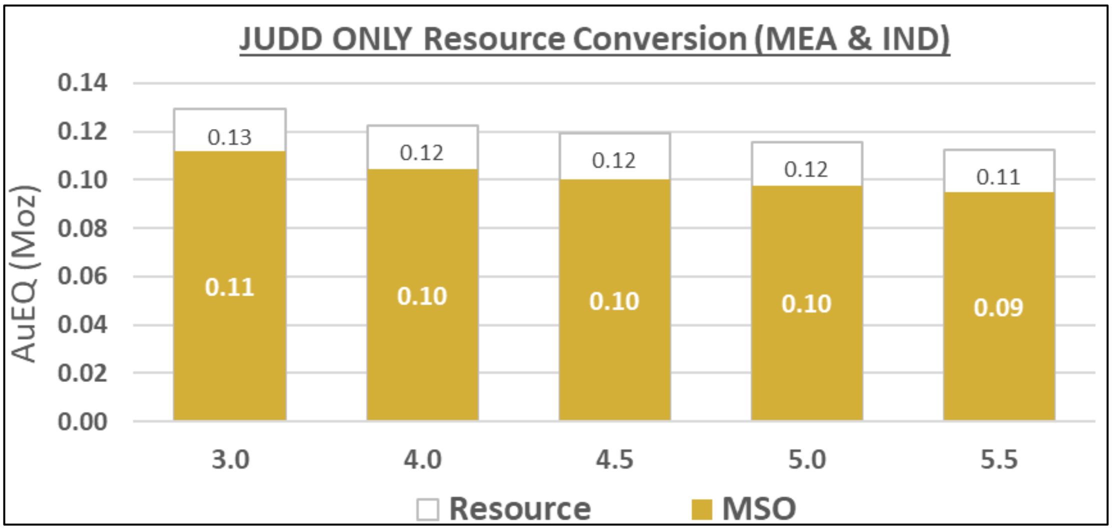
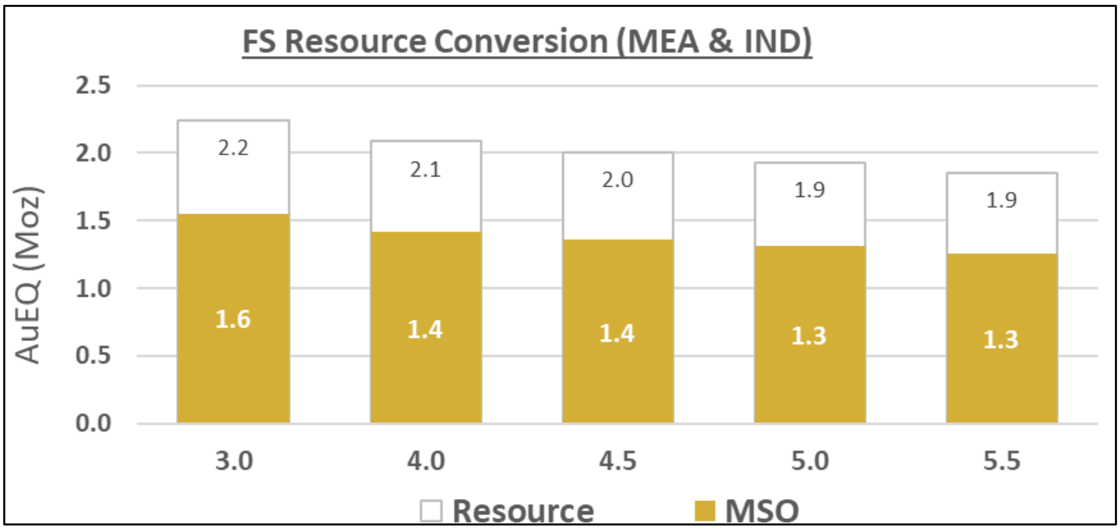
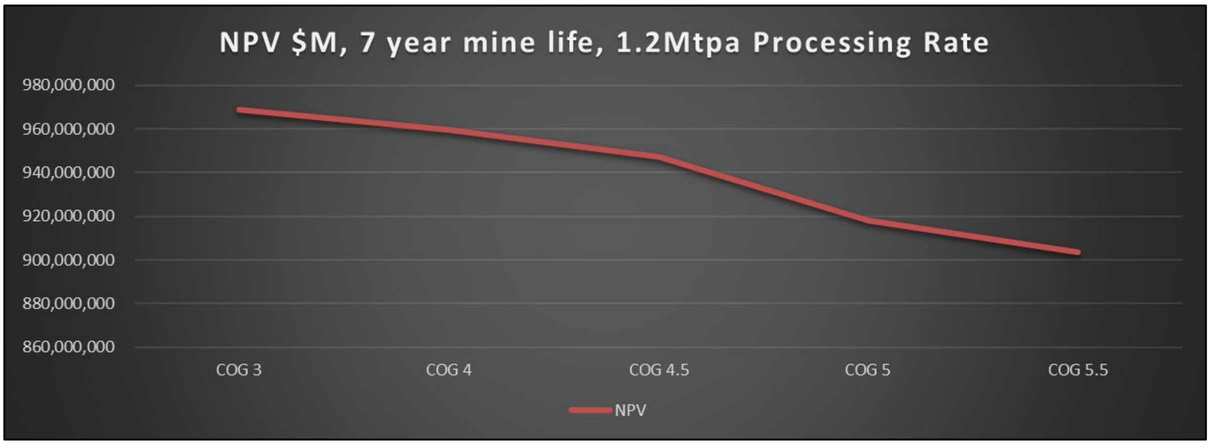


Figure 16.5.4 Total MSO vs Resource - Moz AuEq (y-axis) vs Cut-off Grade (x-axis)



A comparative net present value (NPV) analysis was carried out to assist in selection of the appropriate cut-off grade for the FS. 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 16.5.5 below.

Figure 16.5.5 Comparative NPVs at COG Range 3 – 5.5 g/t AuEq



16.5.3 Cut-Off Grade Selection

K92 selected the 3.0 g/t AuEq stope optimizations to carry forward for the final mine design and FS 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.

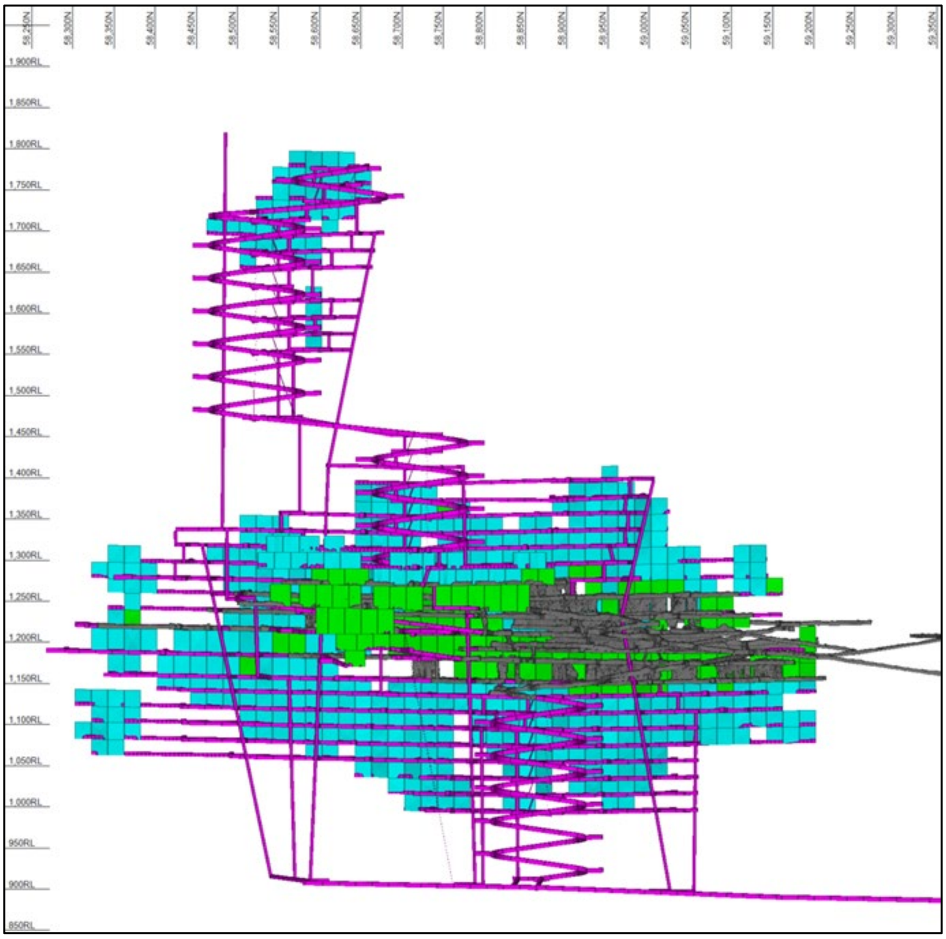
16.5.4 Stoping Methodology

The extraction methods that are proposed for the feasibility study are:

- Longitudinal LHS with paste fill
- Longitudinal Avoca and Modified Avoca mining

The spatial application of these mining methods is shown in Figure 16.5.6.

Figure 16.5.6 Stope Extraction Methods – Longitudinal LHS with Fill (blue), Avoca (green), As-builts in Grey



Stope Design Parameters

The mining method selection outcomes dictated the stope design parameters as per the MSO parameters described. Following the stope optimization 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.

Modifying Factors

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 optimization phase. An additional 0.5m of dilution was applied to stope shapes that were within 2.0m from the fault gouge zone. These dilution parameters were based on Entech's geotechnical study.

An additional 2.5% dilution from paste fill and 5% waste rock dilution for Avoca stoping methods were applied in the schedule. Paste fill and Avoca dilution are based on benchmark and site dilution data. The LOM average stope dilution is 20%.

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.

Drilling, Blasting and Slotting Methodology

The current drill design philosophy for Kainantu is based around 89 mm blast holes, although a 76mm blasthole could also be suitable for some stoping areas. Underground blasting with 2.0 - 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 16.5.4 outlines typical burden and spacing parameters that would be used for this hole size.

Table 16.5.4 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

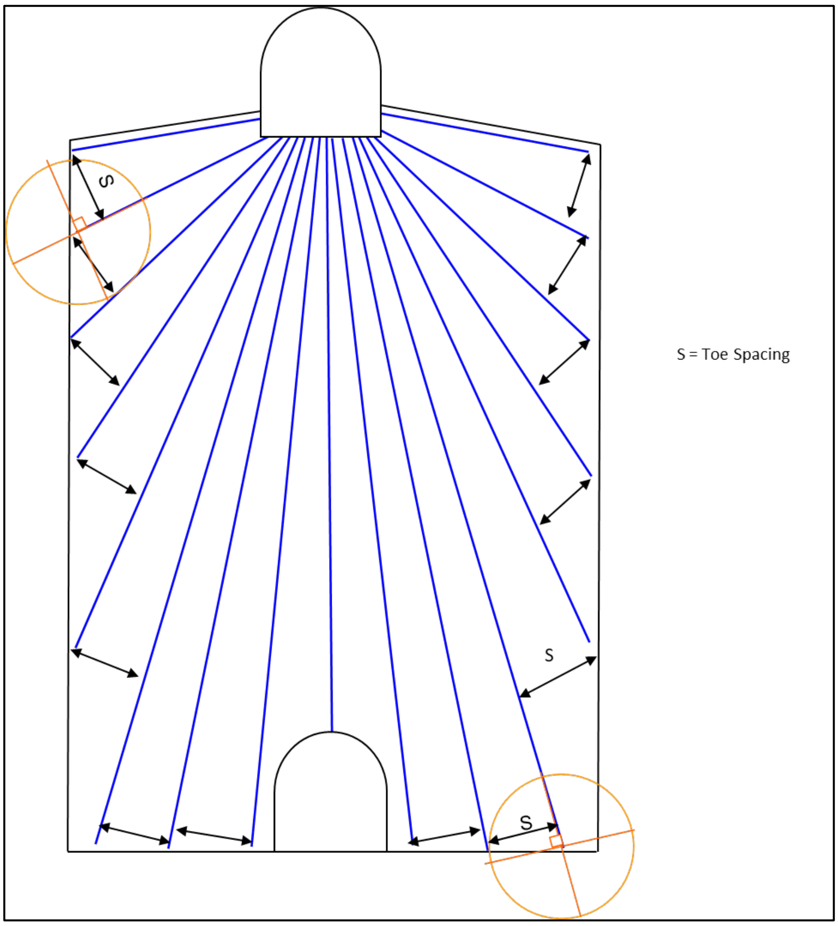
When completing ring design, the toe spacing is determined by the perpendicular distance between the toe of the shorter hole and the adjacent longer hole as per the example in Figure 16.5.7. Benchmark data shows spacing is typically linked to the burden as per the following formula:

S = (1.0 to 1.4) x B, where

S is the Spacing and

B is the Burden

Figure 16.5.7 Production Drill Spacing Example



The drill yield used for the FS is 5.5 t/dm which aligns with current site parameters and is appropriate for the average stope width.

Table 16.5.5 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 16.5.5 Charged Amount of Drill Metres by Stope Type (%)

Stoping Area	% of Drill Metres Charged
LHS paste fill/Avoca	80%

For the LHS stopes, 0.76 m diameter slots are planned to be developed using 'Rhino' (or equivalent) raises for blasting voids as illustrated in Figure 16.5.8. 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 16.5.9 shows the 'Rhino' (or equivalent) rig.

Figure 16.5.8 Example Slot Around 0.75 m Rhino Boxhole

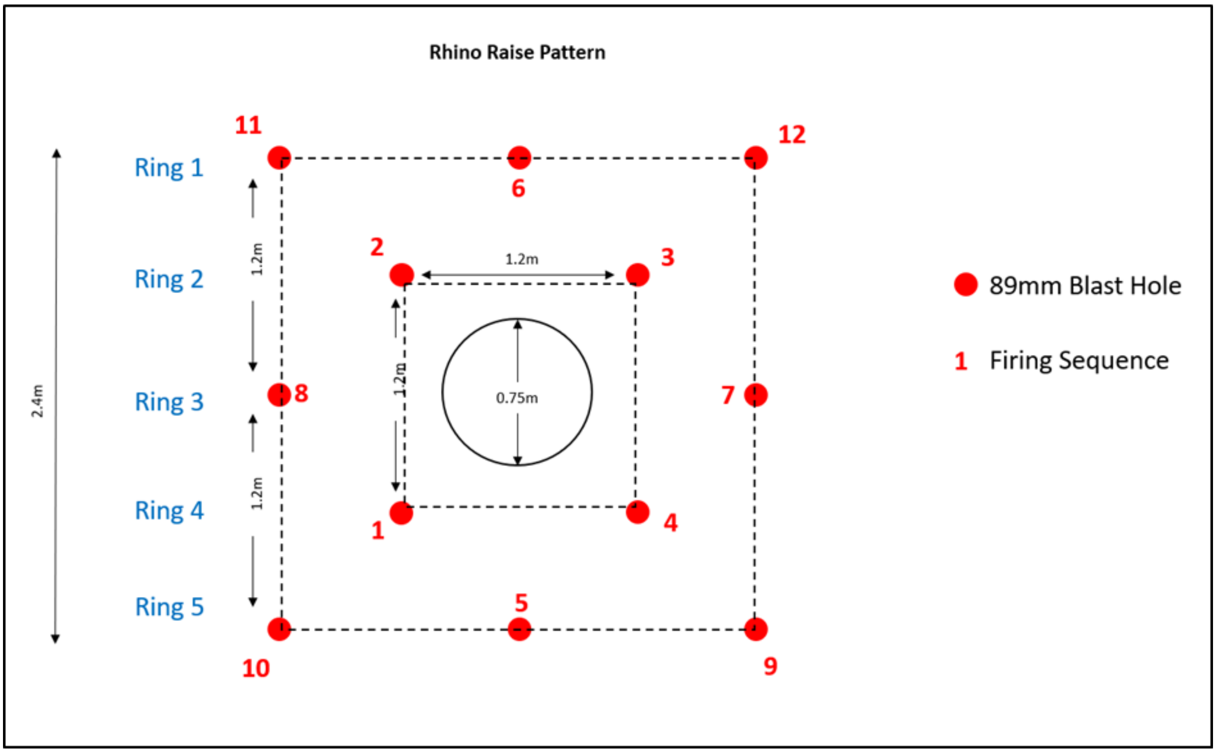
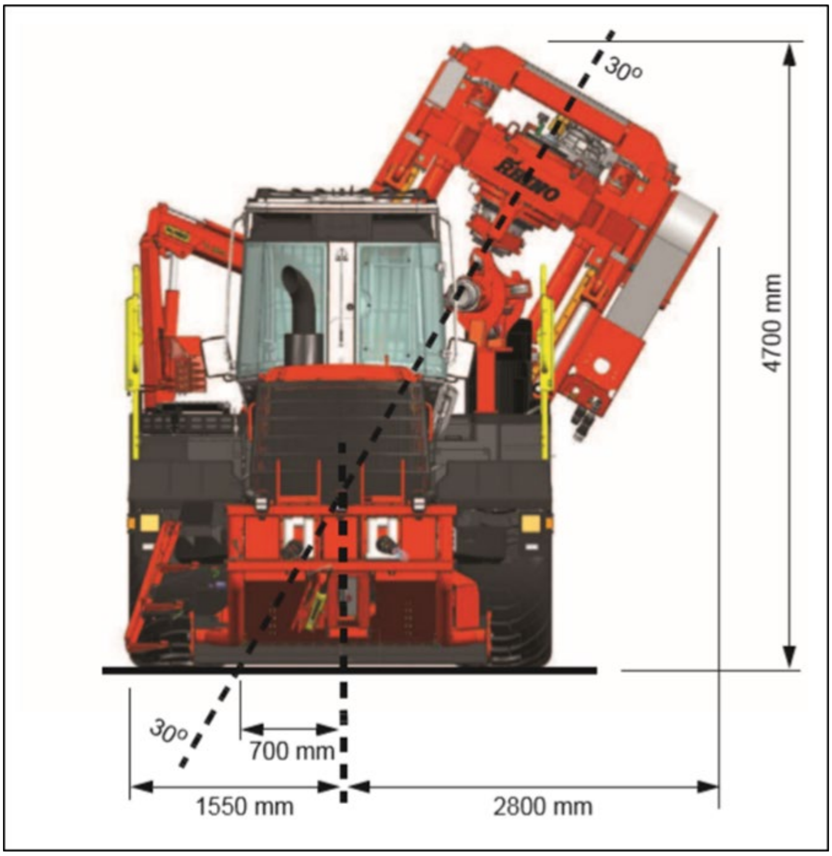


Figure 16.5.9 Rhino 100M Boxhole/Raisebore Rig

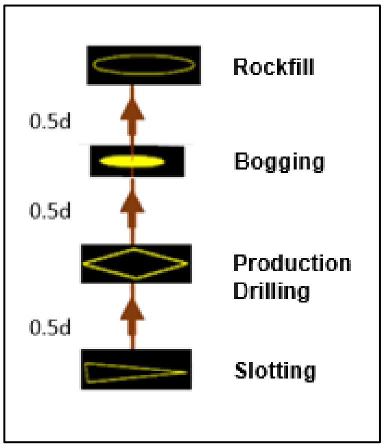


Avoca and Modified Avoca Stoping

Stope Cycle

The stoping cycle is depicted in Figure 16.5.10 and a stope cycle time in Table 16.5.6.

Figure 16.5.10 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 16.5.6 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	1200 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)			0.5 d
Backfill stope	3,340 m^3	500 m^3 / d	6.7 d
Total Stope Cycle			**20 d**

Longitudinal Sublevel Stoping with Paste Fill

Stope Cycle

The stope cycle activities, and stope cycle time for an average size LHS are shown in Figure 16.5.11 and Table 16.5.7. For a vertical paste exposure, the backfill cure activity is 14 days and 28 days for horizontal paste exposure.

Figure 16.5.11 Stoping Cycle By Activity as Linked in Deswik

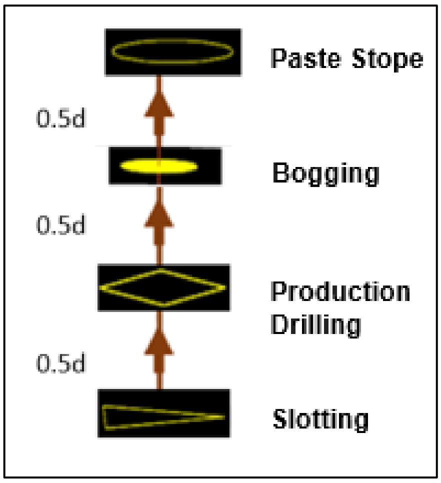


Table 16.5.7 Stope Cycle Activities for an Average Size LHS with Paste Fill Stope

Activity	Quantity	Task Rate	Duration
Boxhole drilling (Rhino)	15 m	10 m/d	1.5 d
Delay (Survey, engineering)			0.5 d
Production drilling	1350 m	250 m /d	5.4 d
Delay (Survey, engineering, blasting)			0.5 d
Loading	6,800 t	1000 t / d	6.8 d
Delay (Survey, fill walls, fill reticulation)			0.5d
Backfill stope	3,000 m^3	500 / d	6 d
Backfill Cure (Adjacent/Underhand)			14/28 d
Total stope Cycle Adjacent/Underhand			**35/49 d**

Stoping Sequence

The typical stoping and linking sequence for underhand longitudinal LHS with paste fill is shown in Figure 16.5.12, with overhand sequence in Figure 16.5.13.

Figure 16.5.12 Example of a Longitudinal LHS with Paste Fill Underhand Mining Sequence (Sequence will Vary in Areas)

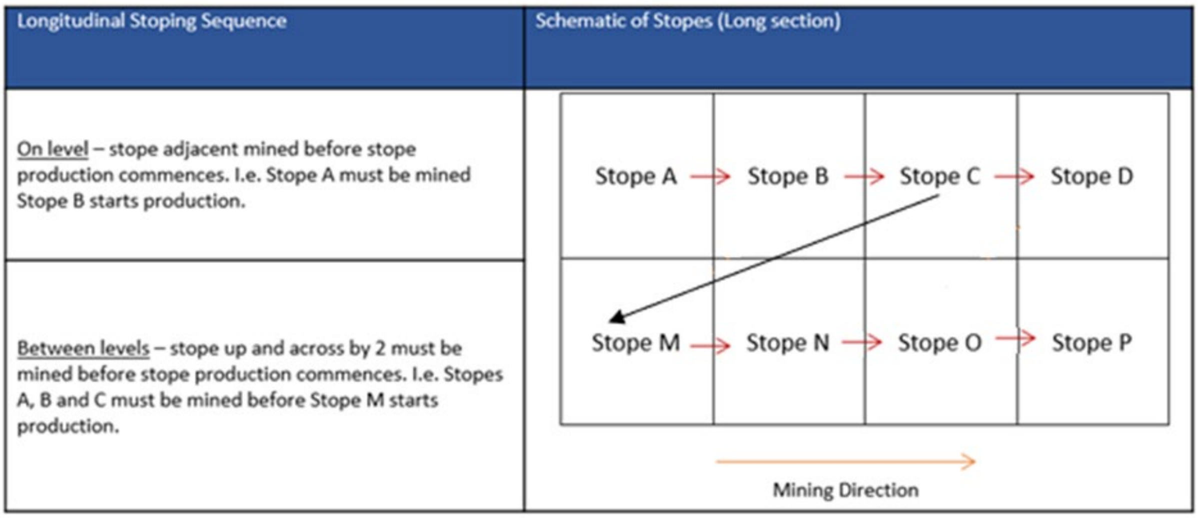
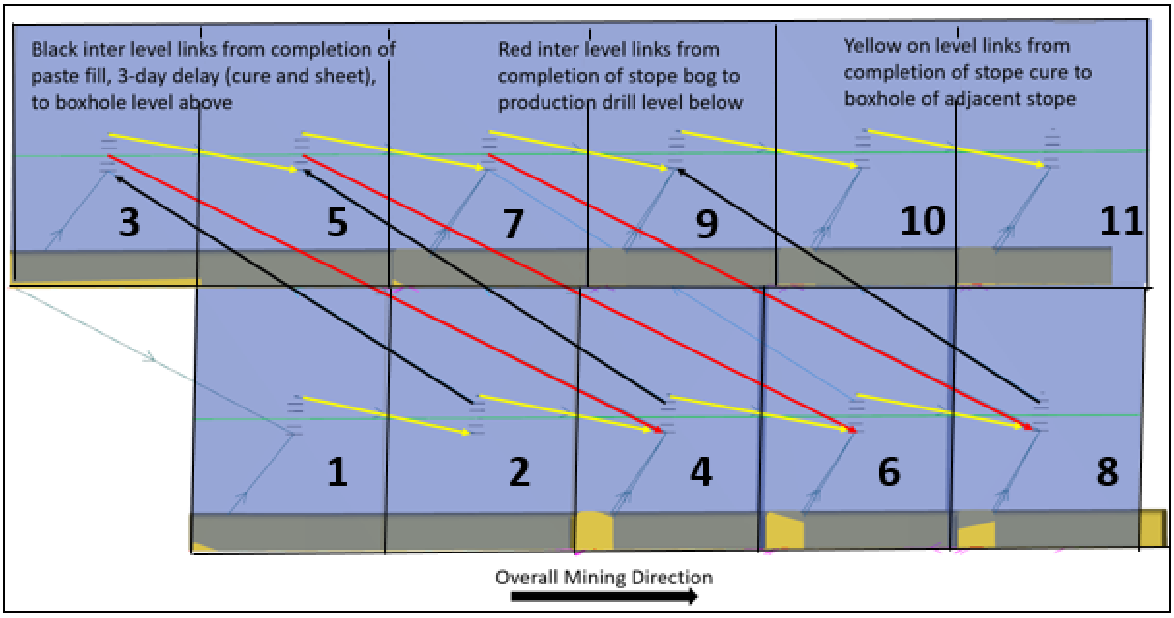


Figure 16.5.13 Example of a Longitudinal LHS with Paste Fill Overhand Mining Sequence (Sequence will Vary in Areas)



16.5.5 Backfill

Paste fill has been scheduled at approximately 90 cubic metres of paste per hour for a total of 17 hours per day. This equates to a filling rate of 1,500 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 16.6.1 and Figure 16.6.2.

Figure 16.6.1 Long Section View Looking West of the Kainantu Underground Development Layout

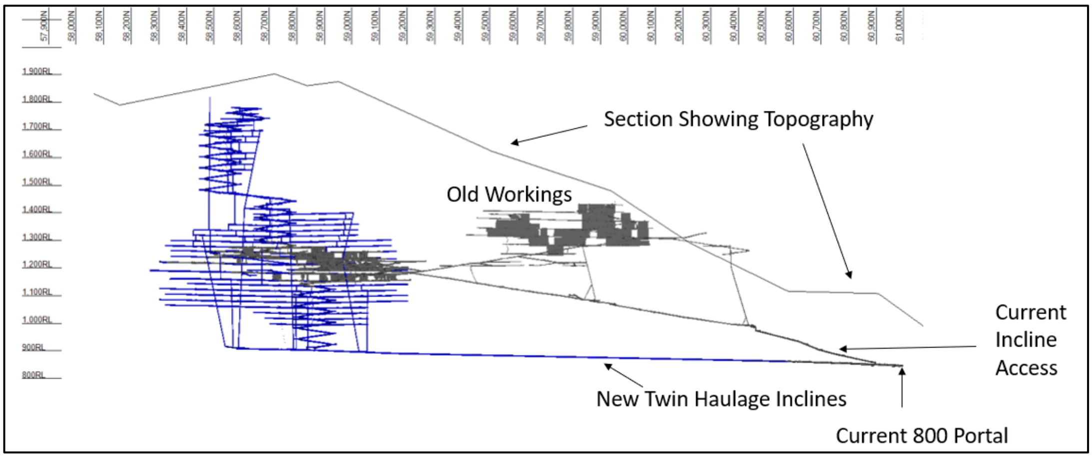
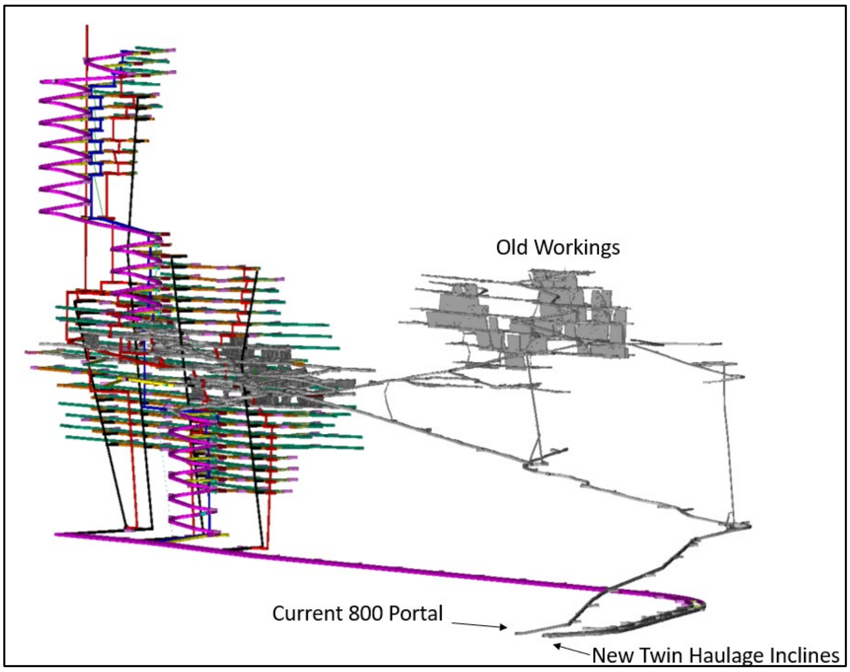


Figure 16.6.2 An Isometric View Looking Southwest of the Kainantu Underground Development Layout



Entech has completed development design from the current as built positions and following the current K92 site design standards. Decline gradients (-1:7), decline standoffs to the orebody (~70 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 'Rhino' rig (or equivalent). 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

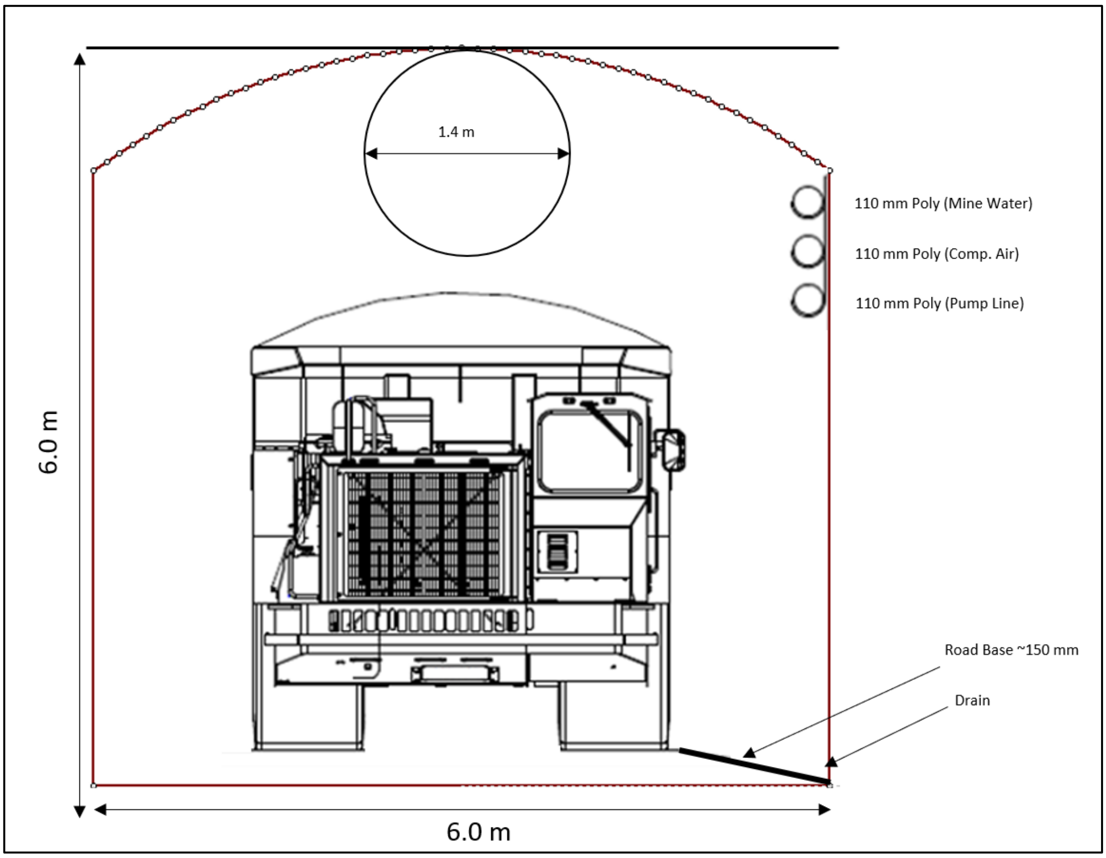
The mine is currently accessed via the 800 Portal which is a 4.5 mW x 4.5 mH access accommodating 45 t payload underground mining trucks. New twin haulage incline drives are being excavated, one at 6.0 mW x 6.0 mH drive size, and the other at 5.0 mW x 5.0 mH. These new drive sizes could allow future 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 which accesses the ore pass network, to allow for future trucking size increase, as per Figure 16.6.3. The other areas of the mine utilize a 5.0 mW x 5.5 mH decline drive size, with the current 800 portal and legacy incline at 4.5 mW x 4.5mH suitable for the current 45t truck size.

The decline-orebody stand-off distance to stopes is a minimum of 40 m to the Kora lodes, and 20-25 m to the Judd lode. This distance will minimize 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.

Figure 16.6.3 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.

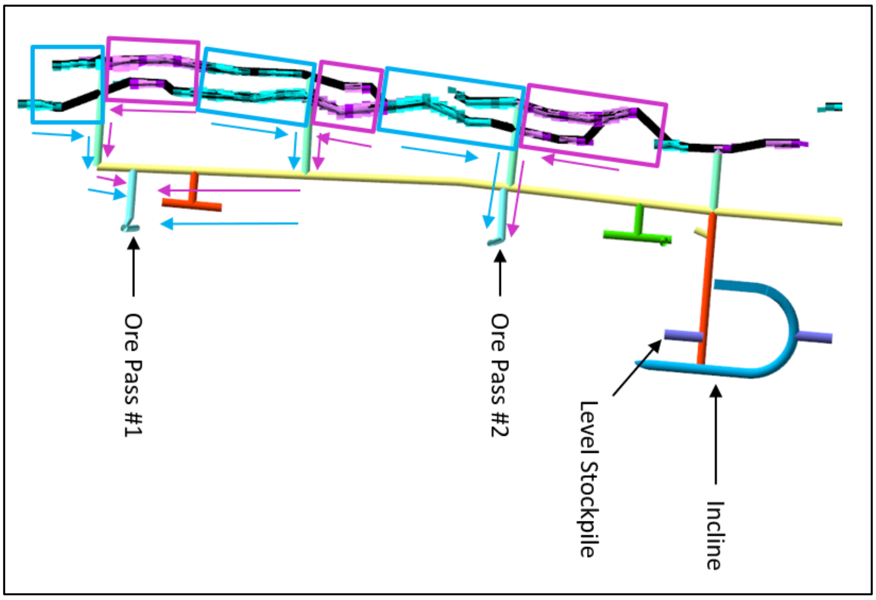
A 'racecourse' style decline configuration with a minimum radius of 25 m has been designed to provide optimal access to levels every 20 vertical metres whilst also allowing trucks to operate at a productive speed. Stockpiles are typically designed every 140 m along the decline, which is sufficient for high-speed development and allows for numerous locations to pass equipment and position infrastructure such as stockpiles, electrical substations and refuge chambers.

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.

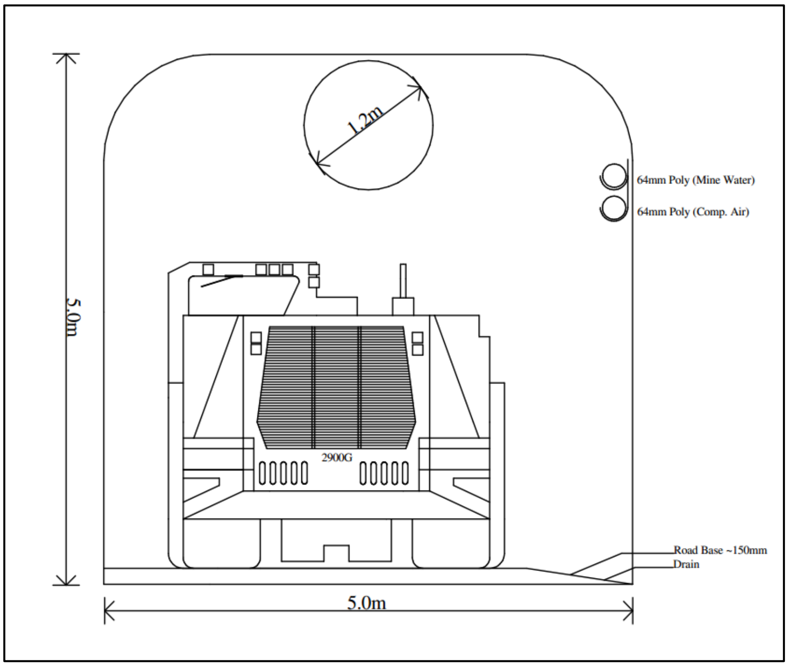
A square profile stockpile has been designed on each level access at 5.0 mW x 5.5 mH and 25 m in length. The level stockpile will be used for the stockpiling of development and stope ore that is produced from the level until it can be loaded into a truck or ore pass. A typical level layout is shown in Figure 16.6.4.

Figure 16.6.4 An Example of the 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 16.6.5.

Figure 16.6.5 An Example of the Ore Drive Profile



16.6.4 Vertical Development

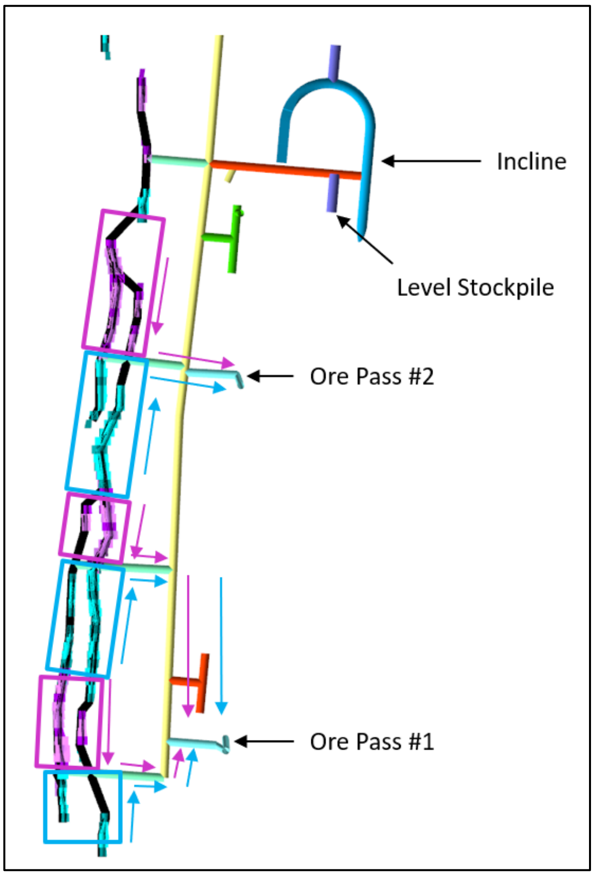
Vertical development is separated into capital and operating development. Capital development consists of ore passes, escapeway rises, ventilation return rises, and fresh air intakes. Escapeway rises are planned at 1.1 m diameter to be excavated with the 'Rhino' (or equivalent) boxhole rig. Ventilation 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 0.75m diameter boxhole rise excavated with the 'Rhino' (or equivalent).

16.6.5 Materials Handling

Two material handling methods were considered. The first option considered all material to be hauled via trucking from the extraction level where the material was blasted. This material would be loaded onto trucks by a bogger utilizing the nearest stockpile. 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 below.

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 16.6.6.

Figure 16.6.6 Plan View Typical Level Layout



The following points resulted in the selection of utilising an ore pass network versus trucking only.

- 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 FS inventory can be found detailed in the table below, with an annual material movement breakdown in Table 16.7.1 below.

Table 16.7.1 Material Movement Breakdown

		Total	2022	2023	2024	2025	2026	2027	2028
Waste	*t*	3,157,900	578,985	761,096	650,236	515,507	533,658	112,630	5,787
Dev Ore	*t*	893,770	124,476	172,491	134,339	168,910	133,182	153,148	7,224
Stope Ore	*t*	5,259,042	311,031	347,267	767,254	1,029,420	1,061,569	1,039,051	703,450
Ore	*t*	6,152,812	435,507	519,757	901,593	1,198,330	1,194,751	1,192,199	710,674
Ore Tonnes									
Measured	*t*	2,464,778	306,161	297,281	473,053	540,901	438,485	315,730	93,168
Indicated	*t*	3,688,034	129,346	222,477	428,540	657,429	756,267	876,469	617,506
Total	***t***	**6,152,812**	**435,507**	**519,757**	**901,593**	**1,198,330**	**1,194,751**	**1,192,199**	**710,674**
Au Grade	*g/t*	6.65	9.07	8.55	7.08	6.33	6.77	5.49	5.53
Cu Grade	*%*	0.88%	0.56%	0.65%	0.75%	0.92%	1.02%	0.88%	1.13
Ag Grade	*g/t*	17.53	11.47	12.94	14.40	17.99	20.03	17.84	23.04
Metal									
Au	*Moz*	1.32	0.13	0.14	0.21	0.24	0.26	0.21	0.13
Cu	*kt*	54.39	2.42	3.35	6.79	11.04	12.21	10.52	8.05
Ag	*Moz*	3.47	0.16	0.22	0.42	0.69	0.77	0.68	0.53

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 (1.2 Mtpa). The mine life is 7 years with a 3-year production ramp up (inclusive of 2022), sustained production of 1.2 Mtpa for 3 years, and final year production of 0.7 Mtpa.

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 16.8.7.

The major constraints on the underground scheduling were as follows:

- Ensure a smooth ramp-up to steady ore production.
- Minimize variations in development rates and production to avoid additional project costs due to under-utilization 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 (m RL).
- 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 summarized into individual activities that provided the appropriate detail for FS 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 16.8.1 below.

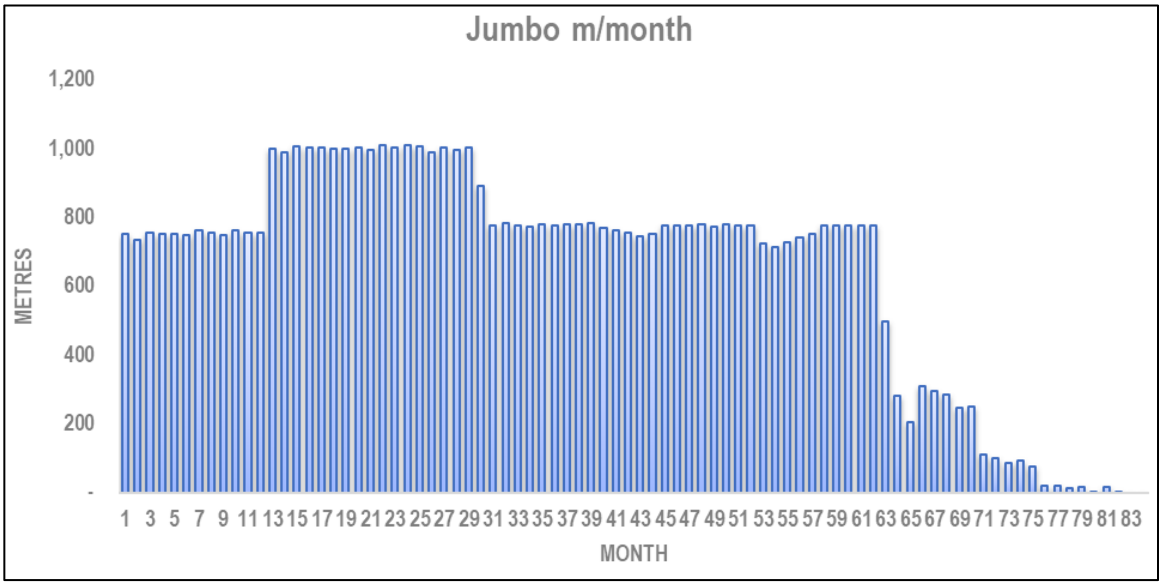
Table 16.8.1 Key Physicals Per Annum

		Total/Av	2022	2023	2024	2025	2026	2027	2028
Lateral Development									
Decline	m	9,918	3,247	2,333	1,549	1,273	1,516	-	-
Other Capital	m	21,837	2,823	5,721	5,455	3,672	3,366	780	19
Ore Drive	m	16,549	2,087	2,854	2,233	3,283	2,982	2,947	162
Other Operating	m	6,123	872	1,120	1,319	1,009	1,229	406	169
Total	**m**	**54,427**	**9,029**	**12,028**	**10,556**	**9,237**	**9,093**	**4,134**	**350**
Vertical Development									
Return Air Rise 5.0 m	m	479	-	479	-	-	-	-	-
Return Air Rise 4.0m	m	1,542	95	639	487	141	99	81	-
Fresh Air Rise 5.0m	m	704	46	174	242	135	93	15	-
Ore Pass 3.0m		2,217	-	207	1,024	705	282	-	-
Slot Rise 0.76m		11,415	705	735	1,980	2,113	2,057	2,486	1,339
Production Drilling									
Upholes 89mm	m	366,186	15,598	27,487	46,178	73,551	87,416	71,611	44,345
Downholes 89mm	m	532,671	40,204	34,769	106,792	93,334	96,170	103,864	57,537
Haulage									
Underground	tkm	26,834,560	2,302,074	3,374,294	4,207,640	5,106,017	5,653,575	3,881,152	2,309,809
Ore Profile									
Ore Tonnes	t	6,152,812	435,507	519,757	901,593	1,198,330	1,194,751	1,192,199	710,674
Au Grade	g/t	6.65	109	103	85	76	81	66	78
Au Ounces	oz	1,316,007	126,929	142,862	205,318	244,031	259,967	210,457	126,442
Cu Grade	%	0.88	7	8	9	11	12	11	13
Cu Tonnes	t	54,391	2,423	3,354	6,785	11,044	12,211	10,525	8,050
Ag Grade	g/t	17.53	137	155	173	215	240	214	252
Ag Ounces	oz	3,466,855	160,559	216,167	417,297	692,991	769,545	683,874	526,422

16.8.1 Jumbo Development

The life of mine horizontal development schedule is shown in Figure 16.8.1.

Figure 16.8.1 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.

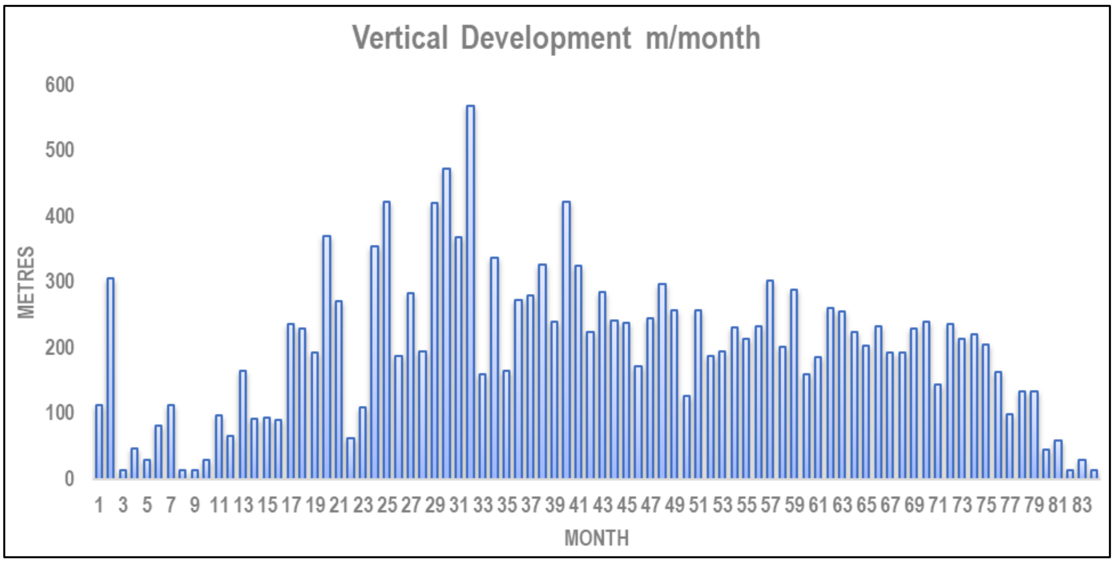
Task priorities followed were:

- 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 16.8.2.

Figure 16.8.2 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 16.8.3, and ore production profile in Figure 16.8.4.

Figure 16.8.3 Bogged Tonnes by Month

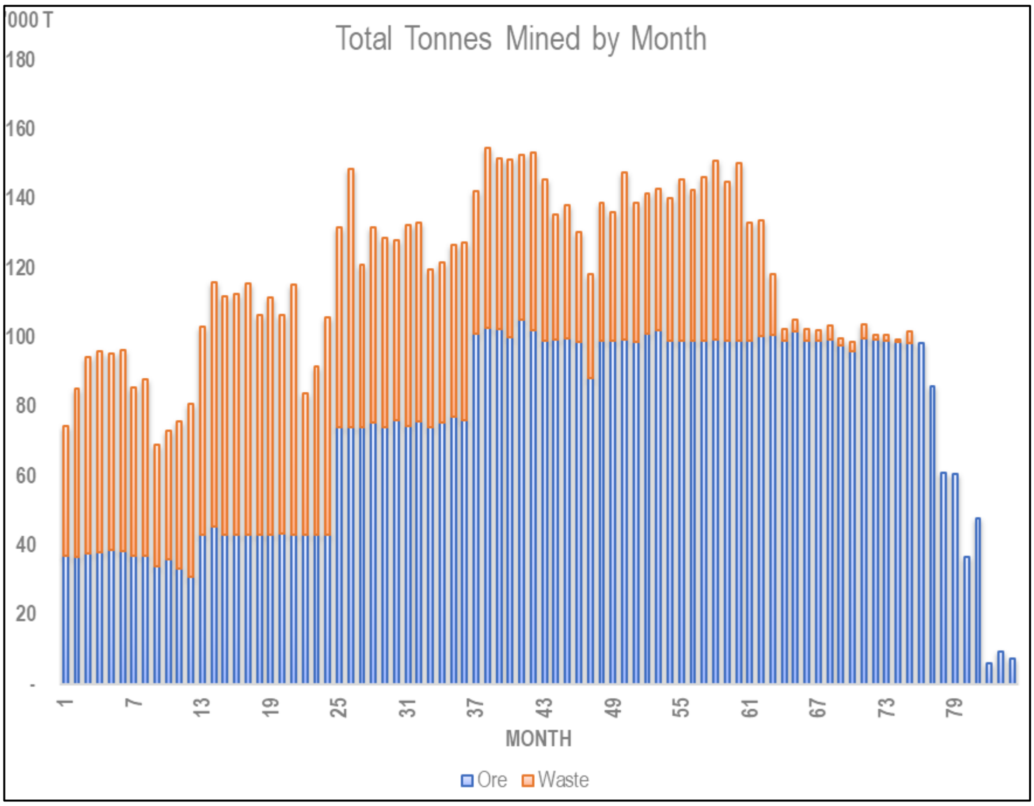
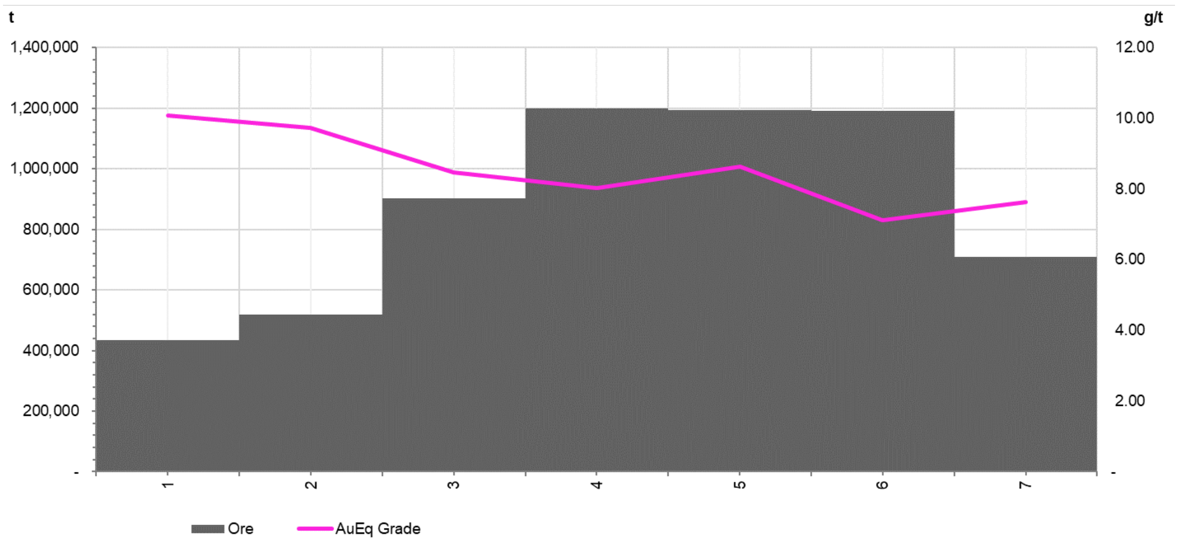


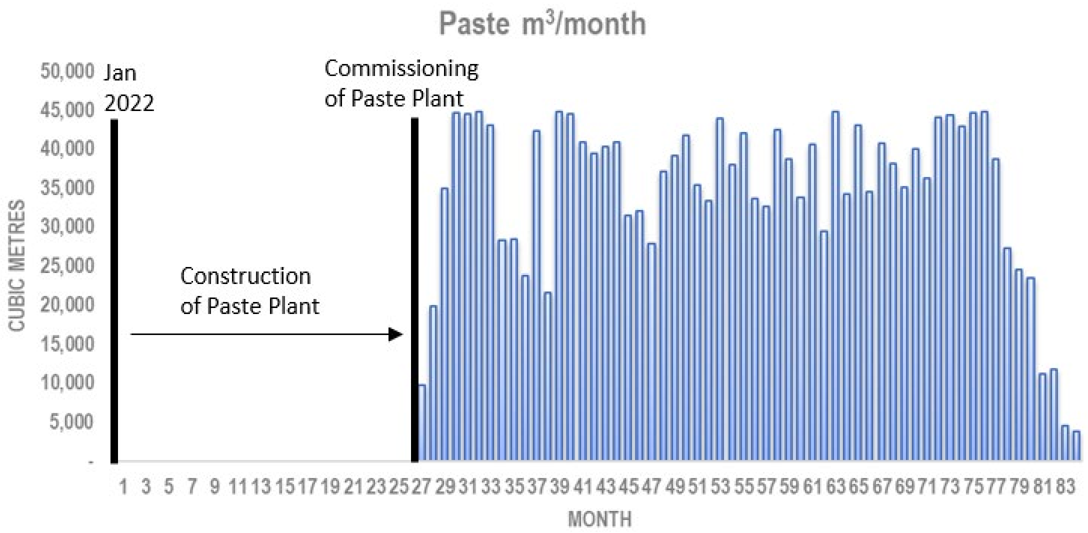
Figure 16.8.4 Ore Tonnes and AuEq Grade per Annum



16.8.5 Backfilling

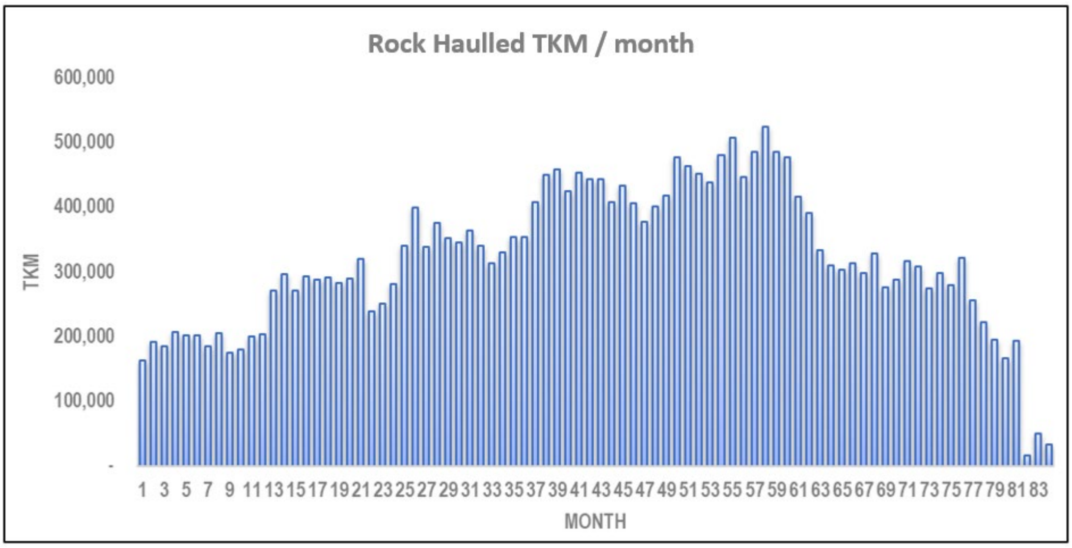
Backfilling of stopes is carried out with paste fill. Paste filling of these stopes has been scheduled at a rate of 1,500 m^3 per day and commences upon the completion of stope loading. Figure 16.8.5 below shows the life of mine paste filling schedule.

Figure 16.8.5 Paste Fill by Month



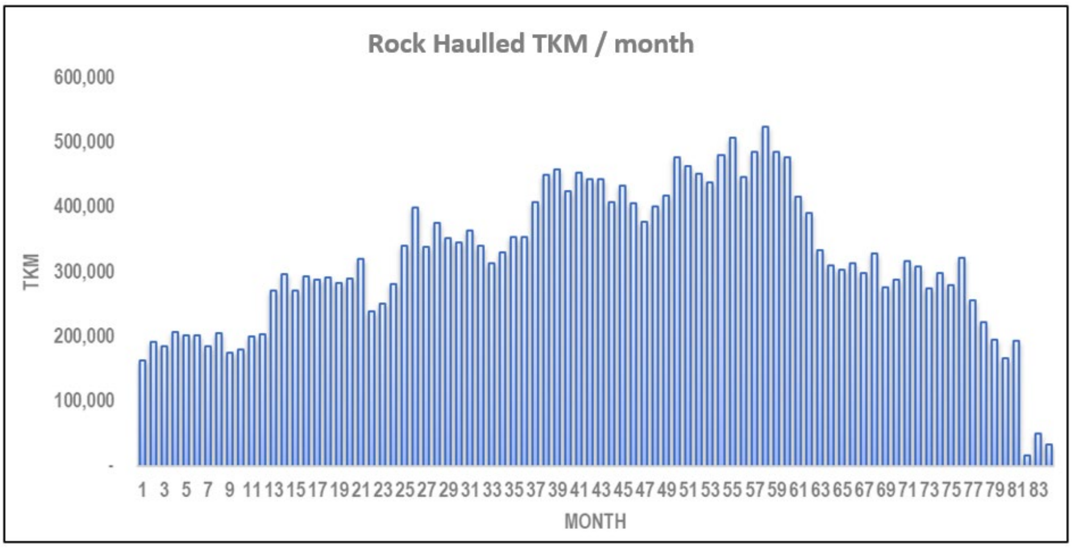
16.8.6 Haulage

The haulage requirement from the mine, inclusive of all ore and waste material, is illustrated in Figure 16.8.6 Rock Hauled TKM by Month



and is stated on a tkm basis. Surface haulage is captured by a separate fleet and has been costed independently of the underground haulage cost and physicals estimates.

Figure 16.8.6 Rock Hauled 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 K92 based on site productivities. Entech have reviewed the specified fleet requirements to ensure they correlate with appropriate productivity assumptions.

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 are based on K92's site productivities (150 m/mo to 175 m/mo per jumbo drill) and were cross-checked by Entech for appropriateness. The schedule has a maximum total of 1,000 m/mo jumbo development at any point in the schedule.

Vertical Development

A development rate of 3 m / day has been applied to all conventional raisebore vertical development for hole diameters between 3.0m – 5.0m. Boxhole activities completed by the Rhino machine are assumed to occur at a rate of 10 m/day for hole diameters <=1.1m. These rates include rig up and rig down, pilot hole drilling, hole reaming and ladderway installation (where relevant).

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 5.5t 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.

Stoping Cycle

Based on Sandvik LH517 loader capabilities a loading rate of 1000 t/d was applied in the schedule. This incorporates a time allowance for stope charging, shift change meetings, meal breaks, breakdowns, maintenance, services installation, and geology/survey control delays.

Backfill

Paste fill has been scheduled at approximately 90 cubic metres of paste per hour for a total of 17 hours per day. This equates to a filling rate of 1,500 cubic metres of paste per day.

Materials Handling

All the ore and waste is currently planned to be hauled using conventional 45-tonne underground haulage trucks to the portal transfer point. At peak production, a total of seven trucks will be required. Productivity for a Sandvik TH545 underground truck in this scenario would average 75,000 tkm/mo (tonne-kilometres per month) per unit. The increased twin haulage drive size allows for future increase to the truck size.

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 FS. 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 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.

Not included in the analysis is the potential for mine expansion.

16.9.2 Model Design Parameters & 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 16.9.1.

Table 16.9.1 Ventilation Design Parameters

	Parameter Description	Design Parameter	Remarks
1.	Recommended air velocity in declines to prevent dust generation.	< 6.0 m/s.	Industry best practice for preventing dust downwind of moving equipment.
2.	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.
3.	Intake shaft velocity.	< 20 m/s (unequipped).	Guide for maintaining safe air velocities in underground workings.
4.	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.
5.	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.
6.	Heat modelling parameters set to the 90^{th} 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
7.	Use of the K92 mine's elevation above sea level for setting barometric pressure.	Elevation set to 800 mRL for portal.	Elevation estimated for the main haulage portals. Air density is set to this elevation.
8.	WB design limit before applying safety measures.	28°C.	Industry best practice for the prevention of heat stress in workers.
9.	WB stop-work upper limit.	32°C.	Industry best practice for the prevention of heat stress in workers.
10.	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.
11.	K92 airway sizes for modelling purposes.	Main incline = 26 m^2 (arched airway). Dual haulage drives = 34 m2 & 23 m2 (arched airways). Return adits = 23 m^2 (arched airway). Return LHR = Ø4.0 m or 12.6 m^2 (RB airway). Return shaft to surface = Ø5.0 m or 19.6 m2 (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).	
12.	Exposure standard for diesel particulates.	< 0.1 mg/m^3 TWA (measured as sub-micron elemental carbon).	Set by Work Safe Australia.
13.	Exposure standard for respirable silica dust.	< 0.05 mg/m^3 TWA.	Set by Work Safe Australia.

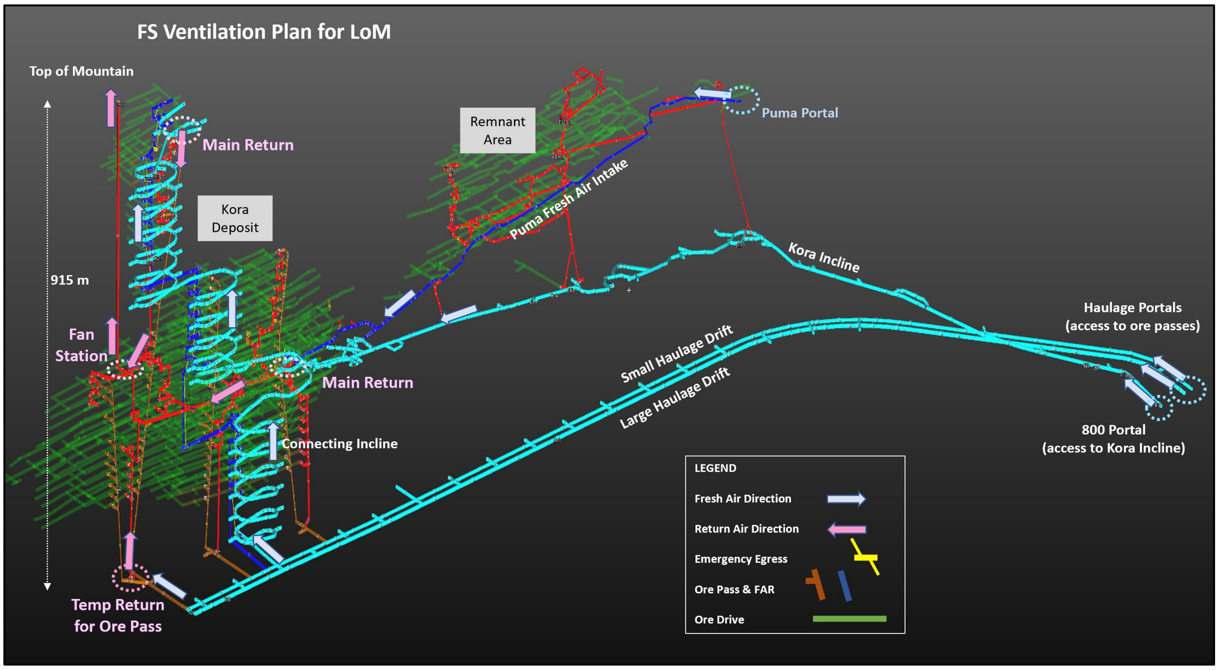
- Design assumptions affecting modelling outputs.
- No more than five production levels operating simultaneously, requiring primary ventilation to support a loader along each footwall drive to each ore pass connection.

- 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 minimized 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 FS plan to ventilate K92 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 single exhaust air raise daylighting at the top of the mountain. The Puma portal will be repurposed as an intake, which is currently the main exhaust for the existing workings. Figure 16.9.1 illustrates the plan.

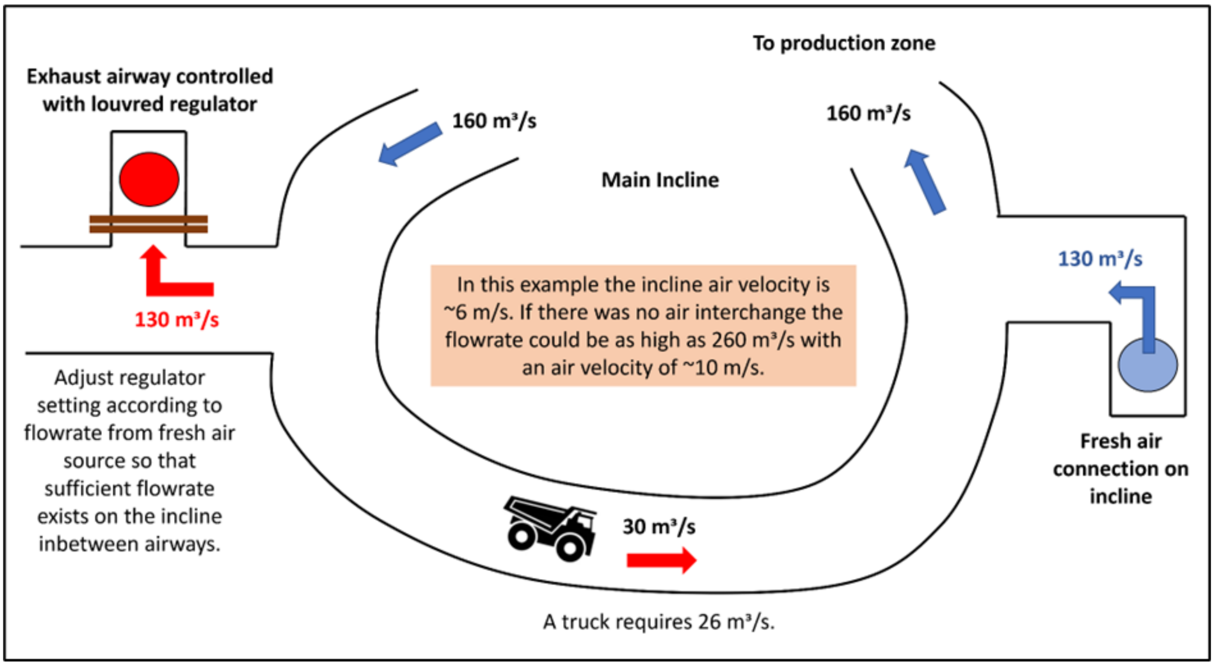
Figure 16.9.1 LOM Ventilation Plan Profile Facing Northwest



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 a single underground fan station, located at the base of a 480 m raise to surface.

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 16.9.2 below illustrates the concept.

Figure 16.9.2 How an Air Interchange Works



The development, that 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.

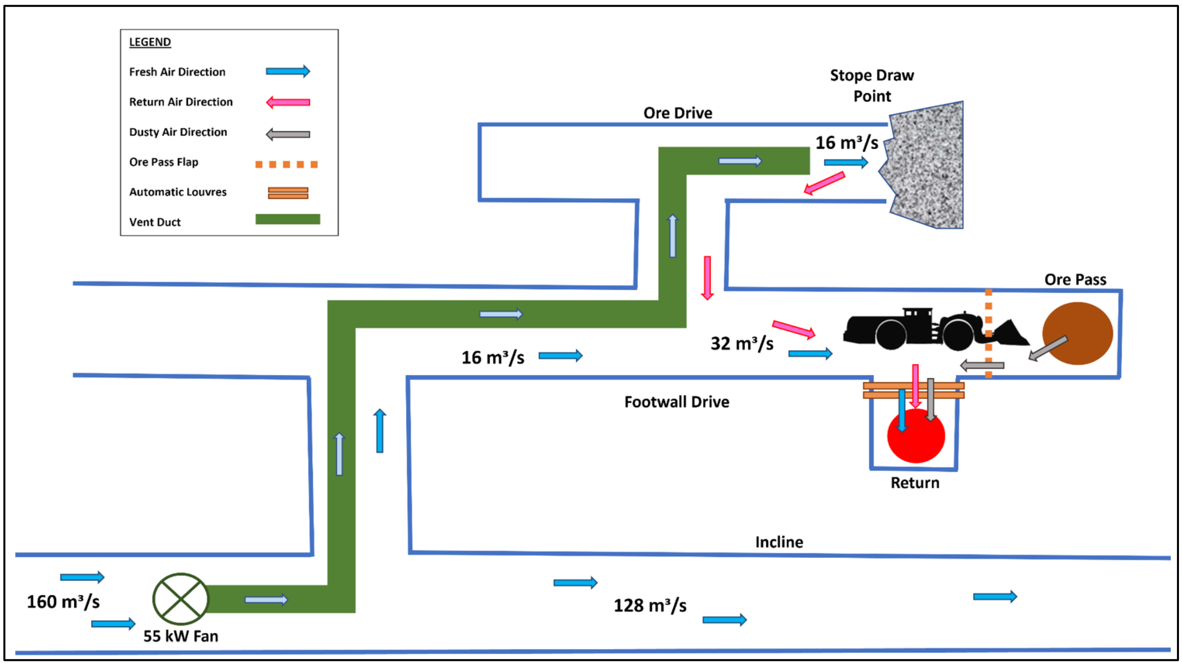
16.9.4 Secondary Ventilation

Production level design will combine secondary air with primary air depending on mining activity.

Development headings will be forced ventilated with 55 kW fans 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 at the end of the drive where ore will be tipped into ore pass. Any dust generated from the ore pass will directly report to the return without polluting the incline. See Figure 16.9.3 for a typical ventilation plan.

Figure 16.9.3 Example of a Production Level Ventilation Plan



16.9.5 Ventilation Analysis

Flowrate Definition by Diesel Exhaust Dilution

Table 16.9.2 contains a list of the mobile equipment intended for use by K92 and shows the peak number of vehicles anticipated with the subsequent flowrate total. According to WA regulations, utilization 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 16.9.2 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	7	180
Loader	Sandvik LH517	310	16	5	78
Charge-up	Getman	120	6	3	18
Development Drill **	Sandvik DD421-60C	110	6	6	0
Production Drill **	Sandvik DL431-7C	119	6	2	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	Normet Spraymec SF 050 D	90	5	2	9
Agitator	Normet LF 700 transmixer	170	9	2	17
IT	Volvo L120H	203	10	5	51
LV	Toyota Landcruiser	151	8	9	68
Total Flowrate for Diesel (m^3/s)					**433**
Leakage @ 15%					65
Total Flowrate Including Leakage (m^3/s)					**498**
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**

***Flowrates calculated for diesel equipment at 0.05 m^3/s per kW of rated engine power.**

****Vehicles 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 peak flowrate of 430 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 500 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 16.9.1) for velocity, will allow a maximum of 160 m^3/s in a travel way before velocities exceed 6 m/s. K92 will need to manage the peak number of diesel units between multiple production zones in meeting the total flowrate requirement. Table 16.9.3 offers an example of a flowrate calculation for a production zone according to the incline velocity limit.

Table 16.9.3 Peak Flowrate Calculation According to Expected Mobile Fleet for a Production Area with Single Incline.

Per Production Zone with 160 m³/s Limit.	Assumed Model	Engine Power Rating (kW)	Flowrate Requirement (m^3/s) *	Count	Total Flowrate (m^3/s)
Development Truck	Sandvik TH551	515	26	2	52
Loader	Sandvik LH517i	310	16	2	31
Charge-up	Normet MC 605D	120	6	1	6
Development Drill **	Sandvik DD421-60C	110	6	0	0
Production Drill **	Sandvik DL432i-9C	119	6	0	0
Cable Bolter **	Sandvik DS422i	119	6	0	0
Grader	John Deer 670G	134	7	1	7
Water Cart	Volvo A30D	242	12	0	0
Fibrecrete Sprayer	Normet Spraymec SF 050 D	90	5	0	0
Agitator	Normet LF 700 transmixer	170	9	0	0
IT	Liugong	203	10	1	10
LV	Toyota Landcruiser	151	8	4	30
Total Flowrate for Diesel (m^3/s)					**136**
Leakage @ 15%					20
Total Flowrate Including Leakage (m^3/s)					**156**
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 m³/s per kW of rated engine power.**

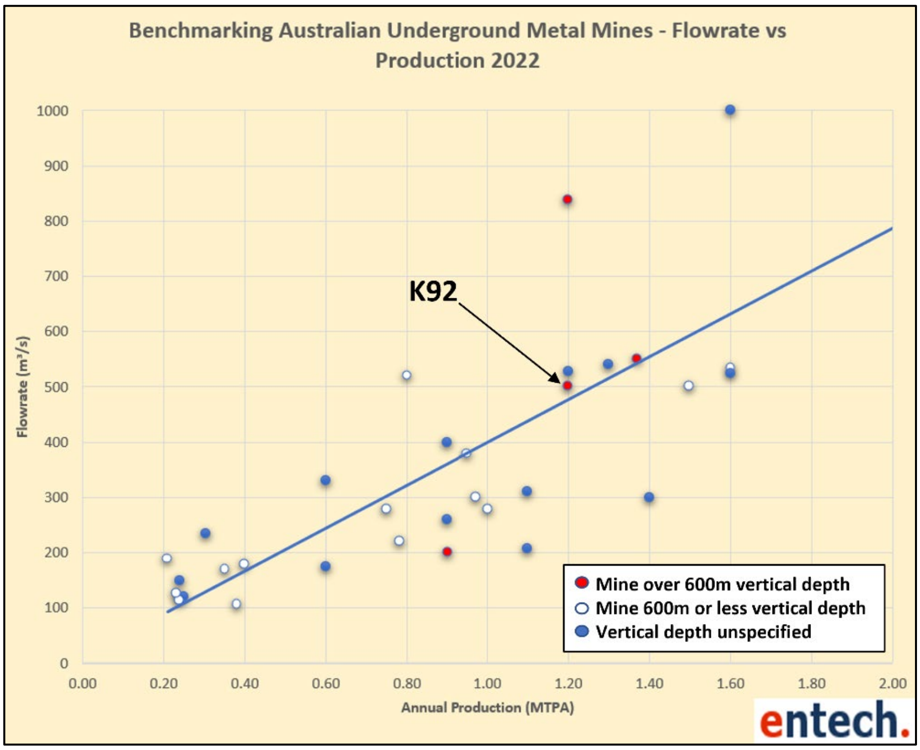
****Vehicle primarily operating under electric power. Flowrate allocation is given when tramming under diesel power, however, unit is omitted in the total count.**

*****Minimum flowrate for activities relying on secondary air. The combination of two fans refers to activities in parallel.**

The truck count in Table 16.9.3 relates to mainly development haulage as most production trucking is isolated to the haulage declines at the base of the deposit.

Benchmarking indicates that K92, with an annual production rate of 1.2 Mt, is representational of other hard rock mines with equivalent flowrate (see Figure 16.9.4).

Figure 16.9.4 Benchmarking Kainantu with Australian Metal Mines



Heat Modelling

A high-level heat analysis was carried out using the following heat settings as per Table 16.9.4.

Table 16.9.4 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.

16.9.6 Ventilation Infrastructure

Airways

The exhaust raise bore to surface, at Ø5.0 m, is sufficient to accommodate the peak flowrate requirement, although it is at the limit. Reducing the diameter of the RB should not be considered without the addition of a second shaft.

Table 16.9.5 provides a comparison of CAPEX versus OPEX to show how airway size can optimize power costs over the LOM. System pressure reflects the LOM resistance and includes representational electrical fan power in overcoming this resistance.

Table 16.9.5 CAPEX vs OPEX Using Shaft Size

Airway	K92 (7 Year mine Life)			
Category	**Raise Bore Shaft**			
Diameter (m)	4.0	4.5	5.0	5.5
Drill Cost per Metre ($/m)	6000	6500	7000	7500
Cross-sectional Area (m^2)	12.6	15.9	19.6	23.8
Shaft Length (m)	485	485	485	485
Airflow Quantity Peak Production (m^3/s)	500	500	500	500
Velocity at Peak Production (m/s)	39.6	31.5	25.6	21.2
System Pressure (kPa)	5.4	3.8	3.0	2.5
Power Input Peak Production (kW)	3,580	2,510	1,950	1,630
Estimated RB Cost (AUD)	$2,910,000	$3,152,500	$3,395,000	$3,637,500
Annual Power Cost ($0.17/kWh)	$4,704,120	$3,298,140	$2,562,300	$2,141,820
Total Owing Cost (TOC) *	**$27,401,530**	**$20,323,936**	**$16,735,359**	**$14,788,672**

*Seven-year mine life @8% compound interest

A surface shaft of Ø5.5 m offers a 13% saving on the proposed Ø5.0 m across the seven-year mine life, however, when comparing the Ø5.0 m shaft to the smaller sizes the optimization comparison is considerably higher, with 21% and 35% saving versus Ø4.5 m and Ø4.0 m respectively.

Ventilation Milestones

The above table comparison is a guide based on factors derived from a LOM model. Major ventilation milestones will occur over the first five years of mine life, in which time system resistance and power costs will change (see Figure 16.9.8). The milestones and the infrastructure changes leading up to the LOM stage are as follows.

Current till Month 17 of Schedule

- The initial stage leads to the commissioning of the surface RAR and fan installation.
  - Production is on the Kora incline

- Primary flow is limited to the Puma exhaust fans @ 80 m^3/s
- Development is limited to the size of secondary fans
- Kora incline development is limited to 50 m^3/s with a single 110 kW fan
- The two haulage drifts, at the base of the deposit, are limited to 100 m^3/s with two 110 kW fans in parallel.

Month 18 – 20 of Schedule

- Commissioning of RAR.
  - Production continues in Kora incline
  - Primary flow changes from Puma exhaust to the surface RAR @ 350 m^3/s
  - Puma fan and its booster fan are removed from their airways
  - Puma becomes an intake
  - By this stage primary flow exists in the haulage drifts with a circuit created by two 110 kW fans plumbed into a bulkhead at one of the portals @ 100 m^3/s
  - Development of spiral incline and decline from the mid-point where the Kora incline and Puma intake converge.

Month 21 – 25 of Schedule

- RAR extension to haulage drifts, with surface RAR now @ 450 m^3/s.
  - Development continues in the incline and decline from the mid-point.

Month 26 – 61 of Schedule

- Internal FAR links haulage drifts with the rest of the circuit as a third fresh air source.
  - Primary flow drops to 400 m^3/s
  - Production extends beyond the old workings and into the newly developed areas
  - Development extends into the upper most section of the deposit and continues downward toward the haulage drifts.

Month 62 – 83 of Schedule

- The Kora decline breaks through and connects the haulage paths creating another access point into the mine from surface.
    - Primary flow eventually drops to 280 m^3/s as production starts to wind up.

Primary Fans

Entech used the following system duties in Table 16.9.6 to select suitable primary fans for the LOM flowrate requirements. 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 16.9.6 System Duties for Meeting Flowrate Requirement

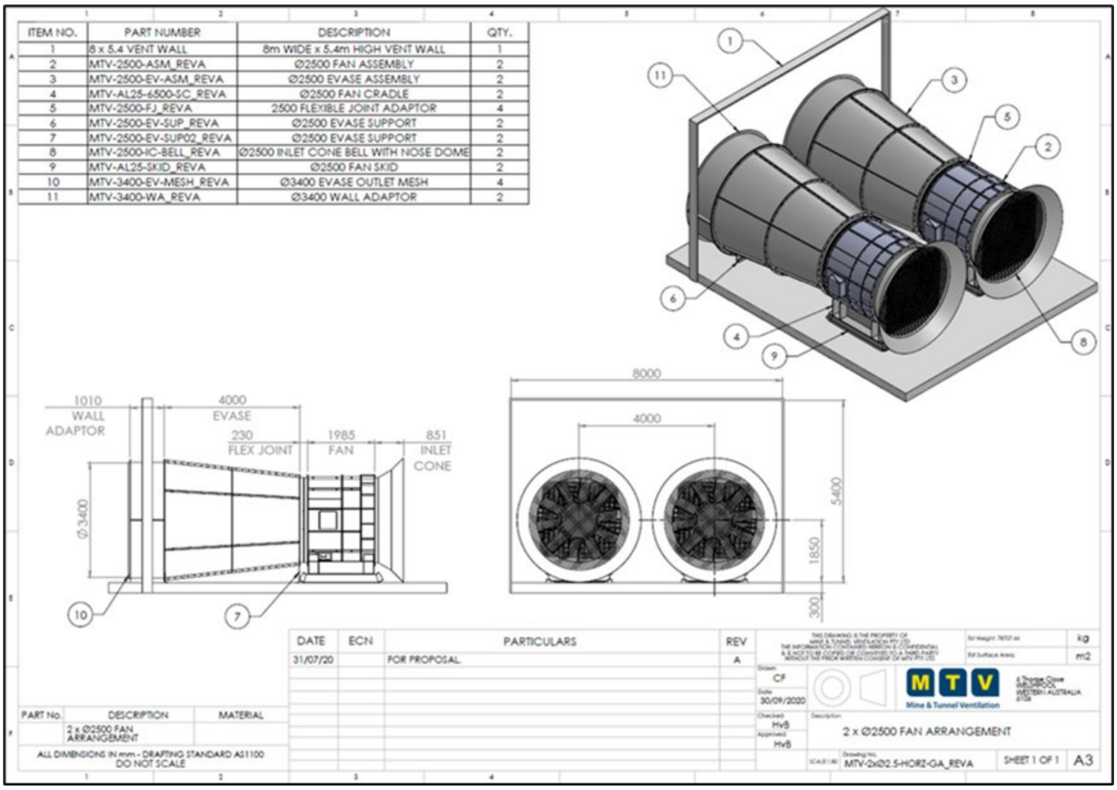
K92 System Duties for LOM							
Staging	**Mine Resistance (Ns^2/m^8)**	**Electrical Input (kW)**	**System Total Pressure (Pa)**	**System Static Pressure (Pa)**	**Inlet Density (kg/m^3)**	**Flow (kg/s)**	**Flow (m^3/s)**
Development Milestone	0.0217	3,360	5,210	5,016	0.95	466	490
Peak Flowrate @ LOM	0.0118	1,950	2,960	2,642	0.97	485	500
Min Flowrate @ LOM	0.0115	330	900	829	0.98	274	280

The 'Development Milestone' in Table 16.9.6 represents the stage in development when the RAR is commissioned, and Puma exhaust becomes the Puma intake. The mine resistance is considerably higher at this point as the connecting incline has not yet broken through to the haulage drives.

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 1.5 MW axial flow fans in parallel would suit the fan station, but this will need to be verified by the supplier. Figure 16.9.5 offers an example of what an underground fan station with two 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 K92 requirements.

Figure 16.9.5 Example of Underground Fan Station care of MTV



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 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. A ventilation on demand (VOD) system can be used to ensure the incline air flow is consistently maintained at the desired flowrate. Examples of ventilation control devices are shown in Figure 16.9.6 and Figure 16.9.7.

Figure 16.9.6 Example of Flap, or Strip Curtain That Can be Used at Ore Pass Access

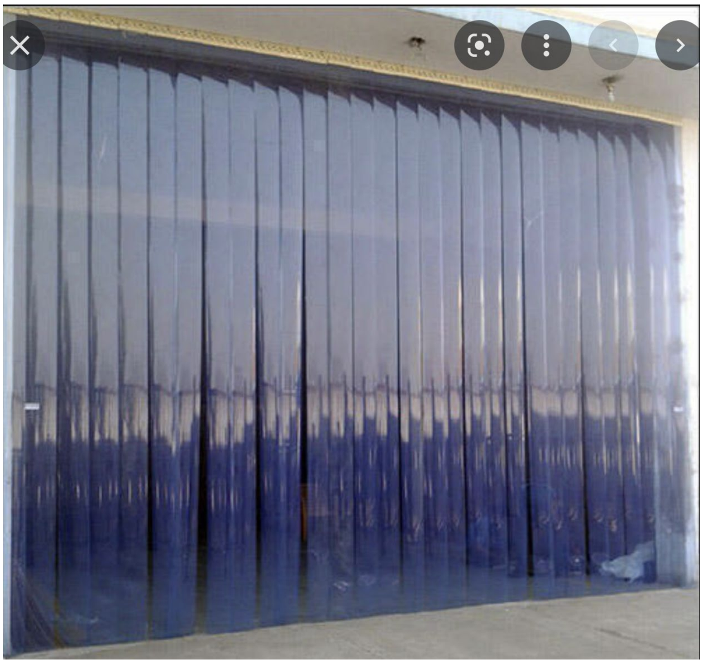


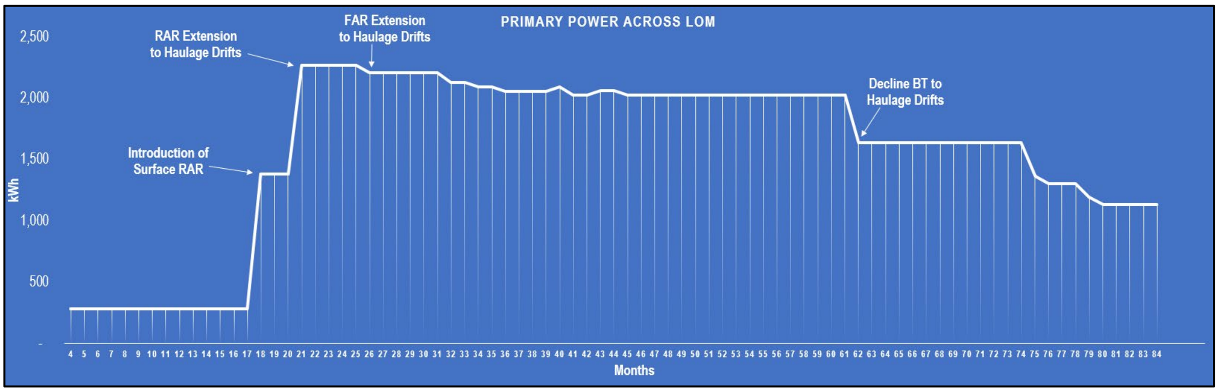
Figure 16.9.7 Example of Ventilation Louvres Care of Clemcorp and Wilshaw

The primary fan power across the LOM is represented in Figure 16.9.8 and is influenced by changes in the mine's resistance.

16.9.7 Power

The primary fan power across the LOM is represented in Figure 16.9.8 and is influenced by changes in the mine's resistance.

Figure 16.9.8 Primary Fan Power as Average kWh per Scheduled Month

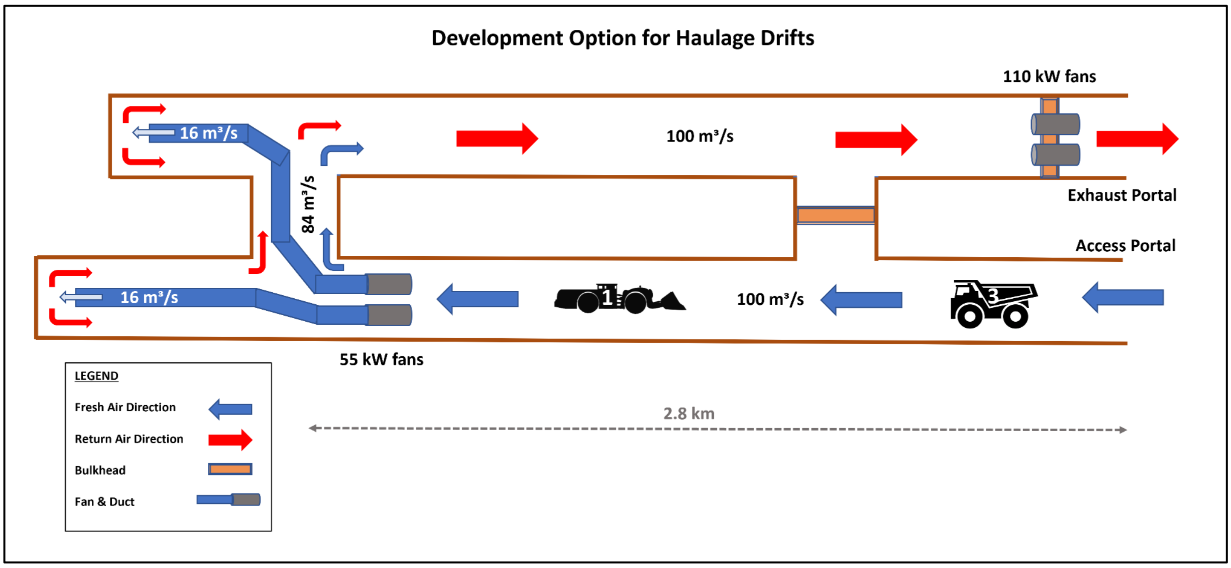


16.9.8 Recommendations and Conclusions

Speed controls like VSDs on primary fans to change the available flowrate and optimize power should be used. This will prevent overventilation when peak flow is not required. Developing the 2.8 km long twin haulage drifts will require two 132 kW auxiliary fans per drift with twin lengths of Ø1.5 m ducting per drift. A simpler option draws primary air into the drifts, as close to the development advance, as the last stockpile connection between the two drifts affords (see Figure 16.9.9).

This will create a ventilation circuit loop, providing all other stockpile connections are sealed off along the route, although vehicle access in and out of the workings can only be done via a single access portal. Two 110 kW fans in parallel, plugged into a temporary bulkhead, will provide the flowrate for three trucks and one loader.

Figure 16.9.9 Initial Development Option for Haulage Drifts



16.10 Underground Infrastructure

16.10.1 Second Means of Egress and Refuge Chambers

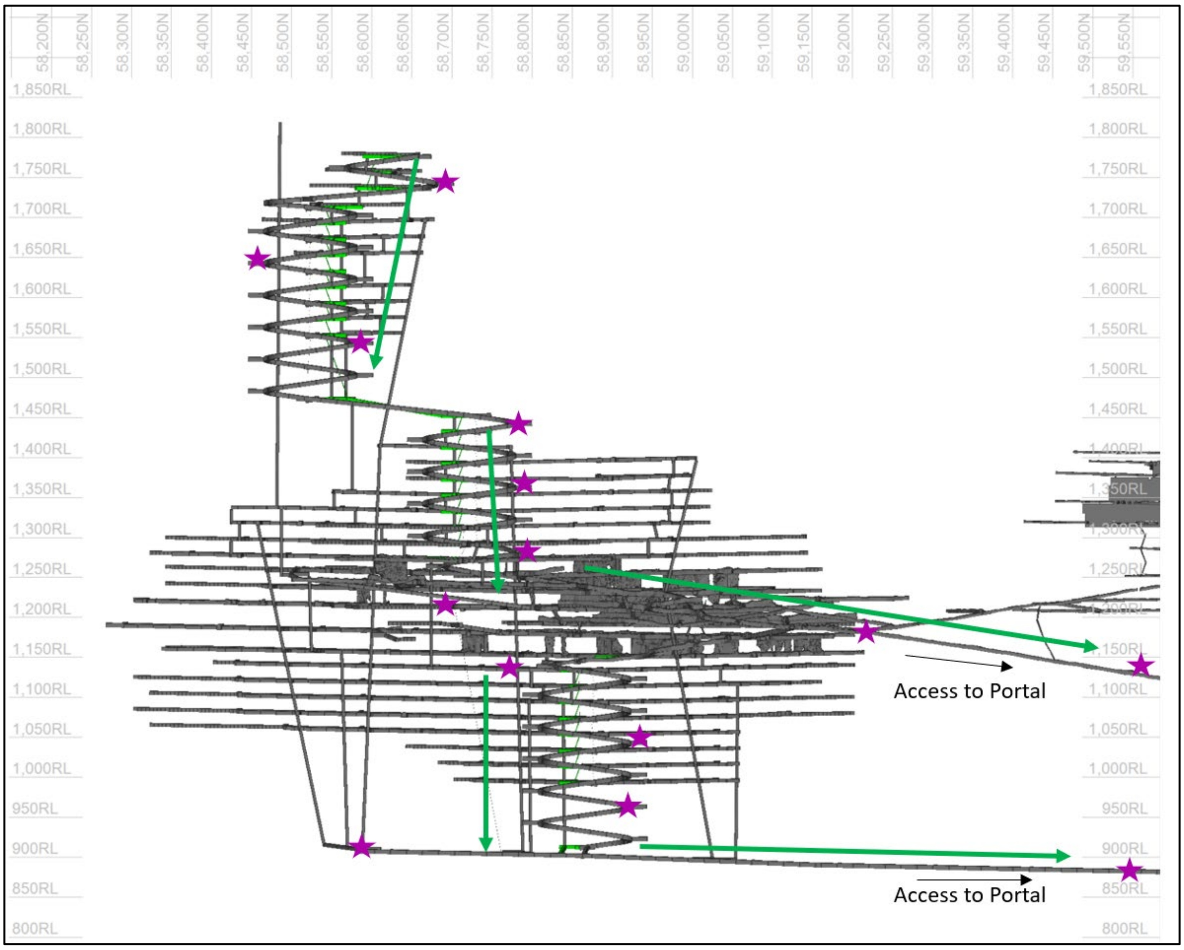
A second means of egress is currently established within the mine to enable personnel to exit the mine in case that one egress (e.g. decline) becomes blocked. To ensure that all personnel can reach a decline connection drive, ladderways are installed in the raise-bored escapeway rises which are fully contained in fresh air.

In certain cases, escaping the mine via a second means of egress in an emergency situation 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.

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-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 16.10.1, which also depicts the escape routes to surface via the portals.

Figure 16.10.1 Proposed Underground Refuge Chambers (Purple) and Escapeway Route (Green)



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.

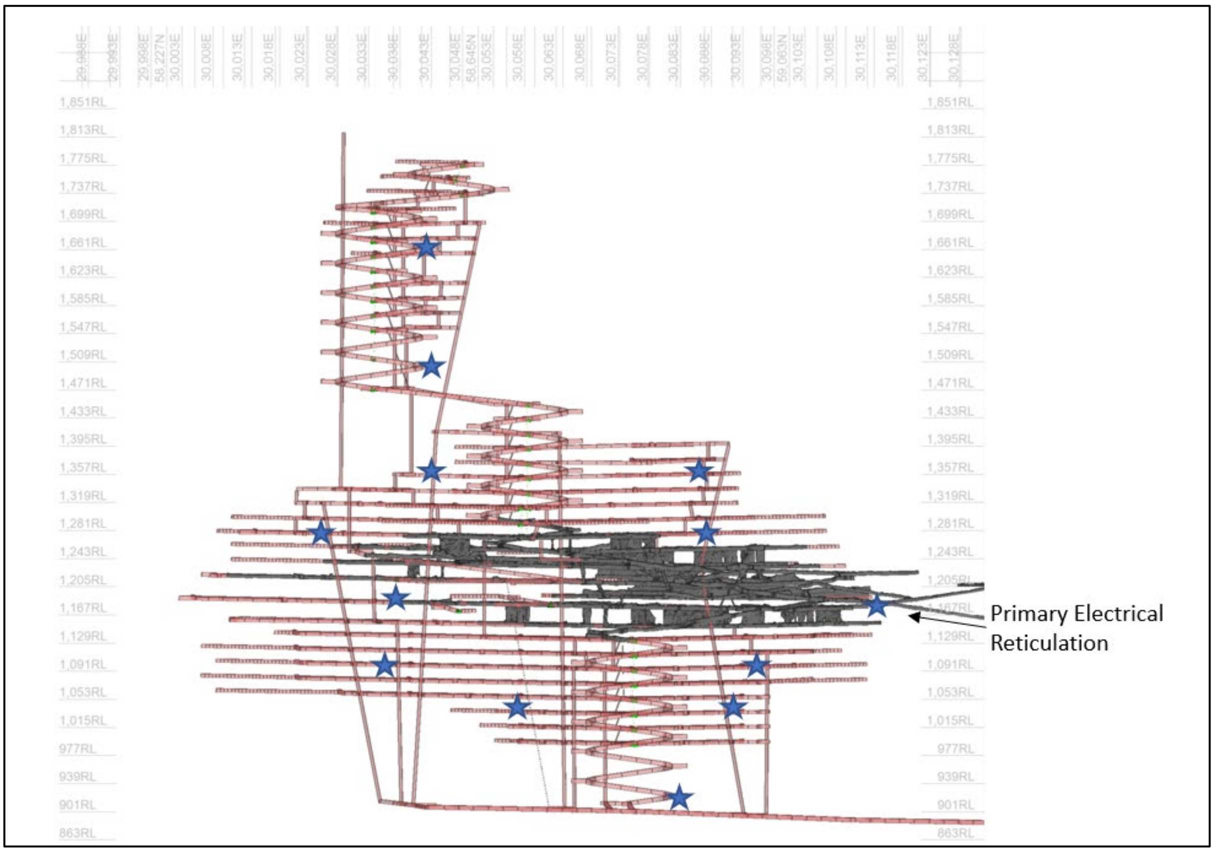
The total number of personnel working in each part of the mine is limited to the refuge chamber capacity and will be controlled by tag board procedures.

16.10.2 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 along existing 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 16.10.2 The number of substations has been estimated using an average of range of 100 m – 150 m vertical distance between sub stations, with most levels requiring two sub stations along strike given the strike length of the orebody and the associated voltage drop that occurs over these distances.

Figure 16.10.2 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 4.2MW.

16.10.3 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.

16.10.4 Hydrogeology and Dewatering

The following is an excerpt from the 'Kora Hydrogeology Assessment' completed by EMM[1]:

"Hydrogeology conditions are highly heterogeneous and primarily controlled by local topography, local recharge zones and mineralization-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 which facilitates 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 60-110 L/s. The majority of the 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 300 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 1,185 m RL. 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 incline will be completed in 2023, changing the nature of dewatering. The twin incline will intersect the Kora resource area at approximately 900 m RL, this will become the new locus of mine-discharge collection and transfer to the 800 Portal (largely negating the need for pumping to the 1,185 m RL sump/dam). Pumping infrastructure will be required for Kora mine-development below the 900 m RL. Risks associated with uncontrolled groundwater discharge will increase as underground mining depths increase.

[1] Kora Hydrogeology Assessment – EMM Dec 2021

K92's 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 2024. Paste backfilling will reduce mine-throughflow, and therefore, slow the effects of ARD. Paste backfilling is planned for Kora mining regions; however, 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 K92 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 summarized in a risk matrix, including risk-mitigation measures and further works. Requirements for additional data include:

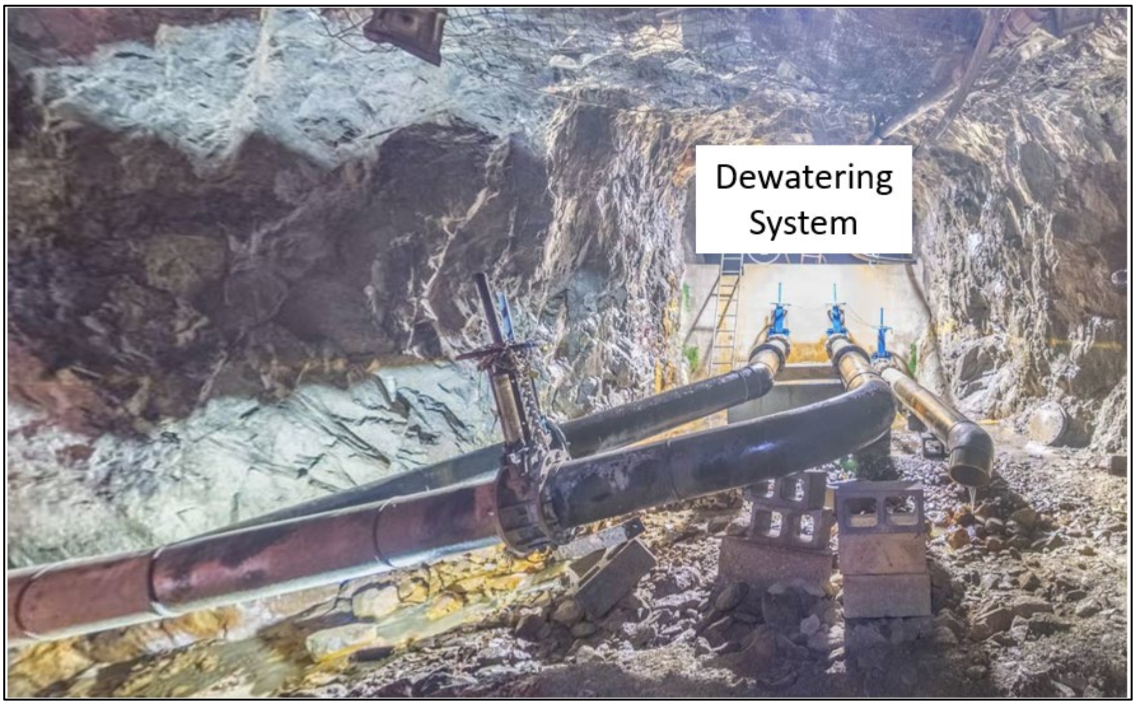
- Installation of flow meters to monitor mine-discharge volumes.
- A groundwater sampling program to characterize ARD discharge (a sampling program is provided in this report).
- Tracer tests to identify recharge locations and pathways.

- Hydrogeological mapping and monitoring of underground workings."

Primary Pumping

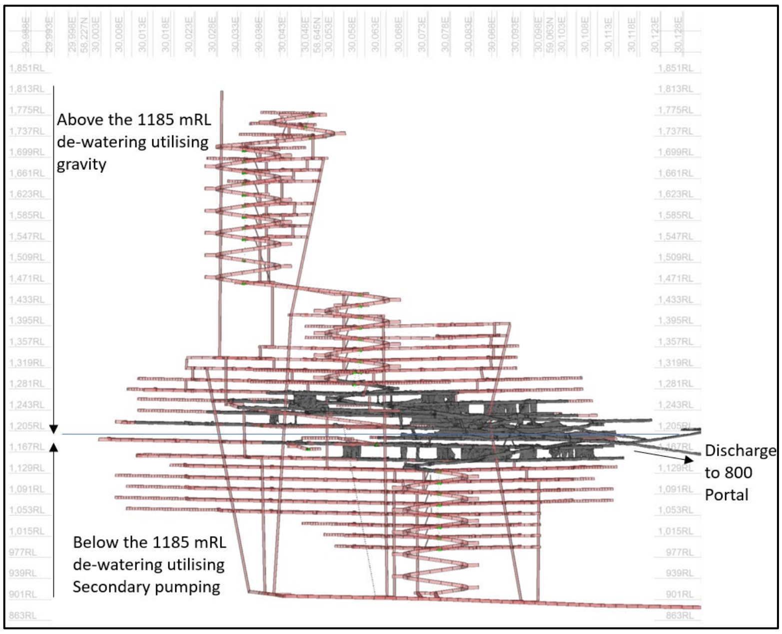
The underground mine dewatering strategy utilizes the gravity advantage of having the mine portal at a lower elevation than the currently mined areas of the deposit. Mine water is pumped via secondary pumping, to a central sump location at the 1185 mRL. From here the water is discharged via the pipes as shown in Figure 16.10.3 where it reaches the portal and discharge to a weir system where it is treated and subjected to water monitoring.

Figure 16.10.3 Central Dewatering System 1185 mRL



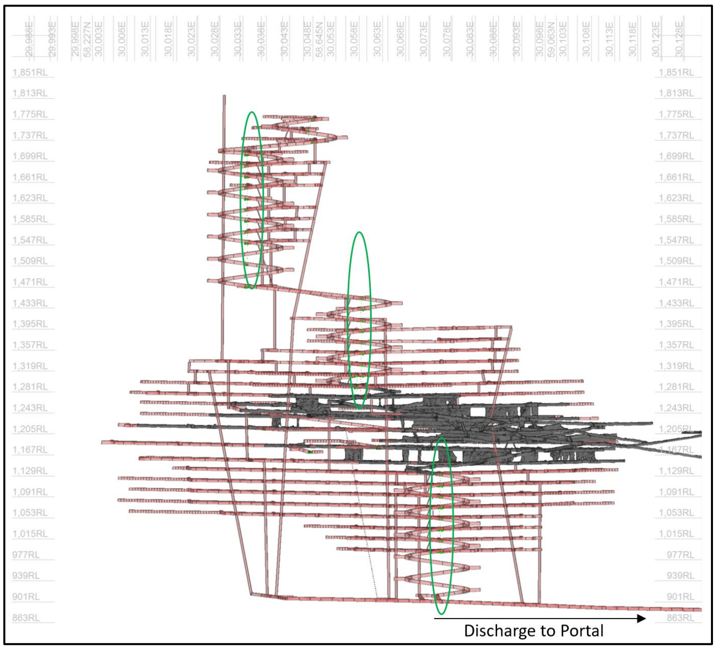
Prior to the completion of excavation of the twin haulage drives, dewatering will utilize the current site methodology as per Figure 16.10.4.

Figure 16.10.4 Looking West - Current Site Dewatering System with Full LOM Design Shown as an Example



Once the twin haulage drives have been excavated, mine water will report to the haulage level where it will be discharged to the twin haulage incline portals as per Figure 16.10.5. The areas circled in the image depict the sump and drain hole positions which will gravity feed to a primary sump location on the twin haulage drive where it will then be discharged to the portal.

Figure 16.10.5 Looking West - LOM Dewatering System (Circled in Green Planned Sump and Drain Hole Network)



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 1185 mRL, until the final system is established at the completion of twin haulage drives. For pumping up to the 1185 mRL there is also an allowance for a Triple WT114 mono pump, and a Triplex Vertical Multi-Stage Booster pump.

16.10.5 Compressed Air

There are two compressors in place that supply compressed air to the underground.

16.10.6 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 is capable of communicating with users of the surface UHF system (e.g., mill control, surface operations) via a dedicated surface-underground channel.

16.11 Underground Operations

16.11.1 Mining Fleet

Assumptions for mine equipment requirements have been derived by K92 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 16.11.1 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 16.11.1 Peak Fleet Numbers

Equipment	Quantity
Primary	
Development Drills (DD421-60C)	6
Loaders (LH517)	5
Trucks (TH545)	7
Production Drills (DL421-7C/DL431-7C/DS 421C Bolter)	3
Slot Drill - 'Rhino' (or equivalent)	1
Production Charge-up (Getman)	1
Development Charge-up (Getman)	2
Ancillary	
Spraymec (6050WP)	2
Agitator	2
IT 856H Development	3
IT 856H Production	2
12K Grader	1
Scissor Lift (Ultimec)	1
Light Vehicles	9
Raisebore	1

16.11.2 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 for expatriate employees.

The total number of employed personnel in the underground operation at peak production reaches approximately 560, with all positions detailed in Table 16.11.2, Table 16.11.3, Table 16.11.4, and Table 16.11.5.

Table 16.11.2 Total Technical Services Personnel

Technical Services Personnel	Quantity
Head of Technical Services	1
Principal	1
Officer - Administration 1	2
Graduate Engineer - Mining	2
Senior Mining Engineer 1	2
Mining Engineer 1	4
Senior Mining Engineer - Production	1
Mining Engineer 2	4
Senior Mining Engineer 2	1
Technical Specialist – Deswik	1
Technician - Mine Ventilation	3
Mining Engineer 3	2
Junior Mining Engineer	3
Senior Mining Engineer 3	2
Driver/Field Assistant	3
Field Technician - Geology	16
Senior Field Technician - Geology	3
Graduate - Geologist	2
Junior Mine Geologist	3
Geologist - Database + Mining	6
Senior Geologist - Mining	4
Superintendent - Geology	1
Assistant - Survey (Chain man)	4
Junior Mine Surveyor	2
Mine Surveyor	2
Senior Mine Surveyor	2
Registered Mine Surveyor	1
Senior Specialist - Survey	1
Field Technician	2
Geotechnical Engineer	2
Senior Geotechnical Engineer	2
Superintendent 1	1

Table 16.11.3 Supervision and Control Personnel

Personnel	Quantity
Head of Mining	1
Mining Manager	1
Mine Deputy	4
Mine Control	6
Operational Engineers	4
Mine Admin	3
Cap Lamp	6
Storeman	3
Training Coordinator	2
Trainers	4

Table 16.11.4 Mining Operations Personnel

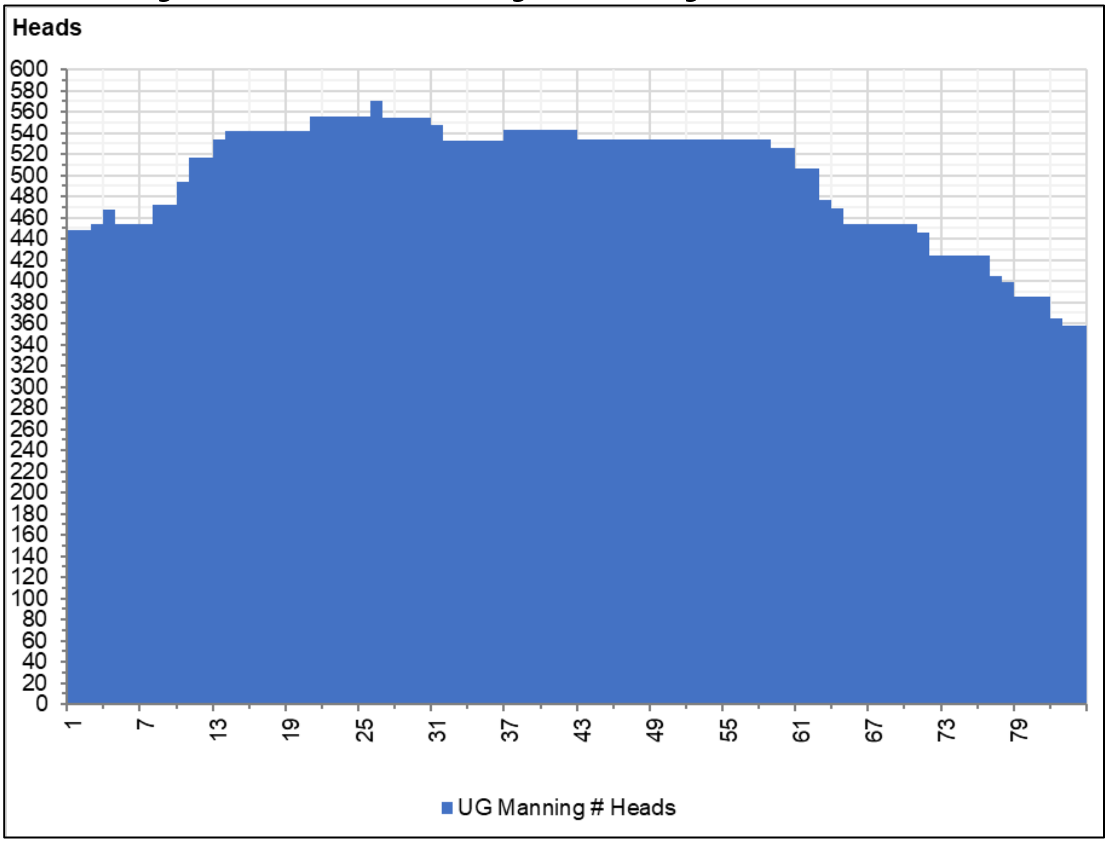
Mining Operations	Quantity
Mine Superintendent	6
Production Forman/Shift Supervisor	4
LH Operator	3
LH Operator - Large	3
LH Operator - Small	3
Bogger Operator	19
Truck Operator	32
Cablebolter Operator	5
Production Charge-up	12
Development Foreman/Shift Supervisor	4
Jumbo Operator 1	3
Jumbo Operator 2	18
Jumbo Off-Sider	23
Development Charge-up	18
Shotcrete Specialist	2
Batch Plant Operator	6
Shotcrete Sprayer	12
Agittator Operator	8
Nipper	4
Constuction Foreman/Supervisor	4
Construction Specialist	2
Service Crew A	27
Service Crew B	18
Construction Crew	9
Boilermaker	3
Grader	4
Drain Cleaners/Labourers	9
Twin Portal	
Supervisor/Jumbo	3
Multi-Skilled Miner 1	3
Multi-Skilled Miner 2	9

Table 16.11.5 Maintenance Personnel

Mining Operations	Quantity
Head of Maintenance	1
Senior Maintenance Planner 1	1
Maintenance Planner	4
Mechanical Engineer - RCM	3
Officer - Administration 2	3
Senior Maintenance Planner 2	2
Apprentice - HEF 1	6
Apprentice - Auto Electrical 1	6
Cost Controller	2
Superintendent - Electrical (UG) 1	2
Superintendent - Electrical (UG) 2	2
Coordinator - Electrical (UG)	3
Supervisor - Electrical (UG)	5
Technician - Electrical (UG) 1	6
Trades Assistant - Electrical (UG)	5
Technician - Radio (UG)	2
Technician - Electrical (UG) 2	2
Superintendent - Mobile Maintenance	6
Coordinator - Mobile Maintenance	2
Coordinator - Workshop	2
Technical Specialist - Maintenance (Automated)	2
Technical Specialist - Maintenance (Workshop)	2
Supervisor - Mobile Maintenance	4
Supervisor - Workshop	2
Supervisor - Auto Electrical	4
Supervisor - Ancillary	2
Snr Technical Specialist - Maintenance	6
Technician - Auto Electrical (UG)	4
Technician - Auto Electrical (Surface)	4
Technician - Fitter (HEF)	25
Technician - LV Mechanic	12
Technician - Fabricator	2
Supervisor - Welding	2
Supervisor - Fabrication	1
Technician - Boilermaker	6
Tool Storeman	6
Trades Assistant - HEF	21
Technician - Tyre Fitting	7
Cleaner	3
Technician - Hose	6
Technician – Panel Beating	2

Peak underground Personnel requirements (total employed personnel, not all on site simultaneously) by month are found in Figure 16.11.1.

Figure 16.11.1 Total Underground Mining Personnel Per Month



16.11.3 Mine Facilities

The site layout is illustrated in Figure 16.11.2.

Figure 16.11.2 Overall Site Layout (~6km From Mine Portal to Processing Plant)

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 in which is in close proximity to the processing plant approximately 6 km from the mine portal.

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.

Fuel Storage

Diesel is stored on surface in existing licensed facilities.

Explosives Storage

There is an existing surface magazine compound.

17.0 RECOVERY METHODS

17.1 Overview

The process plant design for the Project is based on a robust metallurgical flowsheet designed for optimal metal recovery. The flowsheet chosen is based upon unit operations that are well proven in industry.

The key criteria for equipment selection are suitability for duty, reliability, and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements, whilst maintaining a layout that will facilitate construction progress in multiple areas concurrently.

The key project design criteria for the plant are:

- Throughput of 1.2 Mtpa of ore with a grind size of 80% passing (P_{80}) 106 µm.
- Major unit operations and equipment will be sized with a 20% design margin.
- Crushing plant utilization of 68.5% (6,000 h/y).
- Grinding and flotation plant utilization of 91.3% (8,000 h/y) supported by crushed ore storage, stand-by equipment in critical areas and emergency power, if required, for controlled shutdown, emergency lighting and continuous operation of critical equipment.
- Sufficient automated plant control to minimize the need for continuous operator interface and allow manual override and control as required.

Study design documents have been prepared incorporating engineering design criteria and key metallurgical design criteria derived from the results of the metallurgical testwork as discussed in Section 13, along with comminution circuit modelling.

17.1.1 Selected Process Flowsheet

The treatment plant design incorporates the following unit process operations:

- Primary crushing with a jaw crusher to produce a crushed product size of 80% passing (P_{80}) 92 mm.
- A crushed ore overflow surge bin with a design live volume of 120 m^3 and a dead stockpile to provide 16 hours of operation at design plant throughput.
- Crushed ore from the overflow surge bin is reclaimed by an apron feeder positioned under the bin to feed the grinding circuit.

- The grinding circuit is a SAB type, which consists of an open circuit SAG mill and a closed-circuit ball mill with hydrocyclones, to produce a cyclone overflow product size of P_{80} 106 µm. The cyclone overflow stream will flow by gravity to the flotation circuit.

- A gravity gold recovery circuit for removal of coarse gold from the grinding circuit is included. Part of the cyclone underflow is directed to the gravity circuit. The circuit contains one batch centrifugal concentrator followed by two stages of gravity separation using shaking tables. The upgraded gravity concentrate is calcined and smelted in the gold room to produce gold doré bars.

- A flash flotation circuit for removal of coarse gold and copper particles from the grinding circuit is included. Part of the cyclone underflow is directed to the flash flotation circuit, which contains one flash flotation cell. Flash flotation concentrate will flow by gravity to the concentrate thickener for addition to the final saleable concentrate product. The flash flotation tailings will flow by gravity back to the grinding circuit.

- The flotation circuit consists of rougher, scavenger, cleaner, cleaner scavenger and recleaner stages to produce a saleable high-grade gold-copper concentrate product.

- The flotation concentrate will be pumped to the concentrate thickener to dewater and increase the slurry density prior to concentrate filtration. The thickened concentrate will be filtered to achieve a discharge cake moisture of 10% before being loaded into 20 ft sea-containers for shipment overseas.

- The flotation tailings thickener will increase the slurry density for water recovery prior to tailings discharge to either the paste plant for backfill in the underground mine workings or the tailings storage facility (TSF).

- Raw water storage and distribution.

- Process water storage and distribution.

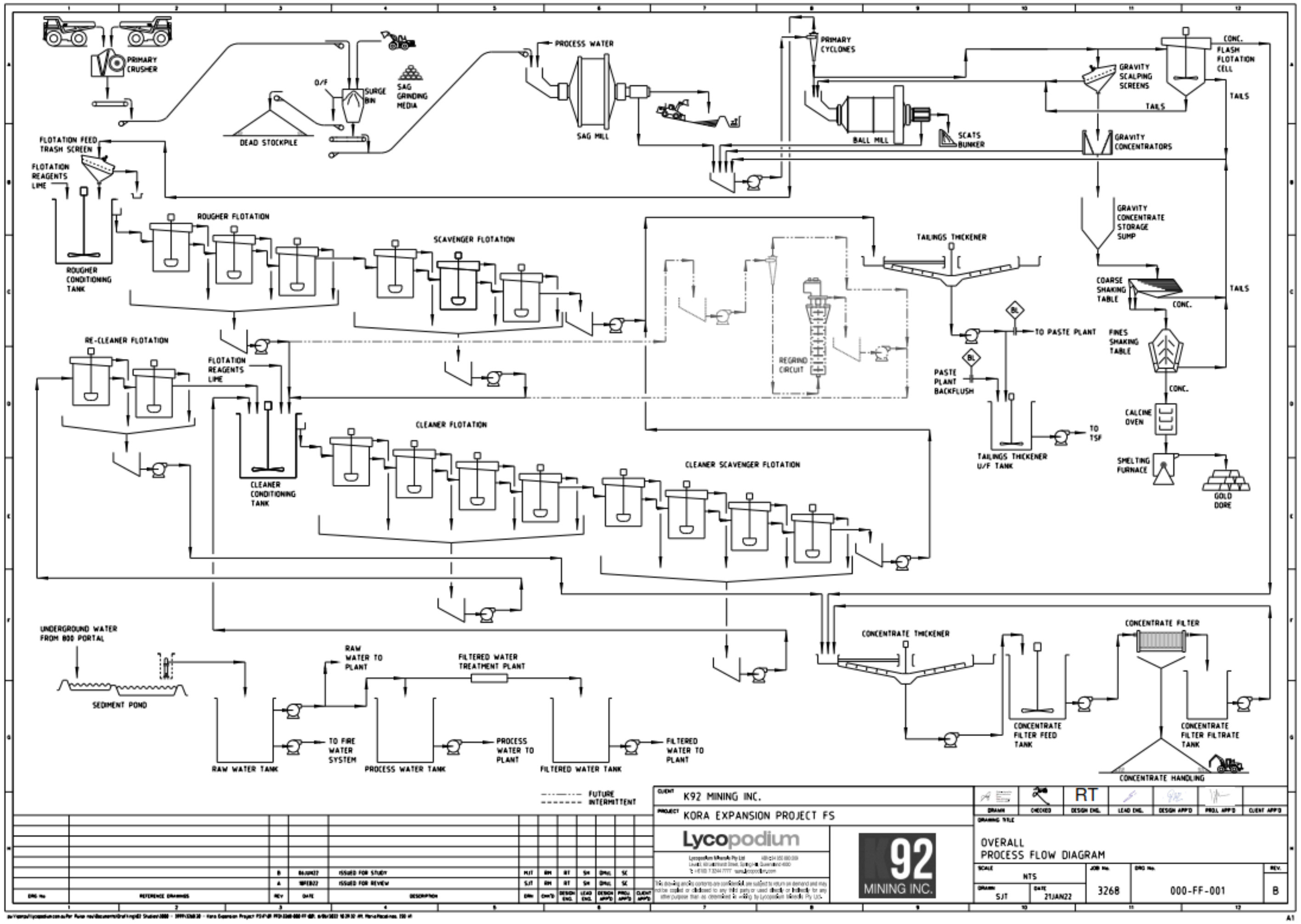
- Reagents mixing, storage and distribution.

- Compressed air services.

An overall process flow diagram depicting the unit operations incorporated in the selected process flowsheet is presented in Figure 17.1.1.

The key issues considered in process and equipment selection are outlined in the following section.

Figure 17.1.1 Overall Process Flow Diagram



17.1.2 Key Process Design Criteria

The key process design criteria listed in Table 17.1.1 form the basis of the detailed process design criteria and mechanical equipment list.

Table 17.1.1 Summary of Key Process Design Criteria

	Units	Design	Source
Plant Throughput	tpa	1,200,000	Client
Head Grade	Au g/t	10.0	Client
	Cu %	1.0	Client
Overall Gold Recovery – Range LOM	%	90 - 95	Client
Design	%	92.9	Client
Overall Silver Recovery – Design	%	80.0	Client
Overall Copper Recovery – Design	%	95.1	Client
Crushing Plant Feed Rate	t/h	200	Client
Crushing Plant Availability	%	68.5	Calculated
Plant Feed Rate	t/h	150	Client
Plant Operating Hours	h/y	8,000	Client
Plant Availability	%	91.3	Calculated
Crushing Work Index, CWi (85th Percentile)	kWh/t	16.4	Testwork
Bond Ball Mill Work Index, BWi (85th Percentile)	kWh/t	16.8	Testwork
SMC Test Parameter, A*b (15th Percentile)		56.4	Testwork
Bond Abrasion Index, Ai (Average)		0.129	Testwork
Grinding Circuit Product Size, P_{80}	µm	106	Client
Gravity Circuit Gold Recovery	%	15	Client
Mass Recovery to Flash Flotation Concentrate	%	5.3	Modelling
Total Flotation Residence Time (Lab)	min	24	Testwork
Plant Flotation Residence Time	min	60	2.5 Scale-up Factor
Mass Recovery to Flotation Concentrate	%	5.0 - 8.0	Client
Concentrate Thickener Settling Flux	$t/m^2/h$	0.25	Testwork
Concentrate Thickener Underflow Density	% solids (w/w)	68	Testwork
Concentrate Filtration Rate	$kg/h/m^2$	390	Testwork
Concentrate Filter Discharge Cake Moisture	%	10	Client
Tailings Thickener Settling Flux	$t/m^2/h$	0.50	Testwork
Tailings Thickener Underflow Slurry Density	% solids (w/w)	59	Consultant

17.2 Process and Plant Description

The overall process flowsheet includes a single stage primary jaw crusher and a SAB grinding circuit, SAG mill in open circuit and a ball mill in closed circuit with hydrocyclones to achieve the target product size. Based on the metallurgical testwork results, part of the cyclone underflow stream is directed to a gravity circuit with centrifugal gravity concentrator and two stages of further upgrading using gravity shaking tables. Also, part of the cyclone underflow is directed to the flash flotation circuit for upgrading coarse gold and copper particles. The flash flotation concentrate will be sent to the concentrate thickener as final product, while flash flotation tailings will be directed back to the ball mill feed box.

The cyclone overflow stream flows by gravity to the flotation circuit. The flotation cells are arranged in rougher, scavenger, cleaner, cleaner scavenger, and recleaner stages. Based on the metallurgical testwork results, pH is adjusted to 10.0 - 10.5 using hydrated lime slurry in the rougher conditioning tank. Various reagents such as a blended collector, a frother, and CMC as depressant will be added to the flotation circuit at various locations. The recleaner concentrate stream is pumped to a flotation concentrate thickener to be thickened to increase slurry density for water recovery prior to concentrate filtration. The thickened concentrate will be pumped to a concentrate filter to be filtered and achieve 10% moisture in the filter cake. The final filter cake will be loaded into 20 ft sea-containers.

The flotation tailings stream is thickened to recover water before being pumped to either the paste plant for backfill in the underground mine workings or the tailings storage facility.

17.2.1 Ore Receiving and Crushing

Run-of-mine (ROM) ore from the underground mine, at maximum lump size of 1000 mm, will be transported to the plant by 60 t capacity dump trucks. The trucks will tip directly into the primary crusher feed bin, or onto the blending fingers on the ROM pad. ROM ore can be reclaimed to the primary crusher feed bin with an installed static grizzly, by a Komatsu WA 500-8 or CAT 988 front-end loader. Ore will be reclaimed from the primary crusher feed bin by an apron feeder to a primary vibrating grizzly with 90 mm aperture size. Ore smaller than grizzly aperture will fall onto the surge bin feed conveyor. The oversize material will be fed to the jaw crusher, which has been selected as most suitable for the material characteristics, tonnage throughput requirements, maximum feed particle size and required particle size distribution.

The crushed ore will be conveyed, via the surge bin feed conveyor, to an overflow surge bin. The surge bin feed conveyor will be fitted with a weightometer, to monitor the primary crusher throughput, and for controlling the primary apron feeder variable speed drive.

The crushing circuit will be serviced by a single dust collection system, consisting of multiple extraction hoods, ducting and a baghouse. Dust collected from this system will be discharged onto the surge bin feed conveyor and then finely sprayed with water for dust suppression.

Auxiliary equipment for the crushing circuit will include:

- Primary crusher grease pump.

17.2.2 Crushed Ore Surge Bin and Stockpile

The crushed ore will be conveyed to the surge bin. Excess crushed ore will overflow and be conveyed to a dead stockpile. During crusher maintenance, stockpiled ore will be reclaimed using an FEL and will be tipped directly into the surge bin.

The overflow surge bin will have designed live volume of 120 m^3 (equivalent to 192 tonnes of dry ore) and a total storage capacity of approximately 77 minutes. The dead stockpile will have a capacity of approximately 2,400 t (equivalent to 1,500 m^3) and a total storage capacity of 16 hours of mill feed throughput for 1.2 Mtpa.

Crushed ore will be reclaimed from the surge bin by one variable speed apron feeder. The feeder will discharge onto the surge bin reclaim conveyor and then onto SAG mill feed conveyor. SAG mill feed conveyor will convey the crushed ore to the SAG mill feed chute. The SAG mill feed conveyor will be fitted with a weightometer and used for controlling the speed of the surge bin reclaim feeder and for mass accounting of feed presented to the grinding circuit.

SAG mill grinding media will be added to the surge bin. SAG mill grinding balls will be loaded from the SAG mill ball bunker into the surge bin via an FEL.

17.2.3 Grinding and Classification

The Kora grinding circuit will be a traditional SAB circuit, consisting of a single semi-autogenous grinding (SAG) mill and a single ball mill. The SAG mill will operate in open circuit, whilst the ball mill will operate in closed circuit with hydrocyclones. The product exiting the grinding circuit in the cyclone overflow slurry stream will contain 80% particles passing 106 µm.

Process water will be added to the SAG mill feed chute, to control the in-mill pulp density. Hydrated lime slurry will also be added to the SAG mill feed chute, to adjust slurry pH prior to the downstream flotation circuit. The SAG mill product will discharge via trommel screen for classification.

SAG mill trommel oversize will be collected in the SAG mill scats bunker and will be sent back to the grinding circuit using a front-end loader. Trommel screen undersize will flow by gravity to the cyclone feed hopper, where it will combine with the discharge slurry from the ball mill, tailings from the gravity concentrator, shaking tables and the upper outlet from the flash flotation cell. The slurry will be pumped to the cyclone cluster. Process water will be added to the mills discharge launders and cyclone feed hopper for cyclone feed density control.

The cyclone cluster overflow will flow by gravity through a metallurgical sampler, for metallurgical accounting purposes, to the flotation circuit. Slurry from the cyclone underflow launder, will be split into three streams via the primary cyclone underflow boil box. One part of the stream will flow by gravity to the flash flotation feed box. Another part of the cyclone underflow will flow to the gravity scalping screen. The remaining slurry will flow by gravity to the ball mill feed chute for grinding. Ball mill discharge will pass through the ball mill trommel prior to discharging to the cyclone feed hopper. Reject oversize material, from the ball mill trommel screen, will be collected within the ball mill scats bunker.

Grinding media will be added to the ball mill via the ball mill feed chute, utilising a dedicated media hoist, kibble, and feed chute.

Spillage within the grinding area will be managed with sump pumps. Any spillage generated in the grinding area will be returned to the cyclone feed hopper.

Auxiliary equipment within the grinding area will include:

- SAG mill lubrication and cooling system.
- Ball mill lubrication and cooling system.
- One common variable speed drive (VSD) for ball mill start-up, followed by SAG mill start-up and rotational speed adjustment for operation.
- Mills liner handler and relining machine, and bolt removal tool.
- Mills feed chute removal winch.
- Ball mill ball charging hoist.
- Primary cyclone maintenance jib crane.
- Metallurgical slurry sampler.

17.2.4 Gravity Circuit

Recovery of gravity gold will be achieved via one operating gravity centrifugal concentrator. Approximately one-third of the cyclone underflow stream via the primary cyclone underflow boil box will flow by gravity to a gravity scalping screen. The scalping screen prevents coarse heavy particles and mill ball steel chips from being recovered by the gravity concentrator.

The gravity concentrator will typically be operated with a concentrating cycle time of approximately 45 minutes. The concentrate recovered will be purged at the end of a cycle. The purged or flushed concentrate will flow to the gravity concentrate storage hopper before being transferred to the gold room for further upgrading using shaking tables.

Oversize from the scalping screen will flow by gravity to the ball mill feed box. Tailings from the gravity concentrator will flow by gravity to the cyclone feed hopper.

The collected concentrate from the gravity concentrator will be transferred to the gold room to be further upgraded using coarse and fine shaking tables in series.

Spillage within the gravity circuit will be managed, utilising a nearby floor sump pump in the grinding area. Any spillage generated in the gravity area will be pumped to the cyclone feed hopper.

Auxiliary equipment within the gravity circuit area will include the gravity concentrator hoist.

17.2.5 Goldroom

The concentrate from the gravity circuit will be first upgraded using a coarse shaking table. The concentrate will then be upgraded further using a fine shaking table. Final concentrate will be transferred by hand to the calcine ovens for drying and calcining. The tailings stream from both shaking tables will be combined and transferred back to the cyclone feed hopper using the shaking table tails return pump. Process water will be used as wash water for both shaking tables, which will be distributed across the shaking tables using spray nozzles.

The upgraded gravity concentrate will be dried in a calcining oven operating at approximately 700 – 800°C. Calcined concentrate will be mixed with a prescribed flux mixture (silica, nitre, soda ash and borax), prior to being charged into the diesel fired smelting furnace. The fluxes added react with base metal oxides to form a slag, whilst the gold and silver remains as molten metal. The molten metal will be poured into moulds, to form doré bars, which will be cleaned, assayed, stamped, and stored in a secure vault ready for dispatch. The slag produced will periodically be returned to the grinding circuit, via the SAG mill feed chute.

The gold room will be serviced by a dedicated gold room area sump pump with a gold trap.

Auxiliary equipment for the gold room will include:

- Smelting furnace dust collector and extraction fan.
- Calcine oven fume scrubber and ventilation fan.
- Goldroom ventilation fans.
- Gold pouring cascade trolley along with a slag cone and cart.
- Doré bar moulds and doré bar cleaning table.
- Flux bin, platform scale, flux mixing table.
- Balances for doré bars and prill samples.

17.2.6 Flash Flotation

Based on the K92's recommendation, a flash flotation circuit will be used to recover coarse gold and copper particles and will reduce the recirculating load on the ball mill. Approximately one-third of the cyclone underflow stream will flow by gravity to the flash flotation feed box. Hydrated lime slurry for adjusting the pH to 10.0 – 10.5 will be added to the flash flotation feed box. Collector for improving the floatability properties of the copper-gold minerals will be added to the primary cyclone underflow boil box. CMC as depressant for depressing gangue minerals will be added to the flash flotation feed box. The concentrate from the flash flotation circuit will flow by gravity to the concentrate thickener. Tailings from the lower outlet on the flash flotation cell will flow by gravity to the ball mill feed box. For the grinding circuit water balance, tailings from the flash flotation cell upper outlet will flow by gravity to the cyclone feed hopper. A manual sampling point for concentrate and tailings streams will be included.

17.2.7 Flotation Circuit

The cyclone overflow stream will flow by gravity to the flotation feed trash screen and process water will be used for the screen sprays. The screen oversize will be collected in a trash bin. The screen undersize will flow by gravity to the rougher conditioning tank. A process sampler will be installed between the trash screen underpan and the rougher conditioning tank to provide a sample to the OSA for process control purposes.

The flotation circuit will consist of following stages:

- Rougher and scavenger flotation.
- Cleaner flotation.
- Cleaner scavenger flotation.

- Recleaner flotation.

The mechanical cell configurations and size selections were based on the smallest number of the largest cells that provided the required retention time without significant short-circuiting. Furthermore, this will minimize installed capital, footprint, and operating costs. The cell selection also considered cell froth carrying capacity (froth area) and cell froth launder lip length.

Ultimately, the mechanical cell selection was based on residence time requirements, which dictated flotation cell size. The design residence times selected are presented in Table 17.2.1. The chosen cell volumes allow for a 15% air hold up.

Table 17.2.1 Flotation Time

Flotation Stage	Scale-Up Factor	Flotation Time (min)	
		Laboratory	Plant Target
Rougher and Scavenger	2.5	10	25
Cleaner	2.5	4	10
Cleaner Scavenger	2.5	6	15
Recleaner	2.5	4	10

Minimum lip loadings and froth carrying capacities to select the launder configuration are presented in Table 17.2.2.

Table 17.2.2 Solids Lip Loading and Froth Carrying Capacity Design Parameters

Flotation Stage	Solids Lip Loading (t/m/h)	Froth Carrying Capacity ($t/m^2/h$)
Rougher and Scavenger	1.5	2.0
Cleaner	1.5	2.0
Cleaner Scavenger	1.5	2.0
Recleaner	1.5	2.0

The flotation reagent addition regime was selected based on the testwork data and the dosing rates were confirmed by K92. Summary of the flotation reagents are presented in Table 17.2.3.

Table 17.2.3 Flotation Reagent Addition Design Criteria

Description	Units	Value
Collector		
Type		Blend 1 (Aero3477, Aero6697 and Aero5100 with mass ratio of 64:16:20)
Source		1,000 L IBC
Physical Form		Liquid
Total Addition Rate	g/t	85
Frother		
Type		Dowfroth 250 or equivalent
Source		1,000 L IBC
Physical Form		Liquid
Total Addition Rate	g/t	15
Depressant		
Type		CMC or equivalent
Source		1,000 kg Bulk Bags
Physical Form		Powder
Total Addition Rate	g/t	77.5
pH Modifier		
Type		Hydrated Lime
Source		1,000 kg Bulk Bags
Physical Form		Dry Powder
Total Addition Rate	kg/t	2.0

Rougher Scavenger Flotation

The rougher scavenger flotation circuit will consist of a single conditioning tank for a residence time of 10 minutes, followed by three 40 m^3 rougher cells and three 40 m^3 scavenger cells in series. Low pressure air will be supplied to each flotation cell. The air flow to each cell will be controlled by vendor-supplied instrumentation. Similarly, cell levels will be controlled by vendor-supplied instrumentation. Lime slurry, collector and CMC (as depressant) will be added to the rougher conditioner tank, along with process water for dilution to the required density, if necessary. Frother will be added to the first rougher flotation cell. Slurry will flow by gravity from the rougher conditioner tank to the rougher flotation cells. Concentrate from rougher flotation cells will report to the rougher concentrate hopper. The rougher concentrate pump will transfer slurry to the cleaner conditioning tank. A pressure pipe sampler will be installed to provide a sample to the OSA for process control purposes.

The tailings from the third rougher cell will flow by gravity to the first scavenger cell. In the scavenger stage, collector and frother will be added to the third rougher cell and first scavenger cell respectively. Concentrate from scavenger cells will report to the scavenger concentrate hopper. The scavenger concentrate pump will transfer slurry to the cleaner conditioning tank. A pressure pipe sampler will be installed to provide a sample to the OSA for process control purposes.

Launder and hopper water sprays will be provided to assist in discharge of concentrate from launders and froth breakdown.

Scavenger tailings will flow by gravity to the scavenger tailings hopper. The scavenger tailings pump will transfer slurry to the tailings thickener feed box. A pressure pipe sampler will be installed to provide a sample to the OSA for process control purposes.

Two separate sump pumps in the rougher and scavenger areas will be provided to return spillage to the rougher conditioning tank.

Cleaner Flotation

The cleaner flotation circuit will consist of a single conditioning tank for a residence time of 4 minutes, followed by four 10 m^3 cleaner cells. Low pressure air will be supplied to each flotation cell. The air flow to each cell will be controlled by vendor-supplied instrumentation. Similarly, cell levels will be controlled by vendor-supplied instrumentation. Cleaner scavenger concentrate stream and re-cleaner tailings stream will be added to the cleaner conditioning tank and will be combined with rougher and scavenger concentrate streams. Lime slurry, collector and CMC (as depressant) will be added to the cleaner conditioning tank, along with process water for dilution to the required density, if necessary. Frother will be added to the first cleaner cell. Slurry will flow by gravity from the cleaner conditioning tank to the cleaner flotation cells. A launder shark fin sampler will be installed to provide a sample to the OSA for process control purposes. A vertical spindle slurry pump will also be installed to transfer the cleaner feed sample to the OSA.

Concentrate from cleaner flotation cells will report to the cleaner concentrate hopper. The cleaner concentrate pump will transfer slurry to the first re-cleaner flotation cell. A pressure pipe sampler will be installed to provide a sample to the OSA for process control purposes. Launder and hopper water sprays will be provided to assist in discharge of concentrate from launders and froth breakdown.

Tailings from the cleaner flotation hopper will flow by gravity to the first cleaner scavenger cell. A sump pump will be provided to return any spillage in the area to the cleaner conditioning tank or the cleaner concentrate hopper.

Cleaner Scavenger Flotation

The cleaner scavenger flotation circuit will consist of four 10 m^3 flotation cells. Low pressure air will be supplied to each flotation cell. The air flow to each cell will be controlled by vendor-supplied instrumentation. Similarly, cell levels will be controlled by vendor-supplied instrumentation. For the cleaner-scavenger stage, collector and depressant will be added to the fourth and last cleaner flotation cell. Frother will be added to the first cleaner scavenger cell.

Concentrate from these cells will report to the cleaner scavenger concentrate hopper. The cleaner scavenger concentrate pump will transfer slurry to the cleaner conditioning tank. A pressure pipe sampler will be installed to provide a sample to the OSA for process control purposes. Launder and hopper water sprays will be provided to assist in discharge of concentrate from launders and froth breakdown.

Tailings from the cleaner scavenger cells will flow by gravity to cleaner scavenger tailings hopper. The cleaner scavenger tailings pump will transfer slurry to the tailings thickener feed box. A pressure pipe sampler will be installed to provide a sample to the OSA for process control purposes.

A sump pump will be provided to return spillage to the cleaner conditioning tank or the cleaner scavenger concentrate hopper.

Recleaner Flotation

The recleaner flotation circuit will consist of two 10 m^3 flotation cells. Low pressure air will be supplied to each flotation cell. The air flow to each cell will be controlled by vendor-supplied instrumentation. Similarly, cell levels will be controlled by vendor-supplied instrumentation. For the re-cleaner stage, collector will be added to the cleaner concentrate hopper. Frother will be added to the first re-cleaner flotation cell.

Concentrate from these cells will report to the re-cleaner concentrate hopper. The re-cleaner concentrate pump will transfer slurry to the concentrate thickener. A pressure pipe sampler will be installed to provide a sample to the OSA for process control purposes. Launder and hopper water sprays will be provided to assist in discharge of concentrate from launders and froth breakdown.

Tailings from the re-cleaner cells will flow by gravity to the cleaner conditioning tank. A launder shark fin sampler will be installed to provide a sample to the OSA for process control purposes. A vertical spindle slurry pump will also be installed to transfer the re-cleaner tailing sample to the OSA.

A sump pump will be provided to return spillage to the cleaner conditioning tank or the re-cleaner concentrate hopper.

17.2.8 Concentrate Thickening and Filtration

Recleaner concentrate plus flash flotation concentrate will be pumped to the concentrate thickener along with filtrate return from the concentrate filter. A ten-metre diameter high rate thickener will be used to dewater the concentrate. The thickener will include an auto dilution feed well, froth sprays to break any froth build-up on the surface, and a bridge supported drive mechanism. The thickener rakes can be raised using an automated lifting device should the torque exceed a pre-set value. Flocculant stock solution will be further diluted with process water in an in-line static mixer prior to addition to the concentrate thickener to aid with particle settling.

Thickener overflow will flow by gravity to the process water tank. Underflow from the concentrate thickener, at 68% solids slurry density, will be pumped to the concentrate filter feed tank using the duty / standby concentrate thickener underflow pumps. The concentrate filter feed tank will have approximately 12 hours of storage capacity.

The concentrate will be pumped by dedicated duty / standby filter feed pumps to the concentrate filter. The filter will remove water from the concentrate to meet the target moisture of 10% using a series of pressing and air blowing steps. After the desired filtration time, the filter press will open and discharge final concentrate directly to the floor of the concentrate building. Following discharge of concentrate, the filter cloth will be washed prior to the next cycle, using process water from the cloth wash tank. Filtrate from the concentrate filter will be returned to the concentrate thickener using the concentrate filter filtrate pump. The filter area will be completely covered. The filtered concentrate will then be loaded into 20 ft sea-containers for transportation.

The concentrate thickener area will be serviced by a dedicated floor sump pump. Spillage and wash down collected by the sump pump will be returned to the concentrate thickener feed box.

Auxiliary equipment within the concentrate thickener and filter area will include:

- Concentrate thickener electric rake drive.
- Concentrate thickener feed metallurgical slurry sampler.
- Concentrate thickener underflow process slurry sampler.
- Concentrate filter maintenance hoist.

17.2.9 Flotation Tailings Thickening

Tailings from the flotation circuit, viz scavenger tailings plus cleaner scavenger tailings, will be transferred via dedicated pumps to the flotation tailings thickener feed box. A metallurgical sampler will be installed on the thickener to provide a tailings sample for metallurgical accounting purposes.

Flocculant stock solution will be further diluted with process water in an in-line static mixer prior to addition to the tailings thickener to assist with solids settling and maintaining overflow clarity. Overflow solution from the flotation tailings thickener will be pumped to the process water tank where it will mix with concentrate thickener overflow. The process water tank will provide process water for all duties in the plant area and water to the paste plant during flushing.

Flotation tailings thickener underflow, at a slurry density of 59% solids, will be pumped by dedicated tailings thickener underflow pumps to the paste plant most of the time. During periods of thickener maintenance and/or paste plant backflush, the tailings thickener feed or underflow stream, and paste plant flush / recycle will be directed to a tailings thickener underflow tank. The tailings thickener underflow tank will have approximately 30 minutes of storage capacity. The slurries then will be pumped by dedicated pumps to the tailings storage facility.

The flotation tailings thickening circuit will be serviced by one floor sump pump. Any spillage collected within this area will be pumped to the tailings thickener feed box or overflow to the event pond.

Auxiliary equipment for the tailings thickening circuit will include:

- Tailings thickener electric rake drive.
- Tailings thickener feed metallurgical sampler.

17.2.10 Tailings Storage Facility Disposal

During periods of tailings thickener maintenance or paste plant maintenance or change to discharge location in the underground workings, the flotation tailings and paste plant flush / recycle slurry will be directed to the existing tailings storage facility. The supernatant water decanted from the tailings storage facility will be pumped by a TSF decant return pump to the process water tank.

17.2.11 Reagents and Consumables

The major reagents utilized within the process plant will include:

- Hydrated lime slurry for pH adjustment in grinding and flotation circuits.
- Blend 1 collector (Aero3477, Aero6697 and Aero5100 with mass ratio of 64:16:20) for froth flotation circuit to enhance hydrophobicity of the sulphide minerals.
- Dowfroth 250 or equivalent frother to provide a stable froth for froth flotation.
- Carboxymethyl cellulose (CMC) as depressant to minimize the recovery of gangue minerals.
- Flocculant for thickening.
- Fluxes for smelting.
- Diesel fuel for the gold smelting furnace.

- Grinding media.

Hydrated Lime Slurry

Hydrated lime will be delivered to site in 1,000 kg bulk bags. The bulk bags will be lifted, by the lime hoist, to the bulk bag splitter mounted above the lime bag hopper.

A rotary valve and screw feeder will deliver hydrated lime to a vortex mixer, where it will be contacted with process water and fed to the agitated lime mixing tank. Hydated lime at 20% solids (w/w) will be transferred from the lime mixing tank to a mechanically agitated storage tank by a dedicated lime transfer pump.

Hydrated lime slurry will then be circulated around the processing plant, via a ringmain, for dosing to the various locations around the grinding and flotation circuits.

The lime area will be serviced by a dedicated floor sump pump. Any spillage generated within this area will be pumped to the tailings thickener feed box.

Blend 1 Collector

Blend 1 collector will be used in the flotation circuit to enhance sulphide minerals' hydrophobicity. The Blend 1 collector is blend of Aero3477, Aero6697 and Aero5100 collectors with mass ratio of 64:16:20. Blend 1 collector will be delivered to the site in 1,000 L totes or intermediate bulk containers (IBC). The Blend 1 collector will be decanted by gravity from the totes or intermediate bulk containers to the collector storage tank. Metering pumps will distribute collector solution from the collector storage tank to the flotation circuit.

The collector storage area will be serviced by a dedicated air operated floor sump pump. Any spillage generated within this area will be pumped to the tailings thickener feed box.

Frother

The frother will promote bubble dispersion and provide a stable froth in the flotation cells. Frother liquid will be delivered to site in 1,000 L totes or intermediate bulk containers (IBC). The frother reagent will be decanted by gravity from the totes or intermediate bulk containers to the frother storage tank. Metering pumps will distribute frother solution from the frother storage tank to the flotation circuit.

The frother storage area will be serviced by a dedicated air operated floor sump pump. Any spillage generated within this area will be pumped to the concentrate leach tailings tank.

Carboxymethyl cellulose (CMC)

Carboxymethyl cellulose (CMC) will be used in floatation circuit as depressant to decrease hydrophobicity of the gangue minerals and decrease their recovery. CMC will be delivered to site in 1,000 kg bulk bags in powder form.

The bulk bag will be lifted, by the depressant area hoist, to the depressant bag breaker and hopper. CMC will be mixed in a proprietary mixing system, comprised of a bulk dry hopper, depressant blower, depressant wetting head and mixing tank. Raw water will be added to the depressant wetting head, to hydrate the CMC powder, prior to discharging into the agitated CMC mixing tank. The required storage concentration of CMC will be 2.0% (w/v). Upon completion of the mixing cycle, the CMC will be transferred to the depressant storage tank, by the depressant transfer pump. The depressant will be distributed from the storage tank to the required locations by dedicated dosing pumps.

The depressant area will be serviced by a dedicated floor sump pump. Any spillage generated within this area will be pumped to the tailings thickener feed box.

Flocculant

Flocculant powder will be delivered to site in 25 kg bags. Flocculant will be manually added to the flocculant plant storage hopper. Flocculant will be mixed in a proprietary mixing system, comprised of a bulk dry hopper, screw feeder, flocculant blower and mixing tank. The flocculant plant will mix flocculant powder with raw water to achieve the required stock solution concentration of 0.25% (w/w).

Flocculant will be withdrawn from the storage hopper by the flocculant screw feeder. The screw feeder will convey flocculant to the flocculant eductor, from which the flocculant powder will be pneumatically conveyed, to the flocculant mixer, by the flocculant blower. Raw water will be added to the mixer, to hydrate the flocculant powder, prior to discharging into the agitated flocculant mixing tank. Upon completion of the mixing cycle, the flocculant will be transferred to the flocculant storage tank, by the flocculant transfer pump.

The process plant layout means that there will be two flocculant mixing plants, one adjacent to the concentrate thickener, and the second one next to the tailings thickener. Flocculant will be distributed to the respective thickener by dedicated duty / standby dosing pumps from the respective storage tank.

Each flocculant area will be serviced by a dedicated floor sump pump. Any spillage generated within the concentrate and tailings flocculant areas will be pumped to the concentrate thickener and tailings thickener respectively.

Fluxes

The following fluxes will be delivered to the plant in 25 kg bags, and used in the gold room, viz Borax ($Na_2B_4O_2$), Sodium Nitrate (Nitre or $NaNO_3$), Soda Ash (Sodium Carbonate or Na_2CO_3), and Silica Sand (SiO_2).

Diesel Fuel

Diesel fuel will be delivered to the processing plant by a mobile mine diesel truck. The truck will have an on-board pump to transfer diesel fuel to the plant diesel tank.

Diesel fuel will be circulated around the processing plant via a ringmain to the gold smelting furnace in the goldroom.

The plant diesel area will be serviced by a dedicated floor sump pump. Any storm water collected within this area will be pumped to site drainage.

Grinding Media – SAG Mill

SAG mill grinding media, typically 100 mm diameter balls, will be delivered in 1 tonne bags and stored in the SAG mill ball concrete bunker. SAG mill grinding media will be added to the surge bin. SAG mill grinding balls will be loaded from the SAG mill ball bunker into the surge bin via an FEL.

The quantity of SAG mill grinding media to be added daily will be based on the SAG mill power draw and ore properties. SAG ball usage will be based upon delivery amounts and monthly inventories.

Grinding Media – Ball Mill

Ball mill grinding media, typically 50 mm diameter balls, will be delivered in 1 tonne bags and stored in the ball mill ball concrete bunker. Ball mill grinding media will be removed from the bulk storage bunker using an FEL and transferred to a ball charging kibble. The charged kibble will be transferred by the ball charging hoist for loading the ball mill.

The quantity of ball mill grinding media to be added daily will be based on the ball mill power draw and ore properties. Ball mill grinding media usage will be recorded based upon delivery amounts and monthly inventories. Daily usage rate will be based upon an average weight per kibble.

17.2.12 Water Services

The process plant will utilize raw water, filtered water, process water, gland water and potable water. Make-up water requirements for process water will be pumped from the raw water tank.

A storm water pond, which will hold storm water from area outside the processing plant and surroundings. The collected water will overflow to the nearby creek when necessary.

Raw Water

Raw water to the plant will be sourced from 800 portal underground water. The underground water will be transferred to the sediment pond for settling of the solids in water. For efficient settling of particles, the sediment pond will be constructed in two sections. The water will be pumped to the raw water tank via a dedicated decant pump.

Duty / standby raw water pumps will distribute raw water to the plant. Raw water will be used in the following areas:

- Filtered water treatment plant.

- Process water tank.
- Primary crushing area.
- Concentrate flocculant mixing.
- Tailings flocculant mixing.
- Depressant mixing.

Firewater will be supplied from the raw water storage tank, via a dedicated suction manifold. The firewater system will comprise:

- An electric jockey pump.
- An electric firewater pump.
- A diesel stand-by firewater pump.

The firewater system pressure will be maintained, by the jockey water pump. An electric firewater pump will automatically start on a drop in line pressure. The diesel firewater pump will automatically start if the line pressure continues to drop below the target supply pressure, which will occur when there is significant firewater demand or during a power failure.

Filtered Water

Filtered water from the raw water tank will be utilized for various locations in the plant. The raw water will be treated in a water treatment package that will produce suitable water and be stored in the filtered water tank. From the filtered water tank, filtered water will be distributed to the SAG and ball mills lubrication cooling water system, as well as for the on-line stream analyser by dedicated filtered water pumps.

Gland Water

Gland water for gland sealed centrifugal pumps will be pumped from the filtered water storage tank by dedicated gland water pumps to a distribution network that services all centrifugal pumps requiring gland water across the process plant.

Process Water

Concentrate and tailings thickener overflow solution will flow by gravity and be pumped respectively to the process water tank. Additionally, recovered supernatant liquid from the TSF will also be pumped as decant water to the process water tank. Finally, make-up water for the process water tank will be pumped from the raw water tank into the process water tank via the raw water reticulation system.

From the process water tank, the majority of the process water will be distributed through the plant via dedicated process water pumps. Process water will be mainly used for the following:

- Grinding circuit.
- Gravity circuit.
- Shaking tables.
- Flotation circuit.
- Concentrate and tailings thickener flocculant dilution.
- Concentrate filter.
- Hydrate lime mixing.
- Service points around the plant.

Potable Water

Potable water used around the processing plant will be sourced from the potable water source. The incoming potable water supply will be further treated by a vendor package treatment plant and will be stored in the potable water tank.

Potable water will be distributed for human consumption to buildings in the processing plant area, control rooms in the primary grinding and primary crushing areas, gold room, warehouse, laboratory, workshop, administration building, core shed, and contractors' area using dedicated potable water pumps. The potable water will also be used for the safety showers and eye wash stations throughout the plant. The safety showers are reticulated from a dedicated safety shower water tank via a ringmain and dedicated safety shower water pumps. If the water temperature in the safety shower water tank becomes too high, then the ringmain will be discharged to the raw water tank whilst the water in the safety shower water tank is refreshed with replacement potable water.

17.2.13 Air Services

Plant and Instrumentation Air

High pressure compressed air (700 kPag) will be generated for general use throughout the plant.

Two air compressors running in a lead-lag configuration will supply instrument and plant air to the process plant. All compressed air will be dried to ensure clean and low moisture air is delivered to all plant instruments and tools driven by compressed air. Dried air will be stored in an air receiver before being distributed throughout the plant.

Dedicated plant and instrument air distribution circuits within the process plant, combined with strategically located air receivers, will ensure sufficient compressed air is available during a power failure. In addition, the stored air in the receivers will allow all pneumatically actuated valves to be actuated to their fail-state and ensure sufficient air is available in high demand areas.

One dedicated vendor package including an air compressor and receiver will supply air for the concentrate filter. The main plant high pressure air circuit will provide emergency backup air for concentrate filter if required.

Flotation Low Pressure Air

Low pressure (LP) air will be supplied by one of two low pressure air centrifugal blowers. Low pressure air will be distributed to the flotation circuit.

17.3 Plant Consumption

17.3.1 Energy Consumption

The power for the processing plant, along with the rest of the site and camp, will be provided by grid power and fully backed up by on-site diesel-powered generators. The average power demand is summarized in Table 17.3.1 and utilized for the operating cost estimate. The average power demand does not reflect the instantaneous power demand for equipment start-up and electrical system capacity sizing.

Table 17.3.1 Average Power Demand Summary

Plant Areas	Installed Power (kW)	Average Continuous Draw (kW)	Total Annual Power Consumption (kW / year)
Crushing Circuit	384	220	1,931,305
Grinding (SAG mill and Ball mill)	3,400	2,636	23,088,000
Grinding and Classification (Excl. Mills)	642	276	2,416,223
Flotation Circuit	721	506	4,435,938
Concentrate Thickening and Filtration	80	43	373,057
Gold Room	102	35	303,616
Tailings Thickening and Handling	441	140	1,222,275
Reagents	212	78	680,969
Plant Water and Air Services	2,077	837	7,334,367
Plant Fuel Storage and Distribution	19	2	20,545
Total	**8,078**	**4,772**	**41,806,295**

17.3.2 Water Consumption

A water balance for the process plant has been completed. Water from the thickeners overflow streams is recycled within the process plant to reduce the external water requirement. Approximately 428 m^3/h of combined overflow water from the concentrate and tailings thickeners will be transferred back to the process water tank. In the normal operating condition, 1 m^3/h of raw water is required to make-up the water consumption for the process plant. At the time of paste plant flush, the required raw water make-up will increase to 320 m^3/h.

At the process plant, the raw water tank will provide a combined raw water and fire water reserve. The required firewater capacity will be stored below the level of the suction piping for the raw water pumps to ensure it is always available.

17.3.3 Reagent and Consumable Consumption

Reagent storage, mixing and pumping facilities will be provided for all reagents for the process plant. Table 17.3.2 provides a summary of wear liner consumables and reagents that are estimated to be used for the design plant throughput of 1.2 Mtpa as specified in the operating cost estimate.

Table 17.3.2 Annual Reagent and Consumable Consumption

Reagent / Consumable	Annual Consumption
Primary Jaw Crusher Fixed Jaw Liners	3.6 sets
Primary Jaw Crusher Swing Jaw Liners	2.4 sets
SAG Mill Liners	0.7 set
Ball Mill Liners	1.0 set
SAG Mill Grinding Media	367 t
Ball Mill Grinding Media	515 t
Hydrated Lime	2,400 t
Collector	102 t
Frother	18 t
Depressant	93 t
Flocculant	46.2 t
Borax	0.1 t
Sodium Nitrate (Nitre)	0.02 t
Sodium Carbonate (Soda Ash)	0.02 t
Silica Sand	0.1 t
Smelting Furnace Crucibles	31 each
Diesel Fuel	727 kL

17.4 Plant Control System

The instrumentation and control system will be designed with the following objectives in mind:

- Ease of operation - operator intervention and remedial action will be easily accomplished.
- Interlocking - the control system will provide process interlocking between individual drives and plant areas.
- Reliability - all field equipment will be selected on its ability to operate in an environment where extreme temperatures and dusty conditions are encountered.
- Ease of maintenance - instrumentation and control systems will be designed and installed such that maintenance requirements are minimized.

The plant control system will be designed around a moderate level of automation and monitoring.

The starting and stopping of most electrical drives and actuated valves will be performed from the plant control system (PCS) operator interface terminals (OITs). There will be no Remote / Local selector switches in the field. However, field operators may start equipment local to the drive if the control room has selected Local at the OIT. Local mushroom head style Lock-Off-Stop stations will be installed which will be hard-wired to the drive starter circuit to enable local stop / emergency stop regardless of drive mode selection.

In general, the use of actuated isolation or control valves will be implemented around the plant for automatic control loops or sequencing as part of the plant control or the elution sequence. All actuated valves and control valves will be operated from the OITs with remote position indication available. Automatic control valves will be controlled by PID loops within the PCS.

Equipment in the plant will be interfaced to the PCS via the Input / Output (I/O) modules, which are positioned in the field and remote to the main PCS hardware. Vendor equipment packages throughout the plant will utilize the PCS for control and monitoring of their respective equipment.

All fixed speed drives are controlled using network connected motor protection relays. The current draw of most fixed speed drives will be displayed on a number of summary mimic pages on the PCS and adjacent to the drive mimic where appropriate. Similarly, the current draw of all variable speed drives will be displayed on several summary mimic pages on the PCS and adjacent to the drive mimic where appropriate.

The PCS will perform all digital and analogue control functions, including PID control. Faceplates on the PCS displays will facilitate the entry of set points, readout of process variables (PV's) and controlled variables (CV's) and entry of the three PID parameters (Proportional, Integral and Derivative).

Most equipment interlocks will be software configurable. However, selected drives will be hard wired to provide the required level of personal safety protection, e.g. the emergency stop buttons associated with each and every motor and the pull wire switches associated with conveyors.

All alarm and trip circuits from field or local panel mounted contacts will be based on fail-safe activation. Alarm and trip contacts will open on abnormal or fault condition. If equipment shutdown occurs due to loss of mains power supply, the equipment will return to a de-energized state and will not automatically restart upon restoration of power.

Sequential group starts and sequential group stops will not be incorporated with non-packaged plant equipment. However, in any sequential process, critical safety and equipment protection interlocks will cause a cascade stop in the event of interlocked downstream equipment stopping (e.g. trip of mill feed conveyor will result in stop of stockpile reclaim apron feeder).

The process plant will be provided with one main control room.

The following Operator Interface Terminals (OIT) will be provided in the control room for operator use.

- Main plant control room (2 OITs).

Additional OITs will be provided for the PCS backup, historical records, controls engineering and remote viewer stations.

18.0 PROJECT INFRASTRUCTURE

18.1 Power

The supply of electric power to the project is via grid connection to the PNG Power Ltd (PPL) network, 22kV overhead line (OHL).

A diesel back-up power station forms part of the study. Generators that are located on the existing power system at the underground and camp will operate independently as satellite systems.

A new diesel back-up power station has been sized based on the maximum demand requirement of 8.8MW servicing the process and paste plant, with no allowance for redundancy. The diesel generators have a continuous power output of 1.6MW which includes an altitude derating of 10%. This requires a total of 6 generator units to meet the maximum demand of 8.8MW continuously. Each generator has an 11kV alternator directly connected to the 11kV distribution system and the power station includes an auxiliary transformer and earthing transformer. 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 22kV supply have been included in the overall capital cost estimate for the study. 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 22kV switchgear and transformers are included as part of the capital estimate.

Table 18.1.1 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 gensets for independent/satellite power back-up
1.2 Mtpa Mill	7,973	5,606	6,378	
Paste Plant	3,596	2,267	2,424	Peak demand will double towards the end of the mine life. Genset upgrade to include all paste plant power requirements
Mine	3,226	1,191	2,581	Utilize existing site gensets for independent/satellite power back-up
Total	**16,295**	**10,064**	**12,484**	
		Total Peak Power	**12,484**	
		Genset Peak Power	**8,803**	Only Paste plant infrastructure and 1.2 Mtpa Mill on the Gen-set upgrade
		New Gen-sets Required	**6**	1600kVA/11kV gen-sets

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.

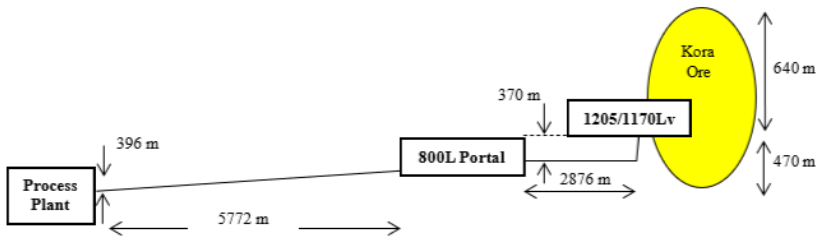
- New fixed plant maintenance workshop and crib room.
- New warehouse.
- Extension to the existing admin office.

18.4 Paste Fill Plant Project Infrastructure

18.4.1 Paste Plant Design Criteria

Section 13.14 presented the overall paste system design criteria. To achieve these criteria, the Kora paste system is to include three different infrastructure sites (Process Area, 800 Portal Area and 1205/1170 Area), which are connected using high pressure paste pipelines. The proximity of these three different sites, to one another, is illustrated in the schematic shown in Figure 18.4.1.

Figure 18.4.1 Schematic Long Section Showing Kainantu Layout and Kora Paste System Profile



To satisfy the system design criteria component design criteria set out in Table 18.4.1, Table 18.4.2, Table 18.4.3, is necessary. Table 18.4.1, Table 18.4.2 and Table 18.4.3 set out the paste plant design criteria at the Process area, 800 Portal area and 1205/1170 Level area sites, respectively.

Table 18.4.1 Kora Paste System General Design Criteria – Process Area

Process Area Infrastructure		
Description	**Units**	**Quantity**
Thickener UF/Micer & Filter Feed Pump		
Flowrate	m^3/hr	150
Power	kW	37
Vacuum Filter		
Filtration Rate	kg/m^2/hr	511
Feed Pulp Density	% solids	59%
Floculation		
Nominal	g/t	15
Design	g/t	100
Floculant makeup rate	L/hr	2800
Filter Performance		
Cake Moisture	%	24%
Cake Production	tph (dry)	85
Water in Filter Cake	m^3/hr	27
Filter Cake	tph (wet)	111
Water in Filtrate	m^3/hr	66
Vacuum Pump		
Quantity		1
Flowrate	m^3/hr	10000
Power	kW	355
Paste Mixing Tank		
Live Capacity	m^3	200
Agitator		
Motor Size	kW	37
Piston Diaphram - Feed Pump		
Flowrate	m^3/hr	123
Power	kW	37
Paste Pump		
Design Flowrate	m^3/hr	123
Design Discharge Pressure	kPa	20,000
Electric Power	kW	800
Potable Water Supply		
Potable Water Usage	m^3/hr	5
Seal Water Recycle Tank		
Volume	m^3	2.5
Seal Water Flows	m^3/hr	36
Description	**Units**	**Quantity**
Cooling Tower		
Entry Temperature	◦C	40
Exit Temperature	◦C	30
Flowrate	L/min	600
Raw Water		
Raw Water Usage	m^3/hr	9
Process Water		
Operational Process Water Usage	m^3/hr	40
Flush Process Water Usage (from Process Circuit)	m^3/hr	150
Filtrate/Spillage Discharge Hopper		
Live Volume	m^3	2
Waste flow rate	m^3/hr	70
Plant Compressed Air Requirements		
Power	kW	90.0
Supply Pressure	kPa	800
Instrument Air Receiver	m^3	1
Power Supply		
HV	V	11,000
3 Phase	V	415
Single Phase	V	240
Transformer Size Reqd. (by others)	kVa	2000
Generator Backup		
Supply		Sitewide backup genset
Backup		Mix Tank Agitator/Safety Shower
Sump Pump (filter area)		
Flowrate	m3/hr	33
Power	kW	5.5
Sump Pump (floculant area)		
Flowrate	m3/hr	33
Power	kW	5.5
Control Room + Kitchenette & Laboratory		
Control Room arrangement		Proces Plant Control Room
Laboratory arrangement		Process Plant Laboratory

Table 18.4.2 Kora Paste System General Design Criteria – 800 Portal Area

800 Portal Infrastructure		
Description	**Units**	**Quantity**
Paste Storage Tank		
Live Capacity	m^3	200
Tailings Storage Tank Agitator		
Motor Size	kW	18
Waste Water Storage Tank Agitator		
Motor Size	kW	18
Vortex Mixer Feed Pump		
Flowrate	m^3/hr	123
Power	kW	22
Slag Mixing Tank Agitator		
Agitator Motor	kW	2.2
Piston Diaphram Feed Pump		
Flowrate	m3/hr	123
Power	kW	22
Paste Pump		
Design Flowrate	m^3/hr	123
Design Discharge Pressure	kPa	20000
Electrical Power	kW	1000
Waste Tank Discharge Pump (duty)		
Flowrate	m^3/hr	133
Power	kW	110
Potable Water Supply		
Volume	m^3	2
Potable Water Usage	m3/hr	5
Raw Water Tank		
Quantity		5
Volume	L	30000
Raw Water Usage	m^3/hr	150 m3/hr (on flush)
Plant Air Compressor		
Source		Mine Air
Supply Pressure	kPa	800
Instrument Air Receiver	m^3	1
No of Compressors		0
Description	**Units**	**Quantity**
Emergency Flush Water Pump		
Design Flowrate	m3/hr	20
Design discharge pressure	kPa	5000
Power Demand	kW	45
Sump Pump		
Flowrate	m3/hr	33
Power Supply		
HV	V	11,000
3 Phase	V	415
Single Phase	V	240
Transformer Size (by others)	kVA	1500
Backup Power		
Source		Sitewide backup power
Size	kVA	200
Backup		Agitator Tanks/High Pressure flush/High flow Flush/Safety Shower
Binder Delivery, Storage & Dosing		
Maximum Dry Slag Addition	tph	15
Delivery Form		20 Ft Shipping Containers
Transfer to Bulk/Duty Silo		Dense phase conveying
Transfer Rate	tph	25
Bulk Storage Silos		
No. of Silos		1
Capacity Silo 1	m3	300
Storage Capacity at average production	hr	53
Binder Metering System		Continious Corriolus metering
Silo - Paste Transfer System		Screw conveyors
Maximum Metering Range	tph	20

Table 18.4.3 Kora Paste System General Design Criteria – 1205/1170 Level Area

1205/1170 Lv - Portal Infrastructure		
Description	**Units**	**Quantity**
Paste Mixer		
Effective Working Volume	m^3	4
Nominal Residence Time	mins	2.0
Potable Water Supply		
Volume	m^3	2
Potable Water Usage	m^3/hr	5
Mine water storage capacity 1245 Level		
Quantity		5
Volume	m^3	30,000L (tanks)
Mine water storage capacity 1285 Level		
Quantity		5
Volume	m^3	30,000L (tanks)
Binder Delivery, Storage & Dosing		
Dry Binder Addition	tph	3
Delivery Form		ISO Container
Number	units	10
Payload	t	28
Transfer Rate	tph	25
Bulk Storage Silos		
No. of Silos		1
Silo Type		Horizontal
Capacity Silo 1	t	50
Storage Capacity at average production	hr	30
Binder Metering System		Loss in Weight
Maximum Metering Range	tph	2.2
Paste Pump		
Quantity		1
Design Flowrate	m^3/hr	123
Design Discharge Pressure	kPa	15000
Electric Power	kW	2 x 380
Emergency Flush Water Pump		
Design Flowrate	m3/hr	20
Design discharge pressure	kPa	5000
Power Demand	kW	45
Plant Air		
Pressure Supply (by others)		Mine Air System
Instrument Air Receiver	m3	1
Flush Air Receiver	m3	14
No of Compressors		0
Power Supply		
HV	V	11,000
3 Phase	V	415
Single Phase	V	240
Transformer Size (by others)	kVA	2500
Generator Backup		
Size	kVA	200
Emergency Power		High Pressure Flush Pump/High Pressure cleanup hose
Control Room + Kitchenette & Laboratory		
Control Room arrangement		20 ft container
Laboratory arrangement		20 ft container

18.4.2 Real Estate and Site Layouts

To overcome the limited real estate and challenging terrain at the Kora mine, the following site locations and layouts were selected.

Process Plant

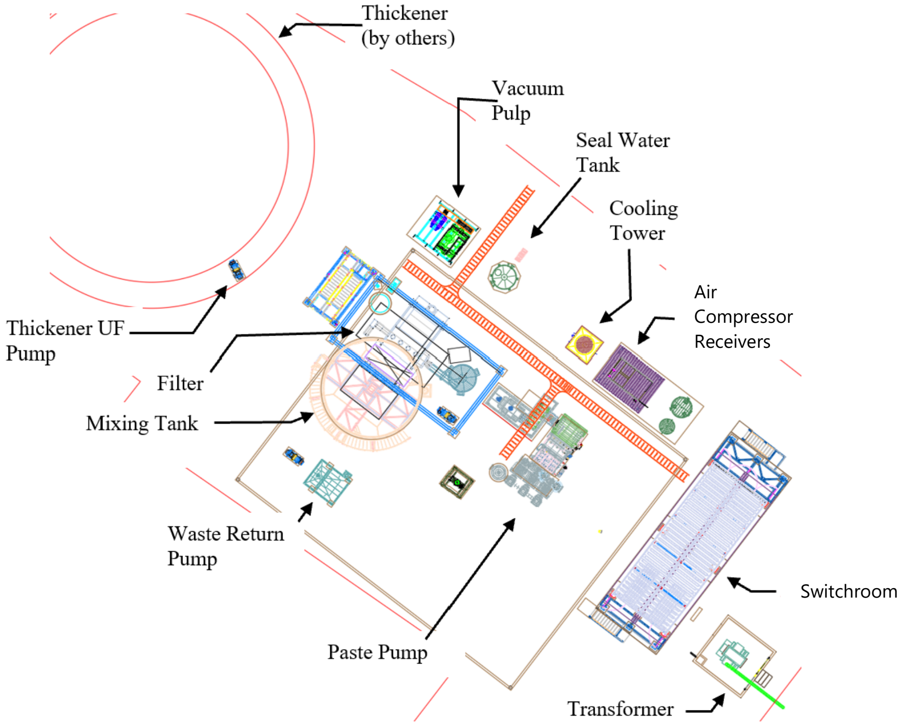
At the Kainantu process plant area, K92 have allocated a region to the North-West of the existing process plant infrastructure for the paste plant. This region is outlined in yellow in Figure 18.4.2. The region allocated at the process plant is understood to be a relatively flat rectangular in area with dimensions of 25 m x 60 m. Also presented in the yellow area (as a blue circle) is the proposed location of the tailings thickener. This thickener is outside the scope of the paste plant but is the source of feed tailings into the paste system and is the outflow battery limit for wastewater. In addition, immediately adjacent to the tailings thickener is a final tailings box to be installed. This tailings box is to be used for disposal of surplus tailings, generated while bringing the paste quality up to the target specification, and while flushing the system.

The large area of stable ground at the process plant location, combined with the convenience of its proximity to the process plant was an important consideration in consolidating the paste dewatering and mixing infrastructure at this location.

Figure 18.4.2 Allocated Paste Plant Location Relative to Existing Process Plant

The consolidation of infrastructure at the Process Area is presented in the plan view showing major equipment presented in Figure 18.4.3.

Figure 18.4.3 Plan View Showing Major Equipment Arrangement at the Process Area



800 Portal

At the 800 Portal area, available real estate is limited, and most flat areas consist of un-engineered fill against relatively steeply dipping natural gradients. An image showing the overall 800 Portal location is presented in Figure 18.4.3. At this location, several possible sites were considered (i.e., sites numbered 1-5 in Figure 18.4.3). Due to interactions with other activities on the site and suitability of the bearing capacity in the different locations, the yellow triangular shape indicated by the Number 2 was selected for the proposed 800 Portal area paste infrastructure.

Figure 18.4.4 800 Portal Location Backfill Infrastructure Site (Designated #2)

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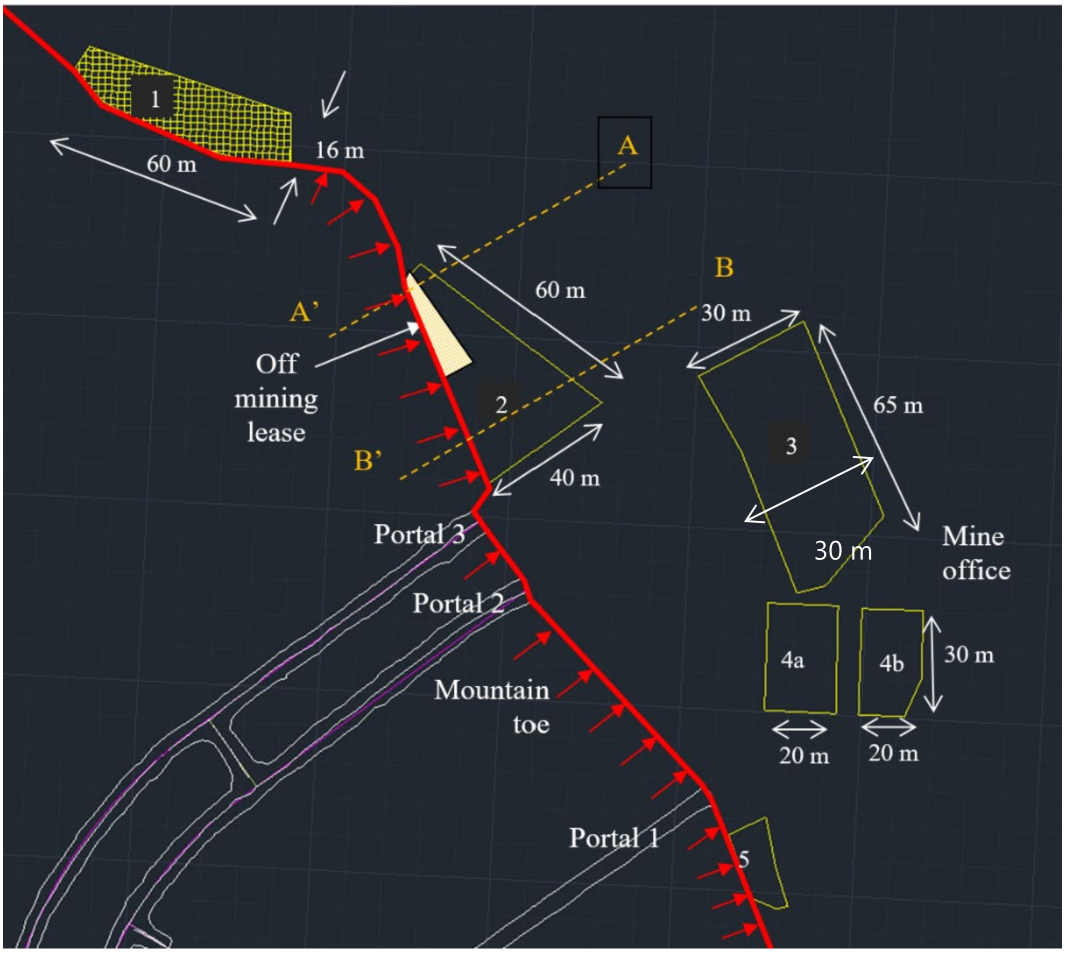
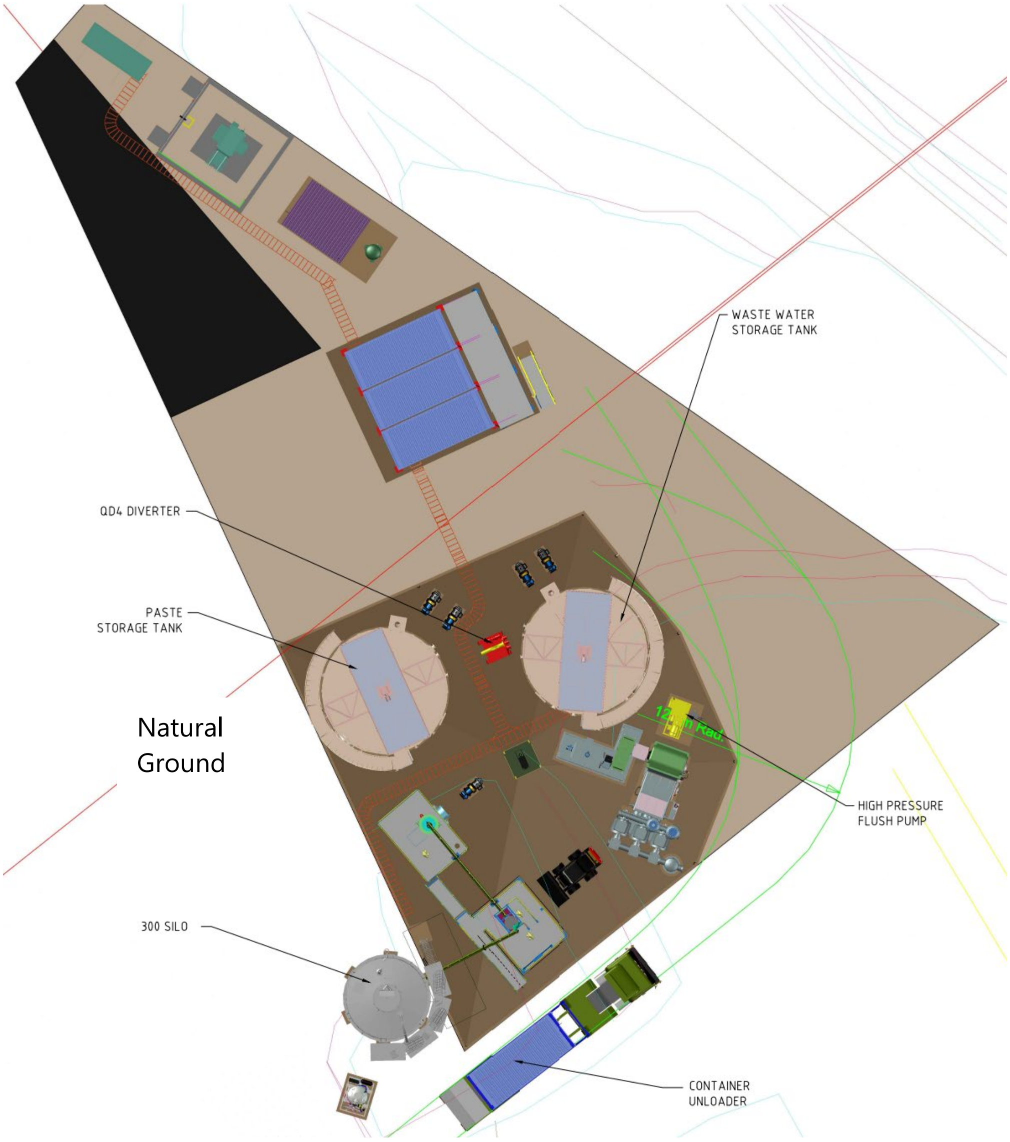


The major challenges associated with Site #2 is the relatively small area available, the unfavourable geotechnical ground conditions and the need to incorporate a Slag silo and 2 x 200 m^3 tailings and flush water storage tanks. This challenge is to be overcome by locating the heavy infrastructure (Binder Silo, and Storage tanks up against the natural ground). This is illustrated in the plan view presented in Figure 18.4.5.

Figure 18.4.5 Plan View of 800 Portal Site Layout

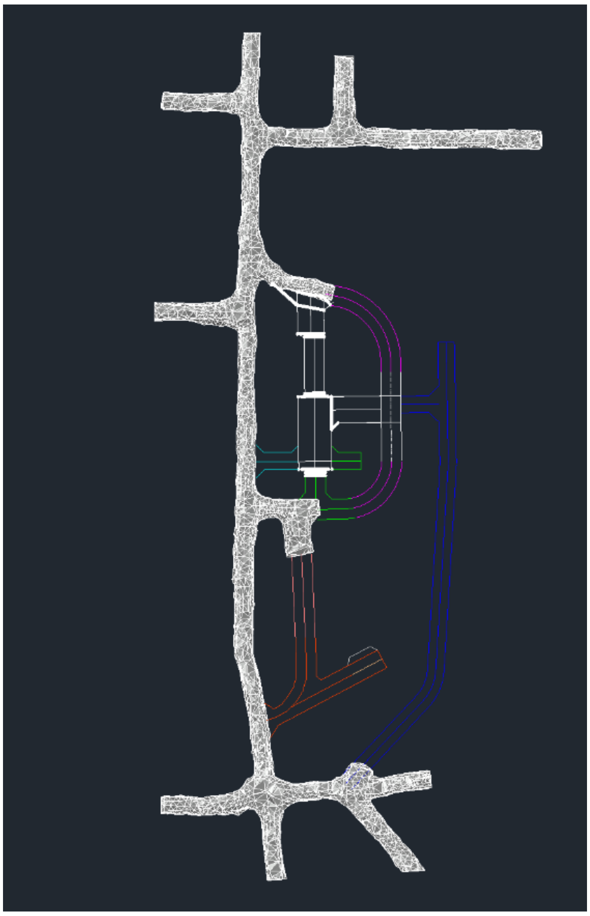


Underground Infrastructure

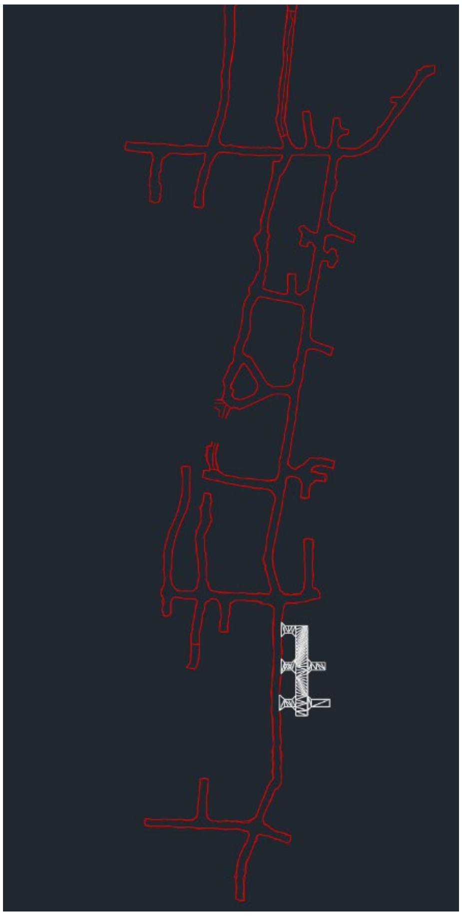
The Kora orebody extends over 1,100 m vertically and, given that the tailings feed is delivered from the lower levels within the mine, it is ideal to position the underground transfer pump on a level that allows single stage pumping to all extremities of the mine and permits gravity driven flows to some lower sections of the orebody. To facilitate transport of paste to the extremities, analysis showed that the underground transfer station must be located close to elevation 1,180 m. As part of the selection process, several locations were considered and, in conjunction with K92, it was decided to position the paste mixing infrastructure on the 1205 Level and gravity feed paste to the 1170 Level, where a pump station would be located.

Plan views of the existing 1205 and 1170 level infrastructure and proposed development are presented in Figure 18.4.6a and b, respectively. Figure 18.4.6a shows the existing development as a white hatched drawing, while Figure 18.4.6b shows the existing development as a red outline.

Figure 18.4.6 Proposed Underground Infrastructure Locations on the (a) 1205 Level and (b) 1170 Level



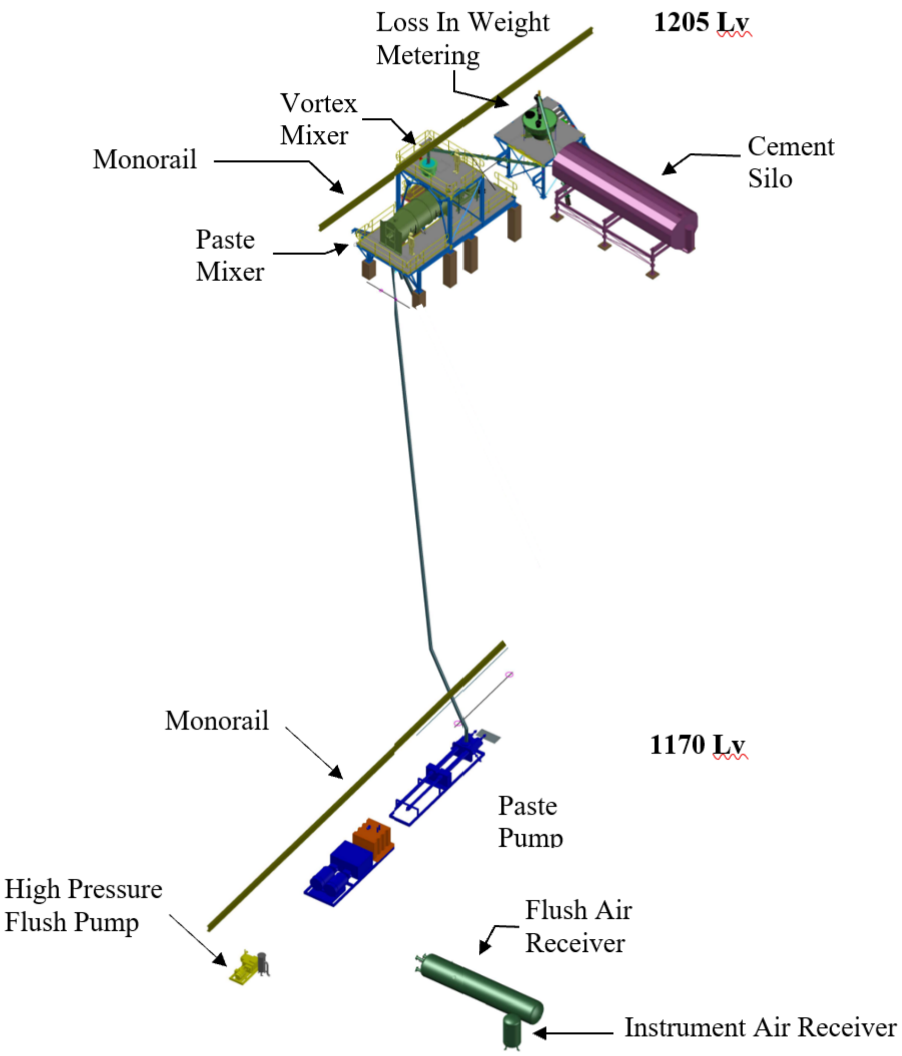
(a)



(b)

The nominated location on the 1205 Level is immediately above that on the 1170 Level, and this location was selected to permit the gravity flow from the (1205 Level) paste mixer into the (1170 Level) paste pumps. This is illustrated in the underground layout isometric drawings presented in Figure 18.4.7. Positioning of the infrastructure permits the paste pumps to be fed with a high net positive suction head, which is important for prolonging the life of pump valves and seals (i.e. minimising maintenance downtime).

Figure 18.4.7 Schematic of Proposed Arrangement on 1205 and 1170 m Lv

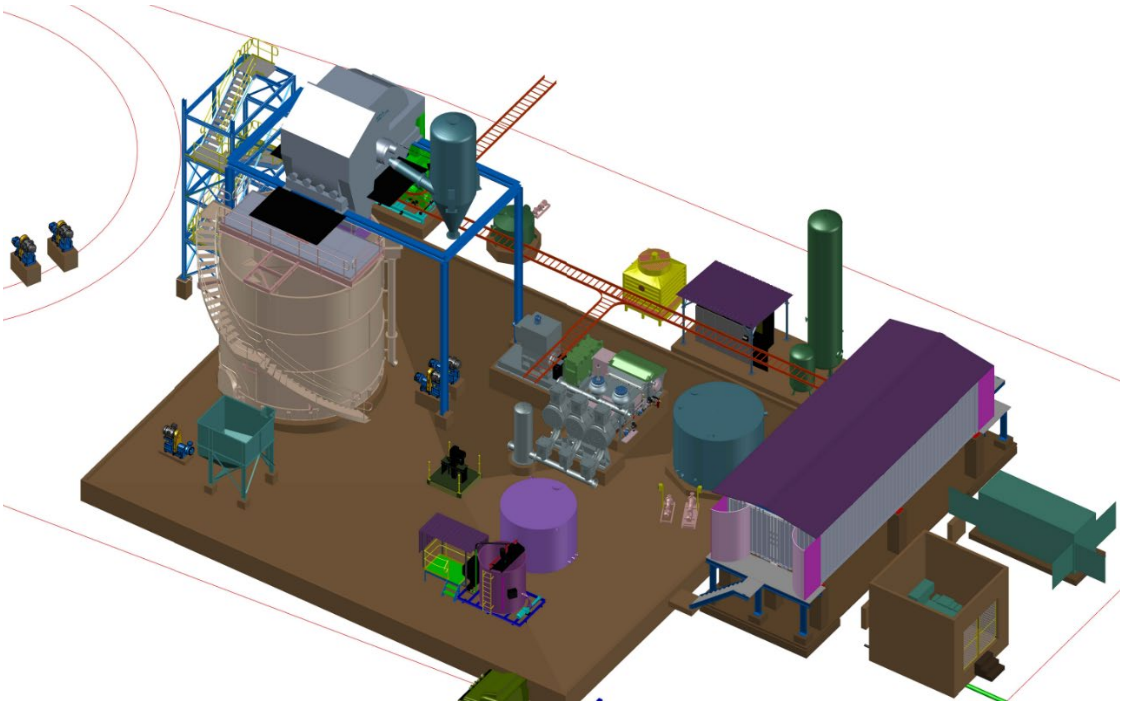


18.4.3 Major Equipment Specification

Process Area – Major Equipment

An isometric drawings figure showing the paste plant infrastructure at the Process Plant area is presented in Figure 18.4.8.

Figure 18.4.8 Isometric Views of the Paste Plant Infrastructure at the Process Area



The objective of the infrastructure located at the process plant area is to dewater thickener underflow (from 59% solids) to produce a paste-tailings product that has a low, but tangible, yield stress of approximately 50 Pa. Consistent production of paste at this yield stress is critical for successful performance of the Kora hydraulic transport system, because excessively higher yield stress paste would overload the paste pump capacity, while excessively low yield stress paste may lead to pipeline settlement issues and issues with 'in-stope' water accumulation (i.e., paste slumping and ponding).

To address this, the Kora paste process is structured to dewater tailings to a solids content slightly higher than the target value and use dilution water (added at the outlet of the paste mixing tank) to regulate the paste solids content. Paste density and pipe friction loss measurements immediately downstream of the piston diaphragm feed pump (which would provide feedback when both rejecting out of specification paste, and during operation) would be used on a feedback loop with the dilution water addition, to control paste solution and ensure a consistent product was generated.

The paste dewatering, mixing and dilution control methodologies adopted for the Kora paste system have previously proven successful at other operations in the past.

The major infrastructure and load list for equipment included at this site is presented in Table 18.4.4.

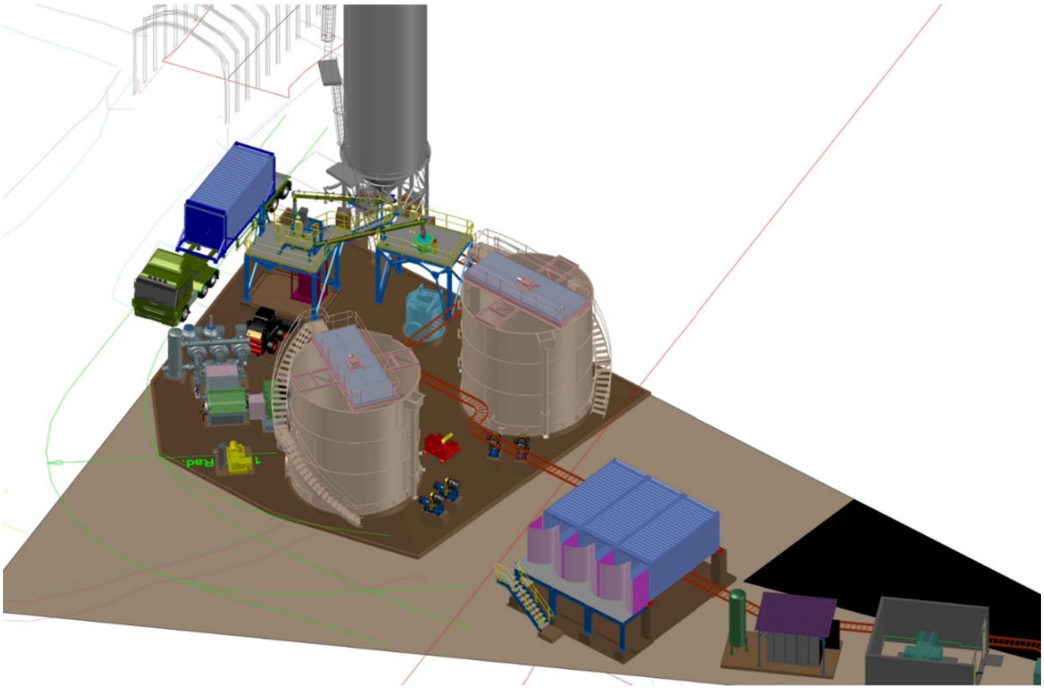
Table 18.4.4 Major Equipment List for Process Area Paste Infrastructure

Qty	EQ description	Operating Volume / Capacity m3/h	Total Volume m3	Installed Power (total) kW	VSD	Emergency Power Requirement (kW)
	PR01 - Thickener Island					
1	Thickener U/F - Filter & Mixing Tank Feed Pur	150m3/h @35m TBC		37	VSD	
1	Floc Dilution pump			2.2	VSD	
	PR02 - Tailings Filtraton & Mixing					
1	LV Electrical Switchroom					
1	Tailings Mixing Tank Agitator			37.0	DOL	37.0
1	Tailings Filter No.1			7.5	VSD	
1	Tailings Filter No.1 Vacuum Pump	10000 m3/hr		250.0	S/S	
1	Filtrate & Spillage Return Pump (Duty)	70m3/h @10m TBC		15	VSD	
1	Floc Dosing Pump			0.8	VSD	
1	Tailings Filter No.1 Filtrate Receiver					
1	Tailings Filter No.1 Snap Air Receiver					
1	Plant Air Compressor			90	DOL	
1	Tailings Storage/ Mixing Tank	200.00				
1	Filtrate & Spillage Return Tank	4 m3 Live				
1	Seal Water Recycle Tank	2.45				
1	Seal Water Recycle Pump					
1	Cooling Tower	600 l/s		3.0	DOL	
1	Filter Area Sump Pump	33m3/hr		5.5	DOL	
1	Cooling Water Circulation Pump	36 m3/hr		7.5	VSD	
1	Hoseup water Pump			0.6	DOL	
1	Filter Area Safety Shower	4.44				
	PR03 - Backfill Distribution (incl. Pumping)					
1	Paste Pump	130m3/h @ 20 MPa		600.0	DOL	
1	Gearbox Oil Cooling fan			3.0	DOL	
1	Lub/Propelling liquid pump			0.8	DOL	
1	PD Feed Pump	144m3/h @9m TBC		22.0	VSD	22.0
5	Cement ISO Container	20T				

800 Portal – Major Equipment

An isometric figure showing the paste plant infrastructure to be located at the 800 Portal area is presented in Figure 18.4.9.

Figure 18.4.9 Isometric of Paste System Infrastructure at the 800 Portal



The objective of this infrastructure is tailings storage and transfer as well as the addition of Ground Granulated Blast Furnace Slag (Slag).

The hydraulic profile demands that a second stage paste pump is necessary at the 800 Portal. While it may be possible to feed directly into the transfer pump, the 800 Portal location also provides a convenient location to add slag and also a convenient location for flushing the various sections of the system. To facilitate slag storage and addition a slag storage silo and metering system is to be included. This system provides continuous metering of the slag addition on a control loop back to the screw feeder, as well as a slag addition 'check-weigher' system that allows the continuous metering instrumentation to be calibrated and verified.

Two 200 m^3 agitated slurry storage tanks are also included at the 800 Portal, both equipped with backup power. One of the agitated tanks is to be used for storage of '*in-spec*' paste, while the second tank is to be used for storing low density paste and flush water (waste), created when 'back flushing' from the 1205 level (underground). Slurry in the waste tank is to be retained until the line between the Process area and 800 Portal is available for transporting that slurry back to the process plant (and subsequently the final tailings hopper) for disposal. The bunded area containing the 800 Portal infrastructure is designed to have a 220 m^3 storage capacity to ensure that the entire capacity of any tank can be contained. The major infrastructure used for this application and load list is presented in Table 18.4.5. This infrastructure is included in the 800 Portal capital cost estimate.

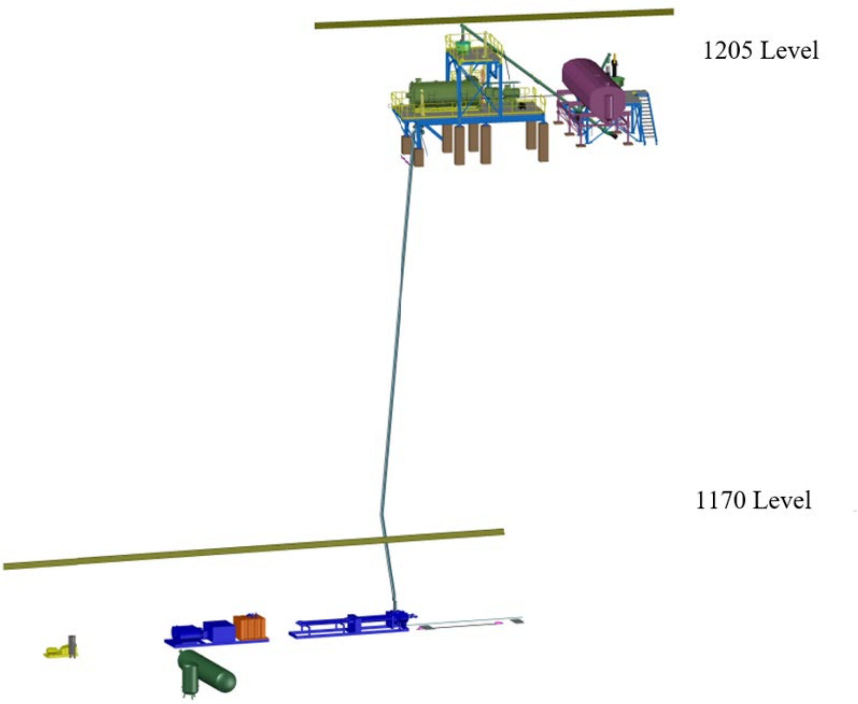
Table 18.4.5 Major Equipment List for 800 Portal Paste Infrastructure

Qty	EQ description	Operating Volume / Capacity m3/h	Total Volume m3	Installed Power (total) kW	VSD	Emergency Power Requirement (kW)
	P001 - Utilities (Auxiliary Systems)					
1	LV Electrical Switchroom					
1	Sump Pump			5.5		
1	Raw Water Pump	Max Flowrate - 48 m3/hr		11	VSD	
1	Potable Water Pump	Max Flowrate - 15 m3/hr		5.5	VSD	
1	Gland Water Pump	5 m3/hr		0.6	DOL	
1	Instrument Air Receiver		1.00			
1	Slag System Air Receiver		1.00			
1	Slag System Drier					
1	Binder Silo Fill Area Safety Shower	4.44				
1	Uninterruptible Power Supply					
5	Raw Water Tank		30.00			
1	Potable Water Tank		2.00			
	P002 - Tailings Storage					
1	Tailings Storage Tank Agitator	6.5 m x 6.5 m	200.00	18.5	DOL	18.5
1	Flush Water Return Pump (Duty)	Max Flowrate - 100 m3/hr @ 80 m head		110.0	VSD	
1	Tailings Storage Tank	6.5 m x 6.5 m	200.00			
1	Return Water Tank	6.5 m x 6.5 m	200.00			
1	Return Water Tank Agitator	6.5 m x 6.5 m	200.00	18.5	DOL	18.5
1	Vortex Mixer Feed PUMP	141 m3/hr @ 6m		15.0	VSD	
1	Slag Mixing Tank Agitator	2 m x 2 m	4.00	2.2	DOL	
	P003 - Backfill Distribution (incl. Pumping)					
1	Paste Pump	123m3/h @ 20 MPa		600.0	VSD	
1	Gearbox Oil Cooling fan			3.0	DOL	
1	Lub/Propelling liquid pump			0.8	DOL	
1	PD Feed Pump	Max Flowrate - 141m3/hr @ 15 m head		18.5	VSD	
	P004 - Slag Storage & Metering					
1	Slag Binder Silo Bulk Storage	300 m3				
1	SlagSilo Bulk Storage Unlaod POD					
1	Binder Mass Flow Meter	20 t/hr		1.1	DOL	
1	Binder Silo Discharge Screw Feeder Drive Cooling Fan			0.3	DOL	
1	Binder Silo Discharge Screw Feeder No.1			11.0	VSD	
1	Vortex Mixer Binder Screw Feeder			11.0	VSD	
1	Binder Silo Dust Filter				-	
1	Vortex Mixer	1.3 m diameter				
1	Binder Silo Fill Area Safety Shower	4.44				
1	Shipping Container Tilter					

1205/1170 Level – Major Equipment

An isometric figure showing the paste plant infrastructure to be located at the 1205/1170 underground chambers is presented in Figure 18.4.10.

Figure 18.4.10 Isometric of Paste System Infrastructure at the 1170/1205 Level Area



The objective of this infrastructure is the addition and thorough mixing of cement throughout the paste to ensure a consistent paste product, as well as hydraulic distribution of paste to all relevant areas of the Kora orebody. Slag-paste is received at the 1205 level and this material is fed directly into the paste mixer, with a portion of the stream entering the paste mixer via the vortex mixer. Cement is delivered underground in 20 t ISO containers where it is blown into a horizontal 50 t silo. From the 50 t horizontal silo, cement is metered into the paste mixer using a low in weight (LIW) system. Given the expected cement addition rate, of 1.5 tph, the 50 t silo storage capacity permits ongoing cement addition for 1.4 days, in the case of any delays supplying cement underground.

From the 1205 level, mixed paste flows under gravity to the 1170 where it is fed into the hydraulic piston pump or under gravity to stopes in suitable locations beneath the 1170 level. 50 m^3 paste sumps are also included on 1205 and 1170 level to dump cemented paste in the case of out of specification paste or issues downstream. This geometry was selected to ensure that the hydraulic piston pump is provided with sufficient net positive suction head, which is important to minimize pump maintenance. The major infrastructure used for this application with load list is presented in

Table 18.4.6. This infrastructure is included in the 1205/1170 capital cost estimate.

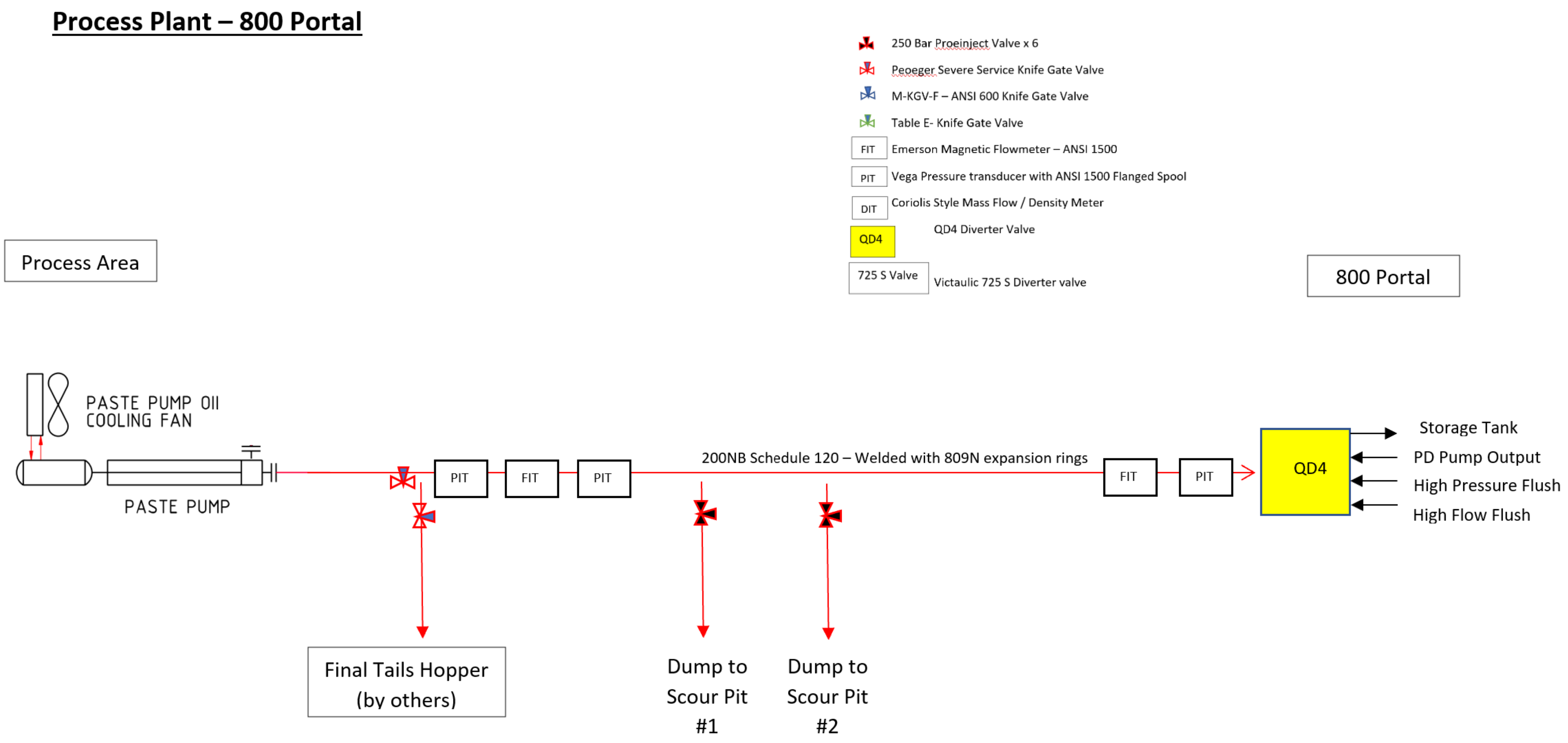
Table 18.4.6 Major Equipment List for 800 Portal Paste Infrastructure

Qty	EQ description	Operating Volume / Capacity m3/h	Total Volume m3	Installed Power (total) kW	VSD	Emergency Power Requirement (kW)
UG01 - Utilities (Auxiliary Systems)						
1	Control Room / Ablution					
1	Paste Laboratory					
1	LV Electrical Switchroom					
1	Plant Air Dryer					
1	Essential Power Generator					
1	Instrument Air Receiver		1.00			
1	Binder Silo Fill Area Safety Shower	4.44				
1	Paste Mixer Eyewash	0.09				
5	1245 Lv - Raw Water Tank		30.00			
5	1285 Lv - Raw Water Tank		30.00			
1	Potable Water Tank		2.00			
1	Uninterruptible Power Supply					
1	Flush Air Reciever		14.00			
UG02 - Tailings & Binder Mixing						
1	Paste Mixer			160.0	SS	
1	Vortex Mixer					
UG03 - Binder Storage & Metering						
1	Binder Silo Unloading Operator Panel					
1	Binder Silo		50.00			
1	Binder LIW Hopper					
1	Binder transfer Blower			75.0	DOL	
1	Vortex Mixer Binder Screw Feeder Drive Cooling Fan			0.3	DOL	
1	Binder Silo Discharge Screw Feeder			22.0	DOL	
1	Vortex Mixer Binder Screw Feeder			4.0	VSD	
1	Binder Silo Dust Filter			-	-	
1	Binder LIW / Surge Hopper Dust Filter					
1	LIW Hopper Binder Screw Feeder			4.0	DOL	
UG04 - Backfill Distribution (incl. Pumping)						
1	Paste Pump	123m3/h @ 16 MPa		760.0	SS	
1	Hydraulic Booster Pump			15.00	DOL	
1	Hydraulic Auxiliary Booster Pump			11.00	DOL	
1	Oil Cooling Fan			11.00	DOL	
1	Emergency Flush Water Pump	20.0		45.0	DOL	45.0
1	Emergency Flush Water Pump Oil Circulation Pump			1.5	DOL	1.5
1	Emergency Flush Water Blowdown Tank					
1	Monorail	22 m travel		5.5	DOL	
1	Monorail	30 m travel		5.5	DOL	

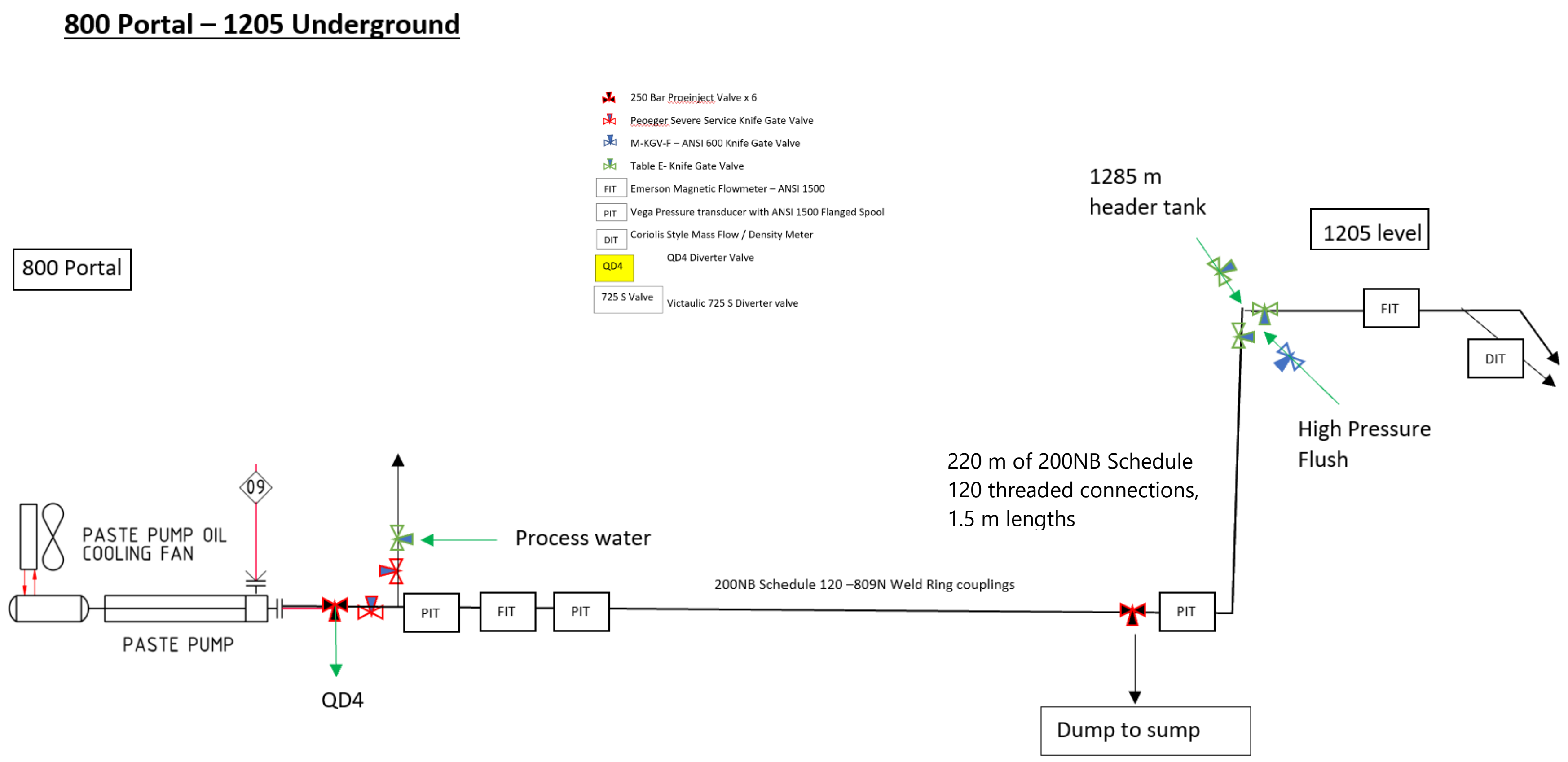
Paste Hydraulic Transport Systems

The paste plant infrastructure at the various locations is to be linked using a series of high pressure paste pipelines. Schematic Piping and Instrumentation Diagram's (P&ID's) showing the proposed overland pipeline, instrumentation and valve arrangements between the Process Area and 800 Portal Area, the 800 Portal to 1205 level underground piping and the paste reticulation piping (included in the capital cost estimate) downstream of the 1170 are presented in Figure 18.4.11a, b and c, respectively

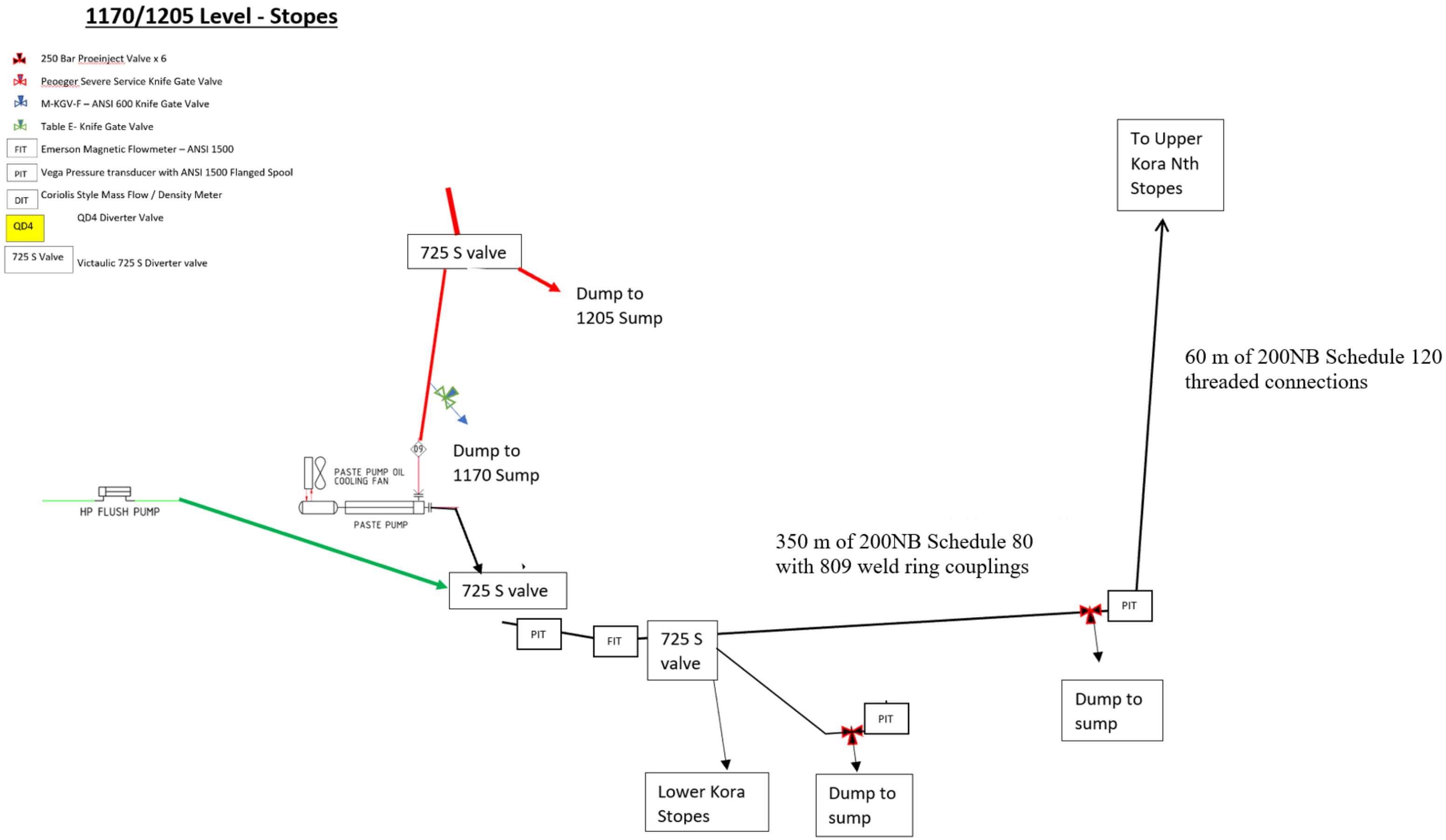
Figure 18.4.11 P&IDs for Paste System Delivery Pipelines for (a) Process to 800 Portal, (b) 800 Portal to 1205 Level and (c) 1170 Level to Stope



(a)


800 Portal – 1205 Underground
250 Bar Proeinject Valve x 6
Peoeger Severe Service Knife Gate Valve
M-KGV-F – ANSI 600 Knife Gate Valve
Table E- Knife Gate Valve
FIT
Emerson Magnetic Flowmeter – ANSI 1500
PIT
Vega Pressure transducer with ANSI 1500 Flanged Spool
DIT
Coriolis Style Mass Flow / Density Meter
QD4
QD4 Diverter Valve
725 S Valve
Victaulic 725 S Diverter valve
800 Portal
1285 m
header tank
1205 level
High Pressure
Flush
220 m of 200NB Schedule
120 threaded connections,
1.5 m lengths
09
PASTE PUMP OIL
COOLING FAN
PASTE PUMP
Process water
200NB Schedule 120 –809N Weld Ring couplings
QD4
Dump to sump

(b)


1170/1205 Level - Stopes
250 Bar Proeinject Valve x 6
Peoeger Severe Service Knife Gate Valve
M-KGV-F – ANSI 600 Knife Gate Valve
Table E- Knife Gate Valve
FIT
Emerson Magnetic Flowmeter – ANSI 1500
PIT
Vega Pressure transducer with ANSI 1500 Flanged Spool
DIT
Coriolis Style Mass Flow / Density Meter
QD4
QD4 Diverter Valve
725 S Valve
Victaulic 725 S Diverter valve
725 S valve
Dump to 1205 Sump
To Upper Kora Nth Stopes
60 m of 200NB Schedule 120 threaded connections
Dump to 1170 Sump
PASTE PUMP OIL COOLING FAN
HP FLUSH PUMP
PASTE PUMP
350 m of 200NB Schedule 80 with 809 weld ring couplings
725 S valve
PIT
FIT
725 S valve
PIT
PIT
Dump to sump
Lower Kora Stopes
Dump to sump

(c)

18.5 Tailings Storage Facility

The tailings produced due to the Kora Expansion DFS scenario from 2022 to 2028 would total approximately 5.9 million tonnes, as shown in Table 18.5.1. From 2022 to mid-2024, the entire tailings production will be deposited into the existing TSF. By mid-2024, once the Paste Backfill plant commences operation, approximately half of the tailings generated will be processed through the Paste Backfill plant to fill the underground void; with the remaining portion deposited in the TSF.

Table 18.5.2 shows the total tonnes and volume of tailings required to be stored in the TSF from 2022 to 2028.

Table 18.5.1 Tonnes of Tailings Generated from 2022 to 2028 (DFS Scenario)

Year	Mill Production	Tailings Generated (*)	Tailings to TSF	Tailings to Paste Backfill Plant (**)
2022	435,000	413,250	413,250	-
2023	500,000	475,000	475,000	-
2024	850,000	807,500	614,977	192,523
2025	1,200,000	1,140,000	569,547	570,453
2026	1,200,000	1,140,000	562,189	577,811
2027	1,200,000	1,140,000	558,091	581,909
2028	848,191	805,781	391,525	414,256
Total	**6,233,191**	**5,921,531**	**3,584,578**	**2,336,953**

(*) Tailings correspond to 95% of the total mill production

(**) Assumed that Paste Backfill Plant would reach total operation capacity by mid-2024

Table 18.5.2 Tailings to TSF from 2022 to 2028

Year	Tailings to TSF (tonnes)	Volume of Tailings to TSF (m^3) (*)
2022	413,250	344,375
2023	475,000	395,833
2024	614,977	439,269
2025	569,547	406,819
2026	562,189	401,564
2027	558,091	398,636
2028	391,525	279,661
Total	**3,584,578**	**2,666,157**

(*) Considered average tailings settled density of 1.2 t/m^3 for 2022 and 2023 periods. Afterwards, settled density is expected to reach an average of 1.4 t/m^3.

Currently, the tailings generated are disposed within the existing TSF located downstream of the process plant adjacent to the Kumian Creek, which flows into the Baupa River, as illustrated in Figure 18.5.1. Since the commencement of operations, approximately 1,074,000 tonnes (894,930 m^3) of dry tailings have been stored within the TSF (based on an expected settled density of 1.2 t/m^3).

Future raises of the existing TSF embankment with their corresponding volume and remaining capacity for tailings storage are shown in Table 18.5.3. The detailed engineering design to raise the embankment of the TSF to Stage 2 (RL 520m) has been developed; however, embankment raise to Stage 3 (RL 530 m) remains at a conceptual level of design.

Figure 18.5.1 Existing TSF

Table 18.5.3 Tailings Volume Capacity for TSF Embankment Raises

TSF Raise Stage	Crest Level (RL m)	Volume Capacity (m^3)	Remaining capacity (m^3)
1A	512	1,227,000	332,070
1B	515	1,552,000	657,070
1C	517	1,782,000	887,070
2	520	2,145,000	1,250,070
3 (Conceptual)	530	3,540,000	2,645,070

18.5.1 TSF Alternatives

Various alternatives were considered to assess the capacity of the TSF to store tailings generated due to Kora Expansion. For instance, identification of potential locations to increase the storage capacity for future tailings included the assessment of surface disposal in the existing TSF (by raising the dam) or in a newly constructed facility. Following is a list of the potential sites considered for tailings disposal.

Existing TSF

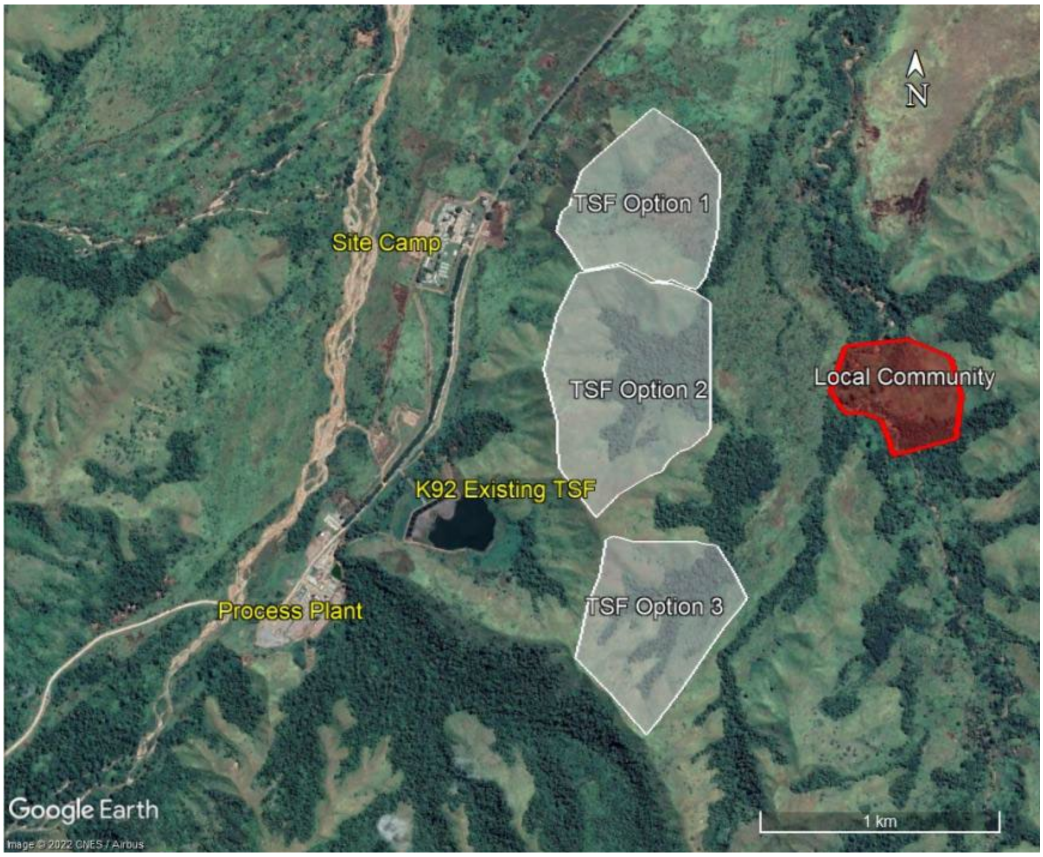
Raising the embankment crest level to RL 530 m (Stage 3) as a maximum elevation would extend the facility's life, allowing for enough volume capacity to dispose of almost the entirety of tailings generated from the Kora Expansion DFS production plan scenario. Similarly, the alternative of raising the embankment beyond Stage 2 (RL 520 m) would provide additional storage capacity, thus delaying the requirement for a new TSF.

New TSF Options

Three options for the location of a new TSF were identified, as shown in Figure 18.5.2. The size and configuration of the new TSF options would depend on the tailings deposition technology and the site conditions. Regardless of whether any technology can be implemented if feasible, various factors such as climate, seismicity, rainfall, regulatory, and logistics require consideration when developing a new TSF in PNG.

Considering that approximately 3.58 million tonnes of tailings will require storage and the expected density ranges from 1.2 t/m^3 for slurry deposition to 1.8 t/m^3 for filtered tailings compacted, the volume necessary to store tailings ranges from 2.98 Mm^3 to 1.98 Mm^3, respectively.

Figure 18.5.2 Potential Locations for a New Tailings Storage Facility



18.5.2 Evaluation of TSF Alternatives

The evaluation of TSF alternatives to store tailings from the Kora Expansion DFS scenario considers the site limitations, the client recommendations, and the best available technology. The best applicable technology does not represent any specific technology, but it means the technology that best adapts to the site conditions that help to mitigate the risks and is approved by the regulators.

While identifying methods to provide reliable containment of the tailings for the mine's entire life, the conservation of water and chemical stability were not recognized as an issue for the site conditions. First, the benefits of water conservation, resulting from the implementation of thickened and filtered tailings, were not regarded as advantageous due to the net water surplus that requires continuous water discharge from the site. Second, the chemical stability was not a concern due to the tailings' chemical characteristics and considering that the tailings will remain saturated over the long-term, reducing the risk of acid formation from a long-term perspective.

The alternatives evaluated to implement technologies for different TSF solutions are shown in Table 18.5.4.

Table 18.5.4 Applicability of Tailings Technology to Site Alternatives

Alternatives	Existing TSF		New TSF		
Applicable technology	Raise to Stage 2 (RL 520 m)	Raise to Stage 3 (RL 530 m)	TSF Option 1 (30 Ha)	TSF Option 2 (15 Ha)	TSF Option 3 (20 Ha)
Conventional Slurry	●	●	●		●
High-Density Slurry	●	●	●		●
Low-Density Paste	●	●			
High-Density Paste				●	
Filter Cake			●	●	●
Classified Tailings					●
Geotubes			●	●	●
Co-disposal			●	●	●

Existing TSF

This scenario evaluates the TSF capacity to Stage 2 (embankment crest level RL 520 m) and the increased capacity by raising the embankment crest level to RL 530 m (Stage 3). Assumptions for this scenario include:

- The underground void capacity to store approximately 50% of tailings generated following the commencement of the paste backfill plant from mid-2024.
- No particle size selection occurs for tailings to the TSF or paste backfill.
- Tailings to TSF would include the adoption of conventional slurry, high-density slurry or low-density paste.
- The average settled density for tailings of around 1.4 t/m^3.
- There is no consideration of adopting the piggy-backing method, which includes high-density paste or filter cake deposited on top of tailings slurry.
- There is enough borrow material (rockfill) to build the embankment up to stage 3 (rl 530 m).

The results obtained for the evaluation of the existing TSF scenario are shown in Figure 18.5.3. The capacity of the TSF with an embankment raised to Stage 2 would not accommodate the forecast tailings generated for the DFS production plan. However, the increase in volume available due to raising the embankment to Stage 3 (RL 530 m) will meet the capacity to store the tailings generated over the life of the mine.

During the production period, the embankment raises should be continuous with a yearly increase of the crest level of the dam, as shown in Figure 18.5.4. Therefore, the sequence to build the dam over the period assumed that the settled density could reach an average of 1.4 t/m^3 in 2024. Thus, actions shall be implemented to enhance the settled density of tailings and maximize the storage capacity utilization.

Figure 18.5.3 Volume Required and Embankment Raised Capacities

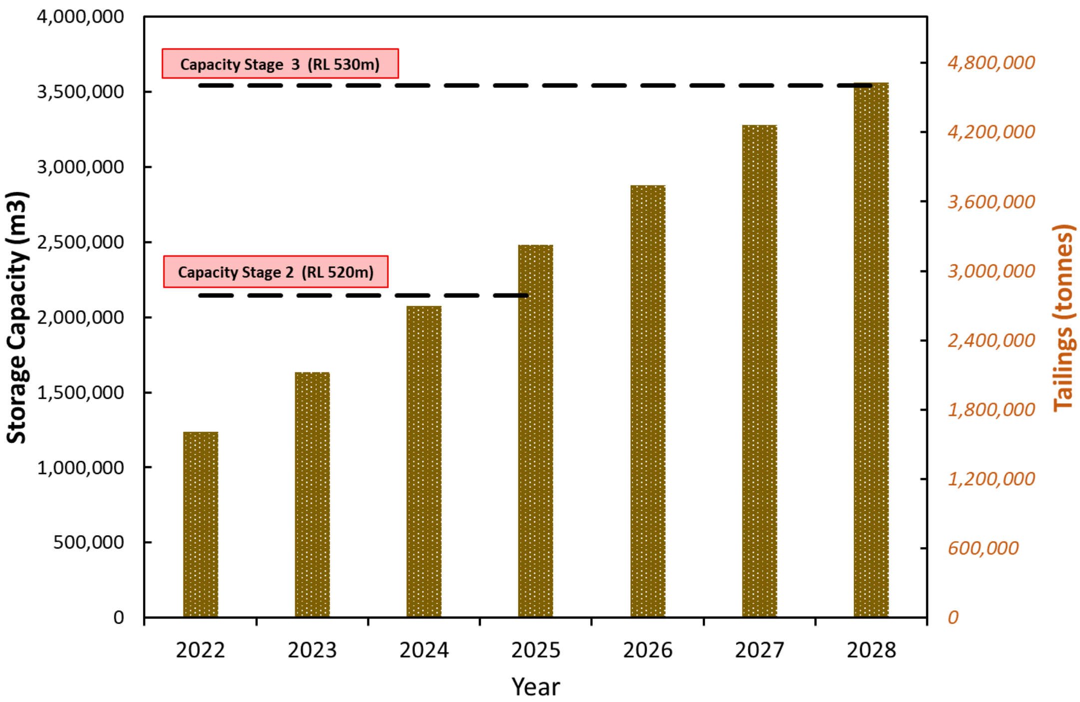
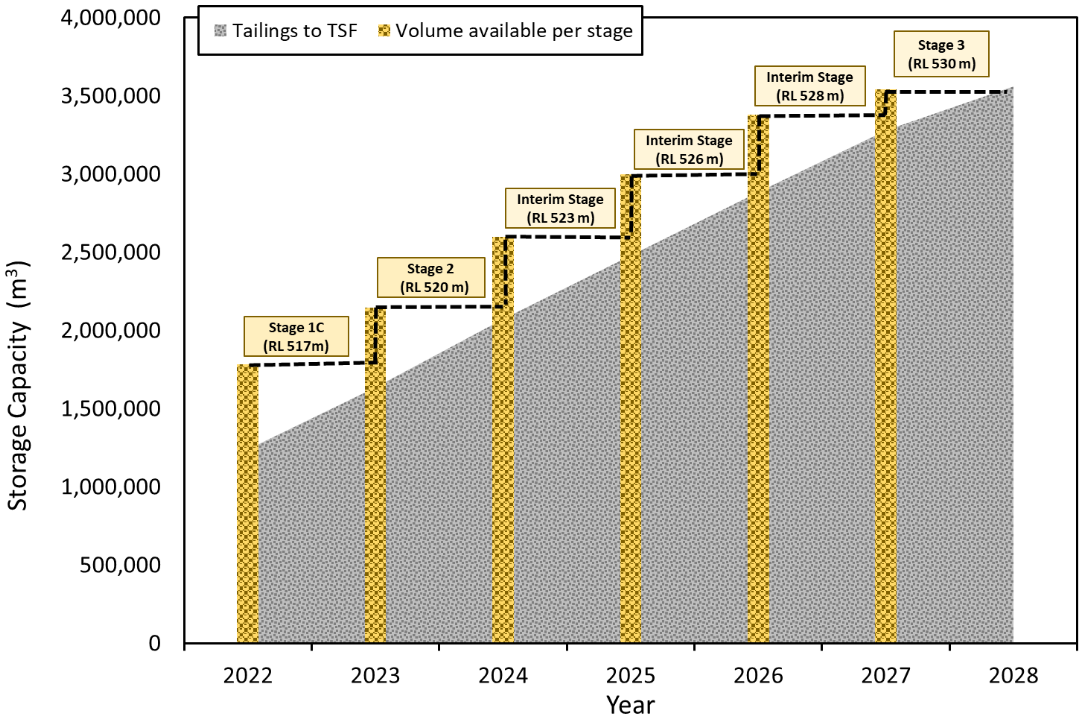


Figure 18.5.4 Embankment Raising Sequence During the Production Period



Managing the TSF effectively by increasing the deposited settled density of tailings to an average of 1.4 t/m^3 will lower the necessity of raising the dam, reducing the requirements for additional borrow material for the embankment construction.

A considerable advantage of the existing TSF is the understanding of the foundation conditions and the available data for an appropriate analysis of the range of loading scenarios. Currently, preliminary data of the foundation conditions suggests that the existing TSF can accommodate additional raises of the dam, providing that the slopes of the dam match the foundation strength.

New TSF's

Three options for a new TSF in different locations were preliminary identified, as described in Section 18.5.1. The size and configuration of a new TSF would depend on the tailings deposition technology; however, in general, the required area for any technology would be similar.

The new site for the construction of the TSF footprint is unlikely to offer more advantages over the existing TSF footprint, apart from the additional storage capacity and the possibility of adopting technology to manage tailings more efficiently.

Implementing a new TSF would require significant investigation, such as characterising the foundation conditions sufficiently to enable appropriate design. In addition, a vast increase in land disturbance area would be associated with the construction of the new TSF, including new borrow areas for dam construction materials, new access roads, new tailings and water conveyance corridors/management facilities, and new seepage collection and water management ponds. Further, the requirement for additional post-closure work following mining completion would be significant.

Finally, it would take at least three years to develop a new TSF, including site investigation, design, environmental assessment, procurement, construction, and approval of permits. Therefore, considering the duration of the DFS scenario, a new TSF is an unattractive option and likely would prevent the continuation of operations (i.e. result in a shut-down condition).

18.6 Tailings Management

It is reported that 1,073,916 tonnes of tailings have been discharged into the TSF since the TSF started operation in 2008. The waste stream from ore processing comprises silty sand tailings from the flotation circuit. The flotation tailings are relatively inert and are composed primarily of quartz and waste rock sand with only a minor fraction of sulphur-bearing minerals. Further, inspections of the tailings material indicate it possesses acid-producing potential. However, due to the excessive rainfall characteristic of the site, water recovered off the TSF remained inside acceptable pH parameters. The addition of lime in the process to a level of pH 10 also assist in maintaining the overall pH of the water reclaimed of the TSF. Nevertheless, no detailed studies have been completed on tailings chemical characterization to confirm the potential of acid generation from tailings from a long-term perspective.

Currently, the tailings from the last scavenger flotation cell are pumped to the tailings dam with a solids content of 34% (w/w) through an HDPE pipeline and then discharged by a spigot pipeline located at 25-metre intervals situated at the wall crest. An average settled density of 1.2 t/m^3 is expected; however, the settled density value of tailings is also expected to be quite variable within the TSF and would be affected by several conditions, including the initial solids concentration, rainfall, earthquakes, and tailings management.

The only water discharged from the plant is contained within the tailings. The majority of the water in tailings is released as decanting water from settling and self-weight consolidation. Decanted water is recovered via the tailings return pumps and returned to the process plant for reuse. Only the excess water, normally after rain events, is discharged to the overflow wetland system. Before discharge to Kumian Creek, the excess from the TSF flows through a wetland system. No AMD is anticipated as TSF water is currently used as process water. The tailings have an elevated pH due to the addition of lime to allow flotation at pH 10^+. Discharged excess TSF water has a pH >7, confirmed by water monitoring downstream from the TSF.

The design of any future embankment will use the maximum potential tailings and water levels to calculate the loads on the embankments so that, under all conditions, the embankments or dams will be stable. The operating principles for the facility will be developed so that under normal operating conditions, the loads imposed on the embankments will be less than the design values. These principles will be based on the following aspects:

- Reduce free water within the impoundment (which reduces loads as well as the consequences of a breach).
- Maintain the free water away from the embankments where possible.
- Promote unsaturated conditions in the tailings (by the provision of drainage).
- Maximize the settled density of tailings in the TSF to maintain an average of 1.4 t/m^3 by undertaking assessment and implementation of various initiatives such as:
  - effectively management of spigots and placement history by improving deposition control reduces water ponding
  - a better understanding of settling behaviour and how sorting particles affect the settled density along the beaches of the TSF
  - assess the feasibility of installing wick drains in specific locations of the TSF every few years to enhance water release from the pores drainage and promote the initial stage of consolidation, which is when the most significant amount of water is released in the consolidation process
  - assess the feasibility of installing carpet drains on the surface of the tailings every few years to reduce the flow path by consolidation. The rate of consolidation decreases as the square of the flow path (H^2), so reducing the flow path can significantly decrease the time required for consolidation
  - enhance the settling of tailings by adding an in-line dosage of coagulants and flocculants to boost the behaviour in-situ
  - promote the water losses from the TSF in the dry season by assessing the feasibility and benefits of engineered water evaporators. This initiative can encourage further consolidation of tailings, and the reduced water pond before the rainy season would assist in lowering the water released to Kumian Creek.

The tailings dam operates under an amended Environment Permit EP-L3 (34). The Papua New Guinea Conservation and Environment Protection Authority (CEPA) issued the current permit on 12 August 2019.

A process of progressive requests for approval is in progress with the Mineral Resources Authority (MRA) for the next TSF embankment raises.

18.7 Bridges

Three bridges over water courses will be upgraded as part of the project scope. These bridges are on the service road from the process plant up to the 800 Portal, located within the mine lease.

- Kokomo bridge.
- Kasese bridge.
- Lower Baupa bridge.

18.8 Accommodation Camp

The Kumian Camp can accommodate 850 personnel after additional accommodation units for both COVID-19 quarantine capability and increased personnel were installed and constructed during 2019 and 2020. Construction of a new kitchen and mess was completed in 2020. The camp also contains a health / first aid clinic for the benefit of K92 employees. In addition, there are recreation facilities including gym, covered basketball court and sports grounds.

Camp upgrades to be made as part of the FS include building three additional 64 bed accommodation units, an additional twenty 2-person ensuites, a new community affairs building, addition of a training and induction centre, an OHS department building, extension to the mess, improving the recreational facility and upgrading the camps water, power and sewage systems.

Additionally, three of the existing 50 bed accommodation units will be demolished and replaced. These upgrades will increase the camp capacity from 850 to 1082 personnel.

19.0 MARKET STUDIES AND CONTRACTS

During 2019, K92ML entered into a new 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.

20.0 ENVIRONMENT AND COMMUNITY

20.1 Introduction

The environment and community aspects relating to the Kora Expansion Project (the Project) are discussed in this section. This includes a concise analysis of key environmental and community risks associated with the project, including associated statutory requirements and assessment of environmental and community impacts. These aspects have been addressed in the context of existing and approved environmental and community impacts from the ongoing operation of the Kainantu Gold Mine.

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 expected to make. 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 Requirements

The environment and community assessment framework applicable to the Kora Expansion Project Feasibility Study (FS) includes regulatory and policy requirements of Papua New Guinea (PNG). National Government regulatory bodies, such as the Mineral Resources Authority (MRA) and Conservation and Environment Protection Authority (CEPA) have granted various approvals, permits and licenses required by K92 Mining to operate the Kainantu Gold Mine, and these approvals are subject to review and likely amendment for 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.

K92 Mining's existing mining lease, (ML150) contains approximately 90% 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 K92 Mining's exploration license, (EL 470). K92 Mining will need to extend ML150 or apply for another mining license to include the extension of Kora to the south prior to mining.

K92 Mining will be required to re-survey its ML boundary as part of an amended proposal for development for the Kora Expansion Project. This is subject to ongoing discussion between K92 Mining and MRA to determine amended ML boundary.

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:

- The PNG Government's rights over reserved land.
- 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. There may be implications for the Kora Expansion Project given a portion of the deposit lies outside the existing ML and may trigger these amendments.

20.2.2 Environment Act 2000

CEPA administers the Environment Act 2000, the legal framework for regulating the environmental effects of mining (and other) projects. Under this act, K92 Mining holds Environment Permit EP-L3(34) for existing activities at the Kainantu Gold Mine. Expansion of the mine for the project will be subject to approval under the Environment Act 2000.

Under section 71(2) of the Environment Act 2000, a proponent may submit an environment permit amendment application to address changes to its permitted activities. Following lodgement of the amendment application, the Director of CEPA determines whether the change constitutes a 'major amendment' or a 'minor amendment'.

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.

The Kora Expansion Project considers an expansion to underground mining at Kainantu with on-site treatment of mine material by conventional milling, gravity and flotation recovery through a standalone 1.2-million-tonne-per annum ('Mtpa') process plant over a 7-year Life of Mine. The tailing storage facility (TSF) is also proposed to sequentially raise the height of the TSF embankment beyond the existing (and original design nominal) crest elevation of 509 mRL to approximately 530 mRL. The proposed increase in throughput to 1.2 Mtpa and the raising of the TSF embankment are substantial changes to what was originally approved in the 2002 Environment Plan and is likely to be subject to a major amendment of the existing environment permit for which an environment impact statement (EIS) will be required.

If, as anticipated, CEPA confirms that the Project will undergo a major amendment, K92 Mining will be required to submit an Environmental Inception Report (EIR) under Section 52 of the act, the objectives of which will be to:

- Identify the potential environmental and community issues of developing the 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 EIS under Section 53 of the act.

Once the EIR is submitted, CEPA can confirm that the scope of the proposed studies is acceptable (or, alternatively, indicate where additional work is required), which will provide greater certainty about the required scope, schedule and cost for the EIS.

The environmental and social impact assessment process will result in the production of an EIS for the project prepared pursuant to Section 50 of the Environment Act 2000. The purpose of the EIS will be to provide government authorities, the general public and other interested parties with information on the nature of the Project, its environmental effects, and the principles and objectives for operational management and monitoring. The EIS typically contains:

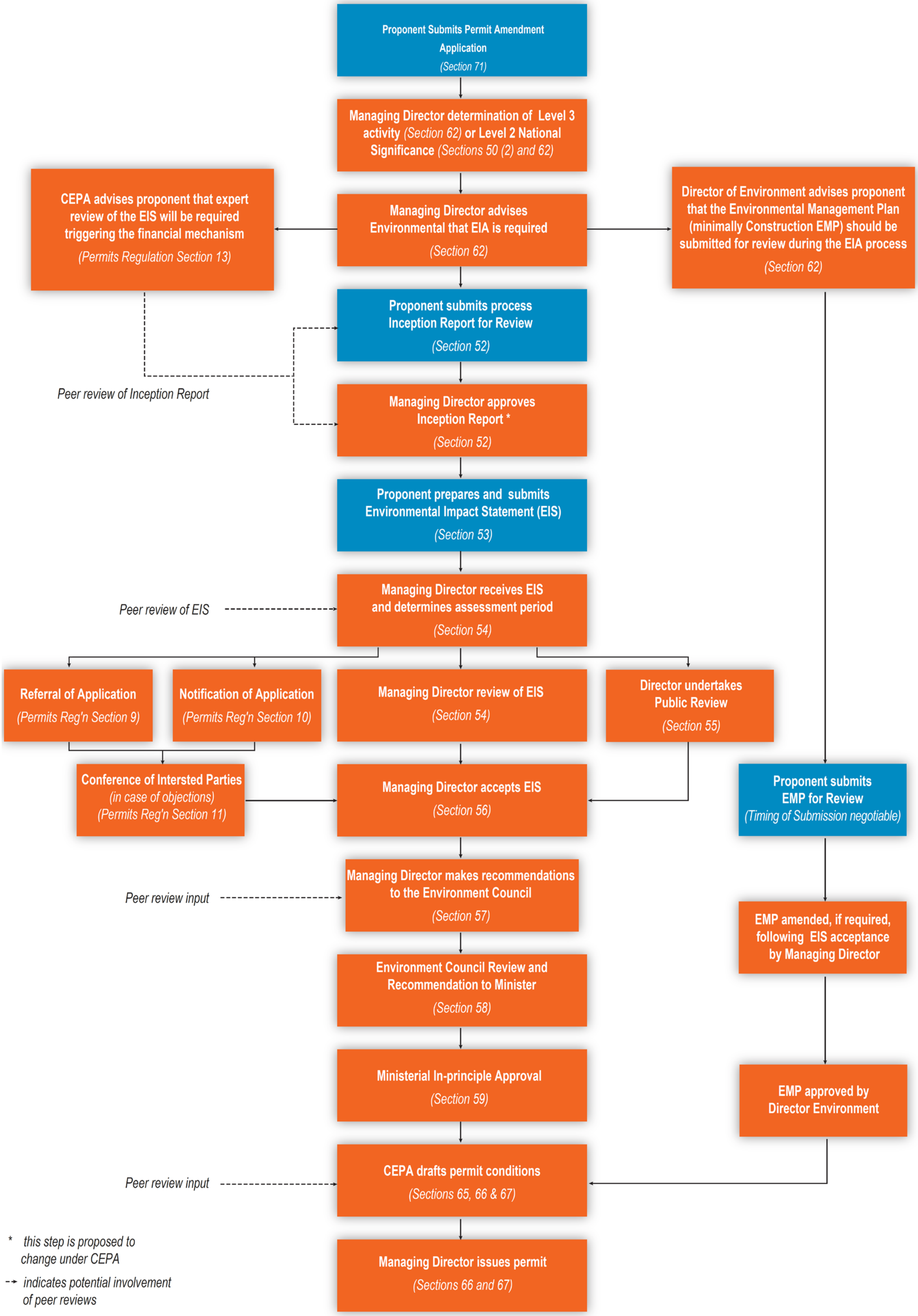
- Executive Summary, in English and Tok Pisin.
- Main Report.
- Technical Appendices.

The DEC (2004) publication Guideline for Conduct of Environmental Impact Assessment and Preparation of Environmental Impact Statement describes the requirements for an EIS. It requires the EIS to assess potential environmental and community impacts of a project and to describe how the proponent intends to avoid, manage, or mitigate these impacts. Since the time this guideline was developed, the level of detail expected by CEPA and other stakeholders has increased and hence these guidelines should be considered the minimum required for an EIS. In addition, this guideline does not consider nor provide sufficient coverage to address contemporary international benchmarks such as the Equator Principles 4 and International Finance Corporation Performance Standards.

20.2.3 EIS Approval Process

The EIS approval process in PNG involves a combination of statutory and non-statutory timeframes as shown in Figure 20.2.1. The EIS is a public document, and the PNG Government is expected to engage an external consultant to conduct an independent peer review of the document. Recently, CEPA has required peer review consultants of the EIS to increase the level of PNG participation in this process. It should be expected that the peer review of the EIS for the Project will also require this.

Figure 20.2.1 Environmental Approvals Process for Level 3 Projects or Major Amendments



There is usually a requirement for an 'EIS road show' by the proponent to present the findings of the EIS to the local community and all levels of government.

The findings from the independent peer review process and the EIS roadshow will form part of a submission (s.57) to the Environment Council to deliberate and recommend for an approval in principle (s.58) from the Minister for Environment and Conservation or request for additional information via the Director of Environment (i.e. Managing Director of CEPA).

20.2.4 Permit Fees

The Environment (Fees and Charges) Regulation 2002 outlines the fees and charges associated with issuing permits by CEPA. Depending on associated capital expenditure to construct the project, the amount payable if capital expenditure is above K500 million will be in the order of K125,000. This amount excludes cost for an independent peer review of the EIS, whereby CEPA can request this review at the proponent's expense. Recent experience on other projects suggests the independent peer review may cost between K500,000 and K1 million (approximately A$200,000 to A$400,000).

20.2.5 Environmental Approval Required for Project Construction

K92 is progressing a series of minor amendments to support current operations. K92 will require CEPA to issue the major amendment to the existing environment permit before the ramp-up of the milling throughput of 1.2 Mtpa can commence. There is provision in the Environment Act 2000 for an environmental bond to be levied on a proponent before construction can commence but this is relatively untested in PNG.

Mine Closure Guidelines

In late 2019, the MRA launched a new mine closure guideline titled 'Mining Project Rehabilitation and Closure Guidelines – Papua New Guinea'. The guidelines are generally consistent with past practice in PNG, with one or two notable exceptions. The greatest area of risk is the requirement for financial assurance to be provided before the mining lease can be granted; this is a significant change for PNG. While not yet legally enforceable, it is likely that the new policy will be applied to mine closure planning for the Project.

Supplementary to legislation, policy and regulation documents, the PNG mining industry environmental code of practice (ECP) addresses the issue of rehabilitation and mine decommissioning. The focus of the code is on the physical aspects of closure, such as building removal, waste seepage, safety aspects of open pits and underground workings, and the geotechnical stability of tailings impoundments.

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. It is accessible via the mine access road that turns off towards Eastern Highlands from the Madang – Lae Highway at Gusap, Morobe Province.

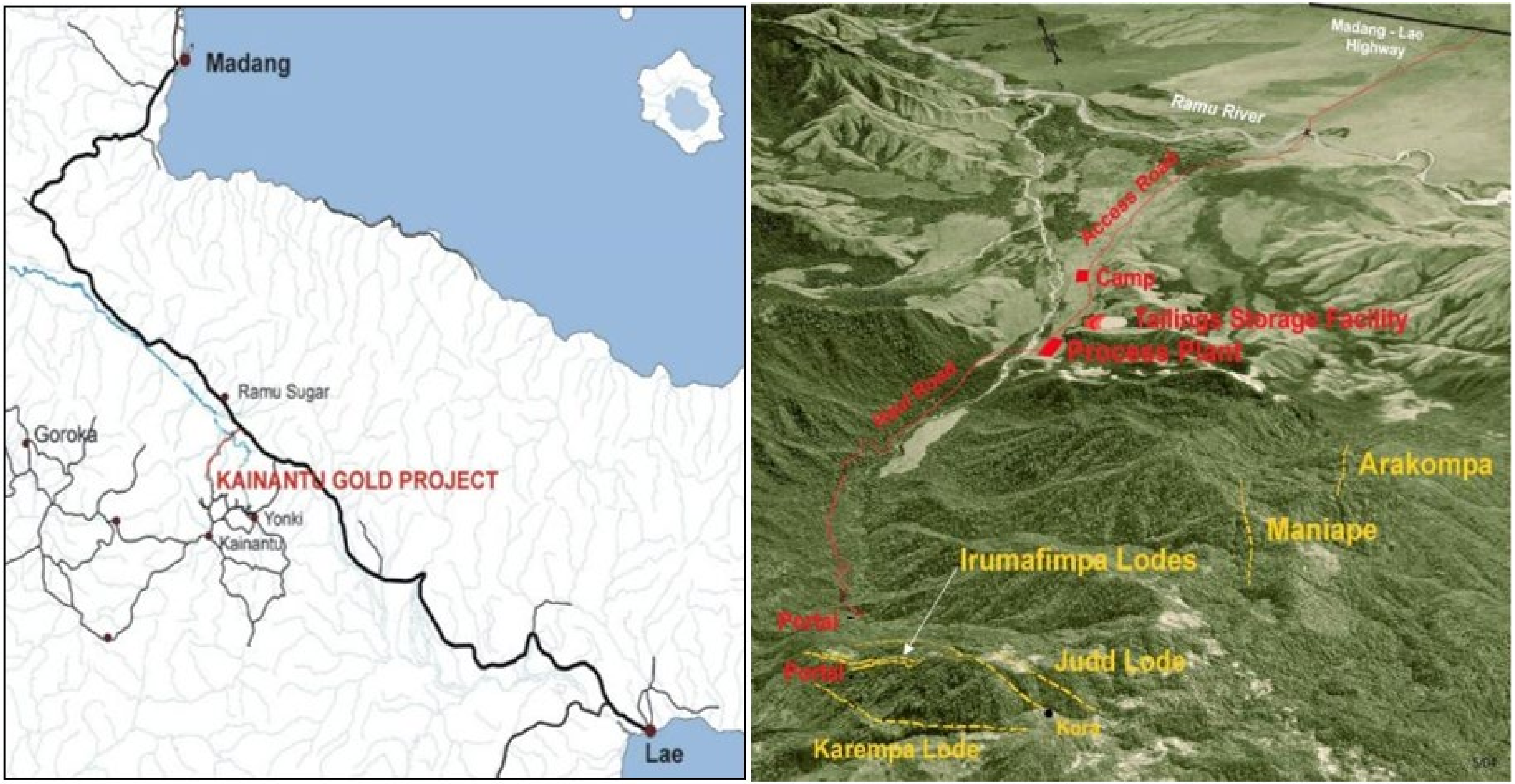
The access road crosses the Ramu River alluvial plain. The mine camp site, TSF and processing plant are in this area along the access road. The access road continues southwest into an area of more rugged mountainous landscape of ridge and ravine landforms with elevations from 400 m to 1000 m above sea level (Loffler, 1974). The access road ends where the mine portals are located (Figure 20.3.1). The mine's dewatering pipeline runs alongside the access road at the southern section of the existing project area (Figure 20.3.2).

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/h to 4.7 km/h, 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 20.3.3).

Figure 20.3.1 Proposed K92 Gold Mine and Associated Facilities

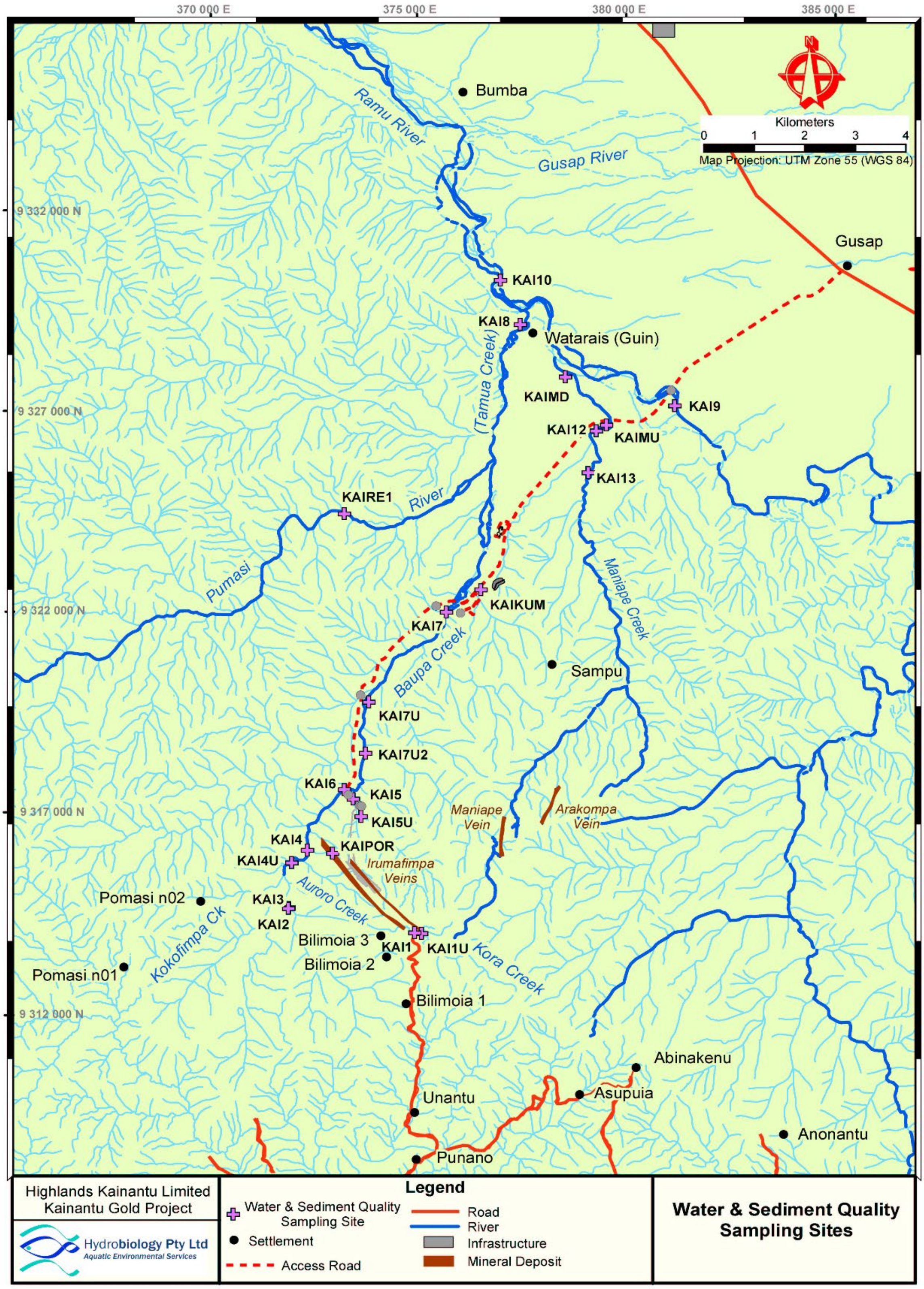


Source: K92 Mining Ltd (2020)

Figure 20.3.2 Mine Dewatering Pipeline, Extraction and Discharge Points Location Map

Source: (K92 Mining Limited, undated)

Figure 20.3.3 Water and Sediment Quality Sampling Sites



Source: Hydrobiology Pty Ltd, 2005

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 (Figure 20.3.3). Water quality monitoring has been undertaken or commissioned by K92 Mining and previous operators. Results have been reported in annual reports from 2004 to 2020. This monitoring is ongoing.

In the most recent water quality monitoring surveys, pH levels at all monitoring sites for the mine ranged from 6.1 to 7.8. Dissolved oxygen (DO) concentrations ranged from 7.09 to 8.27 mg/L. Total suspended sediment (TSS) concentrations ranged from 14.5 to 229 mg/L. The higher TSS measurements were from Ramu River, downstream of the Baupa / Ramu and Maniape / Ramu confluence. The lowest TSS measurements were from monitoring sites along Kumian Creek. Oil and grease at all monitoring sites were measured as <5 mg/L (K92 Mining Limited, 2020).

The annual mean concentrations of most trace metals for the 2020 monitoring period were within the permissible limits described under the existing Environment Permit EP-L3(34) (K92 Mining Limited, 2020). However, Mati downstream and Baupa downstream had higher concentrations (ranging from 1.16 mg/l to 0.502 mg/l) recorded for manganese 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 ore from the underground mine. Mati Creek is the recipient of the mine water discharge from the underground mine at the 800-portal location (K92 Mining Limited, 2020). Dissolved metal concentrations for underground mine water were elevated at the 800-portal monitoring point compared to the freshwater criteria in Schedule 1 of the PNG water quality guidelines. However, concentrations become diluted at the Mati downstream monitoring point, which is the compliance point, 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 TSF.

Water quality for potable water is sampled from the site's two bores at the camp and process plant. Potable water sampled and tested at the process plant and camp sites in 2020 had no presence of the bacteria *Escherichia coli* (E. coli) (K92 Mining, 2020). Coliforms were detected at six sites of the 12 drinking water sample sites. According to the existing Environment Permit EP-L3(34) the maximum permissible level for E. coli (per 100ml sample) is none present and for total coliforms (per 100ml sample) is less than three if E. coli is absent.

In the 2020 environmental monitoring, TSF water quality (including both 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 the Environment Permit 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 (K92 Mining, 2020).

Hydrology

The existing project area is part of the upper Ramu River catchment. The surroundings have dendritic drainage system of smaller creeks and streams draining from the surrounding mountains and hills in the area into four catchments (NSR Environmental Consultants Pty Ltd, 2001). Two of these catchments are directly impacted by the project infrastructure (Figure 20.3.3).

20.3.3 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).

A hydrogeological model is being developed as part of this FS and findings will be used as part of the technical assessments to inform the EIS.

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. Studies are underway to clarify the linkage with regional groundwater resources (Woodward, Tear, Desoe, & Park, 2020).

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 clay-like 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).

20.3.4 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. One-hundred-and-ten 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).

20.3.5 Air and Noise

Noise exposure measurements were conducted by SGC Safety Pty Ltd as part of the Occupational Hygiene Program in 2017 to 2018. Noise surveys for 2019 to 2021 were delayed due to the coronavirus pandemic. Measurements were for personal noise exposure for workers during normal operations. No measurements are taken for environmental noise levels.

Results from 2018 showed elevated personal noise exposure above the New Zealand average allowable exposure limit over 6-hour period (85 dB(A)). Regulation 56 of the Queensland Work Health and Safety Regulations 2011 sets an exposure standard for noise in relation to a person of: $L_{Aeq,8h}$ of 85 dB(A). As personnel worked extended shifts (longer than 8 hours) adjustments were made to the $L_{Aeq,8hr}$ (where $L_{Aeq,8h}$ = 85 dB(A)), in accordance with AS: 1269.1. The length of time a person without hearing protection can be exposed before the standard is exceeded is 82 dB(A) over 12 Hours. Results showed a range of 81.6 $L_{Aeq,\ 8hr}$ dB(A-adj) for a 12-hour shift, which is below the $L_{Aeq,8hr}$ 85 dB(A) standard to 109.5 $L_{Aeq,\ 8hr}$ dB(A-adj) for a 12-hour shift, which is 282 times greater than the $L_{Aeq,8hr}$ 85 dB(A) standard (Gmelig, 2018).

An airborne contaminant survey was carried out as part of the Occupational Hygiene Program in 2019. This survey was conducted to monitor airborne contaminant exposure of employees and contractors during normal operations. This involved measuring levels of airborne contaminants including, diesel particulate matter (DPM), respirable crystalline silica (α-quartz) (RCS), inhalable metals and welding fumes. Exposure to airborne contaminants were recorded mostly below the 12 Hour Shift Adjusted National Exposure Standards. There were instances of higher recordings beyond the 0.071 acceptable level for RCS (<0.005 to 0.090) and DPM (<0.002 to 0.099) (Gmelig, 2019). Environmental air quality monitoring has not been conducted.

20.3.6 Community

The Bilimoia Baseline Study (K92 Mine Limited, 2018) classified the population of the communities in the mine impacted area as uplanders in the Eastern Highlands Province and lowlanders in Morobe and Madang Provinces. The total population of the uplanders is around 7,580. The affected communities in the project area includes Bilimoia, Unantu / Puanono and Pomasi villages.

The villages in the lowland area in the Madang and Morobe Province are Waterais villages and Marawasa/Musuwan villages with a total approximate pollution of 2,460. It is believed that Bilimoia's population has increased by approximately 40% since 2001, that is, at an average rate of 4% per year (Woodward, Tear, Desoe, & Park, 2020).

The Bilimoia Baseline Study (K92 Mine Limited, 2018) provided information about the demography (sex and age groups segregated data) for Bilimoia (1, 2, 3) community. Table 20.3.1 provides a summary of this data.

Table 20.3.1 Demography of Bilimoia 1, 2 and 3

Age Range	Village Age Range Grouping								
	Bilimoia 1			Bilimoia 2			Bilimoia 3		
	Males	Females	Total	Males	Females	Total	Males	Females	Total
1- 9	160	151	311	94	64	158	39	33	72
10- 19	164	150	314	114	97	211	29	31	60
20- 29	170	159	329	72	88	160	27	35	62
30- 39	142	135	277	42	47	89	24	22	46
40- 49	75	53	128	42	36	78	17	18	35
50- 59	61	46	107	37	30	67	12	8	20
60- 69	77	65	142	35	34	69	2	0	2
70- 79	45	37	82	28	33	61	0	0	0
Total	**894**	**796**	**1,690**	**464**	**429**	**893**	**150**	**147**	**297**

It is important to understand the different age and sex segregated groups as it is crucial for any development or intervention by the project. Different sex and age groups have different needs. To date, the population in the affected areas is estimated to be up to 21,000. A population and household survey would need to be done to confirm the current population, including the number of people that have immigrated from other areas.

The main source of income for the area is through the Kainantu Gold Mine and associated support contracts (K92 Mine Limited, 2019). In 2001, it was estimated that 60% of the population's income came from sales of gold from small scale mining and 20% from wages obtained from working with the exploration team. The remaining 20% was earned from sales of coffee, vegetables, and other goods at trade stores.

Community and Social Development

K92 Mining's initiatives up to 2020 included (K92 Mine Limited, 2020):

- Agreements signed between landowner groups and service providers to provide long term supply of services.
- Prioritization of local hiring: developing long term and transferrable skills.
- Water supply: developing and securing safe water supplies.
- Regular community meetings with all members and leaders, sponsorship of events.
- Infrastructure: development and upgrading of roads, bridges, travel ways.
- Maintenance: maintaining roads and travel ways for the communities.
- Medical aid posts including medicine and personnel, donations to hospital.
- Education: donations to schools, scholarship.

Health Program

The existing project commissioned two community water projects which were completed in 2020, for a total investment of US$39,914. Gravity-fed water supply projects are in progress for Bilimoia 1 community and for Pomasi 1 village (K92 Sustainability Report, 2020). There is no available information about accessibility to health facilities and the incidence of diseases for the mine impacted communities.

Education Assistance

The Kainantu Gold Mine continues to provide educational assistance to students from the mine impact area who attend tertiary institutions. A primary school was built in Bilimoia in 2004 and registered with the Eastern Highlands Province Department of Education.

Livelihood Programs

The mine has been developing small businesses and empowering women through a sustainable agriculture livelihoods program, employing 75% women, and successfully growing new types of crops in the lowlands (K92 Mining 2020 Sustainability Report).

Business Development

Bilimaio Business Development Company (BildevCo) co-ordinates all the business activities of the landowners. According to an updated K92 baseline report (K92 Mining Limited, 2018), most directors and executives of the group are from Bilimaio 1 and 2. BildevCo is the link between the K92 mining operation and the landowners. The landowner company is Bilimaio interim landowner association (BILA).

Joint ventures established to date include those associated with transportation, security, ancillary mobile plant, Kumian Camp services, catering (including sourcing of local ingredients from communities), maintenance services and exploration support. The status of how BildevCo is progressing and understanding their sustainability plans will assist in the mine closure implementation planning.

Memorandum Of Agreement (MoA)

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 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 Memorandum of Agreement was reviewed over 13-17 July 2020 by all stakeholders, has been initialled by all of them, but awaits final signature by the government and the landowners.

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 unit. The SPP indicates the type and range of possible contracts and joint ventures for properly qualified landowner entities to take part in, while the BDP is a business training and instruction curriculum and schedule for landowners. These plans, with the Memorandum of Agreement, signed or not, now provide the External Affairs' team's approach to engagement and business with the community.

Part of the Memorandum of Agreement renegotiation was an offer of 5% equity in the mine to the Provincial Government (1.5%), the Blimoias (3%) and the Watarais, Unantu and Pomasi peoples (0.5%). Shareholding would be paid from dividends on the basis of the value of K92ML's sunk costs. Equity issue is, however, dependent on the final signing of the Memorandum of Agreement by all parties.

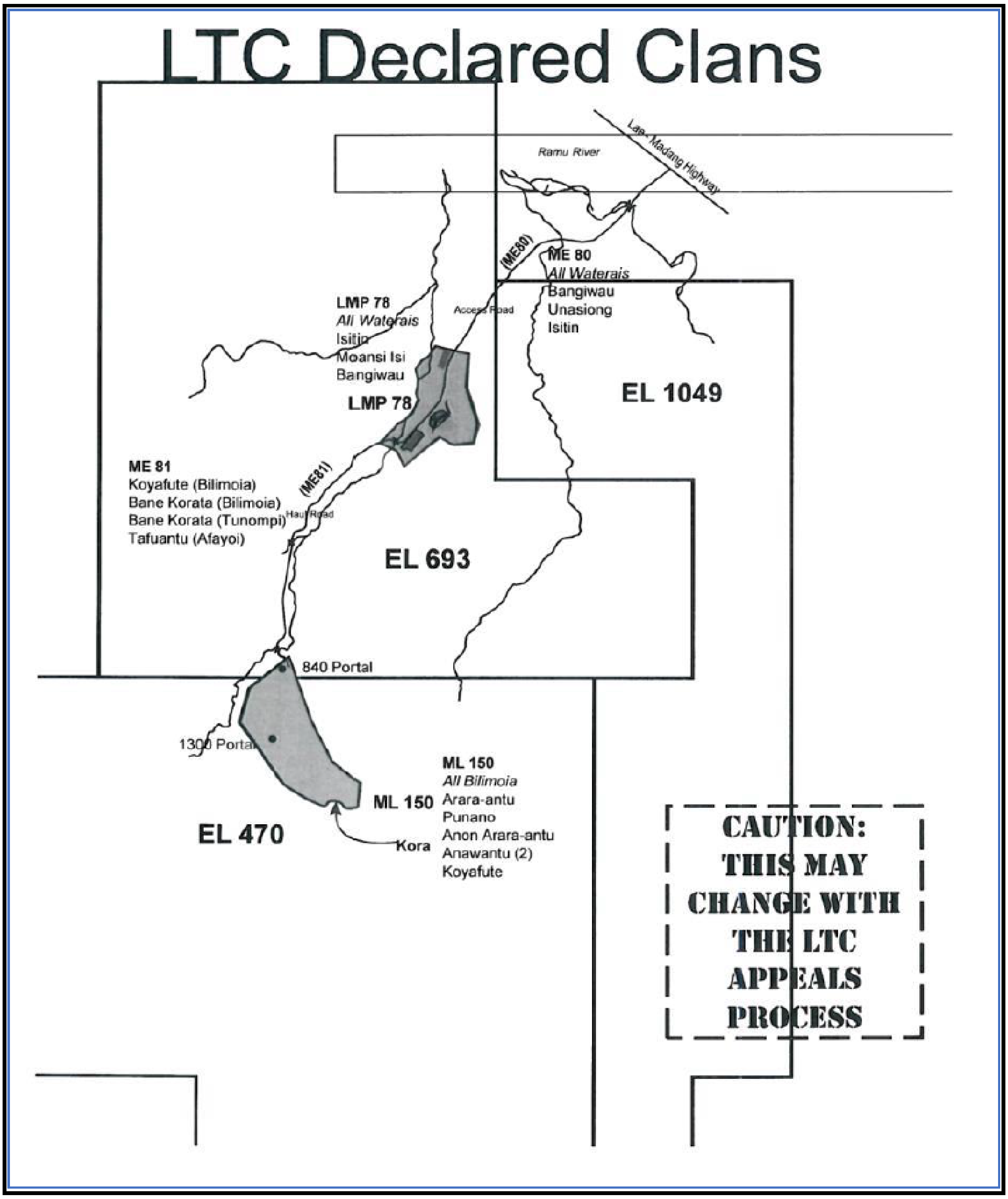
Consistent with the expectation of resource projects in PNG, the local communities have two Landcos which represent them businesswise to K92ML. All community members are shareholders in one or the other. The two Landcos have their own chairmen and boards and are assisted management-wise by K92ML's External Affairs' business development officers, of which there are two Papua New Guineans are two from Australia. A fifth officer, a dedicated Landco finance analyst and controller is soon to arrive at Kumian. K92ML is currently endeavouring to combine the two landcos under one Umbrella, as expected by the government, which would then entail the joint ventures currently enjoyed by the two Landcos with significant goods and services providers to K92ML but distributed discretely between them being redistributed across all four communities.

Compensation and Landownership

According to the Kora PEA report (Woodward, Tear, Desoe, & Park, 2020), K92 Mining has obligations to compensate landowners annually who are affected by the operations of the Kainantu Gold Mine. These compensations are governed by the Mining Act 1992 and land and environment compensation agreement (CA) for mining lease ML 150 with the Bilimoia Interim Landowner Association (BILA) and certain landowners / clans listed in the agreement.

In 2003, Highlands Pacific Limited (previous operator of Kainantu Gold Mine) signed a Lands and Environment Compensation Agreement with identified impact communities (Figure 5). Reviews of the agreement was scheduled to occur every three years however there has been no reviews to date. This is due to delays with the Land Titles Commission (LTC) completing an investigation into landholding within the existing operations leased boundaries. Coupled with appeals from landowner parties to a determination by the LTC in 2009. Landownership will remain under dispute until the LTC declaration of 2009 is resolved (Woodward, Tear, Desoe, & Park, 2020).

Figure 20.3.4 Location of LTC Declared Clans



Source: Woodward, Tear, Desoe, & Park, 2020

20.4 Assessed Environment and Community Impacts

20.4.1 Assessed Environmental Impacts

The Kora Expansion Project proposes to continue underground mining using the long hole stoping method (Woodward, Tear, Desoe, & Park, 2020). Waste rock or mullock will be stockpiled on the surface as opposed to current practice of backfilling in the mine stopes. The management of stockpiled waste rock will need to be assessed and measures determined to manage the potential for AMD.

The existing process plant will be extended as part of the expansion project to increase throughput to 1.2 Mtpa. With this increase in throughput there will be an increase in mine tailings and therefore the TSF is proposed to be raised to increase its storage capacity. Discharge of treated wastewater from the TSF will be minimal or remain unchanged since it is proposed that process water will be reused in the process plant. In addition, co-disposal of mine tails is also being considered via a paste fill plant to supplement 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. As part of the FS, further investigations to incorporate methods to neutralize potential AMD from paste fill will be assessed. Impacts on groundwater will also require assessment.

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.

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. Management of waste rock will therefore be required. Additional geochemical assessment of the waste rock is being undertaken but results are not yet available.

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 testwork undertaken as part of the development were not yet available. The initial metallurgical testwork 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.

For the tailings, a percentage of tailings from the expansion project will continue to be treated and stored in the expanded TSF with the balance proposed to be turned into tailings paste via a proposed paste fill plant and backfilled in the underground mine stopes.

Water Quality and Hydrology

The expansion project will increase gold production, and this will result in more tailings being discharged to the TSF. However, discharge volumes from the TSF into the receiving environment (i.e. Kumian Creek) will be minimal or remain unchanged given process water will be reused. However, this is subject to confirmation during the project design stage and findings addressed during the environmental impact assessment.

Uncontrolled runoff from disturbed areas associated with the TSF expansion may contribute to increased turbidity, heavy metals, residual processing reagents 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. Sediment traps and dust suppressants will be used to minimize potential impacts and contain surface water within the construction site.

Groundwater discharge volumes from the mine portal is 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. A hydrogeological and water balance model were being developed at the time of writing that would have provided further information on the groundwater systems. Flow meters will be installed to measure the volume of mine water discharge. In addition, the project will need to investigate whether groundwater drawdown will impact on community water resources.

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 continuing to erode is proposed in the conceptual mine closure plan to be mitigated via revegetating exposed sites to minimize run-off. The conceptual mine closure plan (K92 Mining Ltd, 2019) 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. This design basis assumption will require careful consideration and further assessment during the environmental impact assessment process. General site inspections, surface water monitoring and rehabilitation monitoring is proposed to continue post-closure.

20.4.2 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, 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.

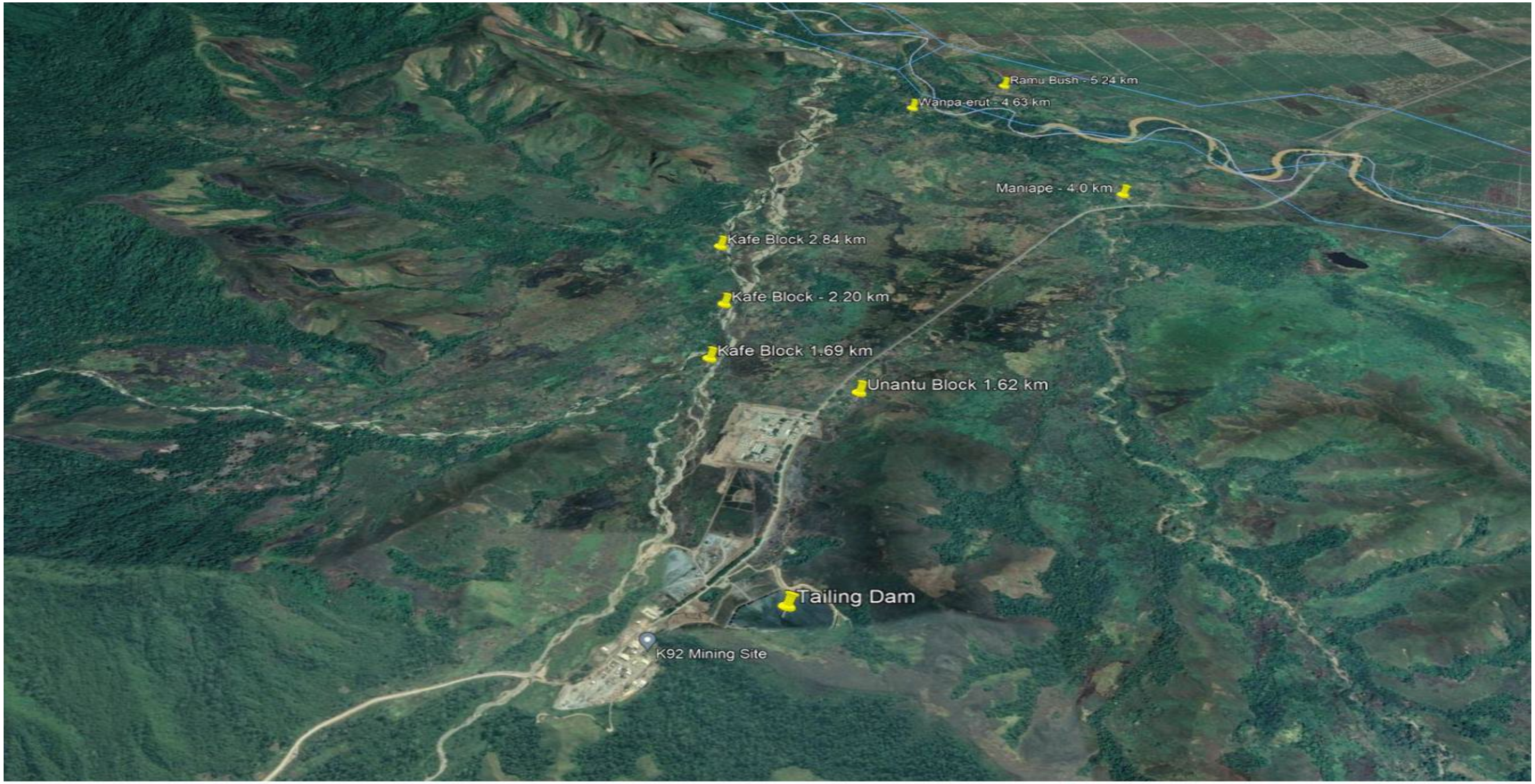
Landowner

While Landowner identification and social mapping would be required to extend the ML approximately 100 metres it is anticipated that there would be no change to the recognized Landowners. It is noted that the outstanding Land Titles Commission (TLC) determination of landholdings within the existing project will also apply to the extension of the ML. At this point some compensation payments, including distribution of a portion of royalties, are being accrued pending the determination from the TLC. A meeting was held in Goroka and the MRA has agreed to provide funding for the LTC.

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. The nearest communities (receptors) in the vicinity of the existing operations include eight small hamlets/communities near Kumain. Distances from the TSF to various hamlets are: Unantu Block (1.63 km), Maniape Hamlet (4.0 km), Ramu Bush (5.24 km), Wanpaerut (4.63 km), Kafe Block (2.84 km), Kafe Block (2.20 km), Kafe Block 2 (1.69 km) and Block opposite White House (0.63 km), as well as nearby Kokomo Bilimoia and Kokomo Pomasi Settlements at Kokomo (Figure 20.4.1).

Figure 20.4.1 Communities in the Mine Vicinity



20.4.3 Greenhouse Gas Emissions

Greenhouse gas (GHG) emissions associated with the stage 3 mine expansion are expected to reduce significantly based on GHG / tonne milled. K92 is committed to transitioning the haulage fleet to electric powered. Further initiatives including the improved materials handling design (reduction in TKM's via orepasses), upgrading of the OHPL, and increased reliance on power supply from PNG Power Limited will contribute to K92's objective of reducing the project's carbon footprint.

20.5 Reconciliation with PEA

20.5.1 Reconciliation with PEA

The PEA for the expansion project concluded that further investigations into the perceived environment impacts were required. Particularly surrounding the acid producing potential of the waste rock and proposed paste fill material. The only outstanding community issue that is yet to be resolved is the outstanding compensation payments due to existing appeals with the LTC determination. These issues are consistent with the findings from this assessment.

Additional environmental and community assessments required for the expansion project to support the major amendment of the existing environment permit are outlined in Section 20.6

20.6 Management Plans

20.6.1 Environment and Community Plan

The existing operations currently operates under the CEPA-approved 2020-2022 environment management plan (EMP). The purpose of the EMP is to satisfy conditions under the existing Environment Permit EP-L3(34), and to provide a policy framework, K92 Mining management commitments, and monitoring and improvement actions necessary to prevent, mitigate or manage environmentally degrading conditions resulting from the existing operations. Under the EMP, there is a series of sub-plans, which provides procedures for the monitoring and management of the following aspects:

- Soil erosion and sediment control.
- Site run-off management.
- Acid rock management.
- Progressive rehabilitation.
- Water resources and aquatic fauna management.
- Chemical and hydrocarbon storage and spillage.
- Emergency procedure and response.

- Solid and liquid waste management.
- TSF integrity monitoring.
- Air quality, dust and noise.
- Terrestrial flora and fauna management.
- Prohibited activities enforcement.
- Social, economic, cultural and archaeological.
- Water abstraction and sewage management.
- Hazard analysis management.
- Compliance monitoring of rivers and creeks.

Routine environmental monitoring as part of the EMP will continue to provide relevant baseline data to assess existing operational impacts, provided it is conducted with a high degree of quality assurance and quality control.

The K92 Mining environment team will be involved in reviewing environmental management plans developed by contractors for the Kora Expansion Project activities to check they are consistent with the EMP 2020-2022 (or updates thereof) and to identify the need for further measures to be included in the contractor EMPs to minimize the environmental impact of the expansion project.

20.6.2 Environmental Management System

The existing operations maintains an ISO 14001 accredited environment management system (EMS) that is integrated with a safety management system based on AS/NZS 4801. The key elements of the EMS are:

- An integrated safety and environment Policy, supported by senior management, signed by the site General Manager, and supported through a dedicated Environment team.
- A risk assessment process, supported by training, which identifies environment aspects and hazards.
- Environmental compliance requirements available to all site personnel via the site intranet.
- Clear objectives as stipulated in the EMP.
- Defined resources, roles, and responsibilities for environment management.
- Relevant training, awareness and procedures developed.

- A functional document control procedure.
- Standard operating procedures including emergency preparedness and response procedures.
- An auditing program to evaluate compliance to procedures and/or environment permit requirements including external auditing.
- A clear process for management of non-conformance, and corrective or preventative actions – through site action tracker, hazard registers, and incident reporting.
- Record keeping process clearly aligned with the stipulated expectations from CEPA.
- Review process of key documents by senior management prior to being finalized for submission to external parties such as CEPA.

20.6.3 Stakeholder Engagement Plan

K92 Mining's community relations team will prepare a Stakeholder Engagement Plan for the Kora Expansion Project. The objectives of the Stakeholder Engagement Plan will be to:

- Establish a knowledge base regarding community perspective of the implications of the expansion project. This enables a base from which recommendations can be made to the project implementation team to avoid possible work stoppages.
- Identify key landowners, assess their influence (for and against the expansion project) and develop strategies for engagement.
- Identify key landowners, assess their influence (for and against the expansion project) and develop strategies for engagement.
- Develop, in conjunction with the expansion project technical team, clear and authorized message tracks to enable Free, Prior, and Informed consent regarding the expansion project.
- Manage issues and expectations that may arise through engagement with concerned landowners and stakeholders.
- Understand the communities' key concerns regarding the expansion project to inform the impact assessment for the EIS.

In addition, K92 Mining's community relations team will prepare a Communications Pack for the Kora Expansion Project which includes a presentation about the expansion project, fact sheet and frequently answered questions (FAQ) sheet to guide community engagement for the expansion project.

20.6.4 Environment and Community Performance

Weekly and monthly internal reports on the results of routine environmental monitoring activities are currently produced and provided to K92 Mining management. This includes periodic internal and external audits that assess compliance with the EMP. A complaint and incident register is maintained whereby all complaints received are investigated and all affected parties are notified either in writing or in person. Complaints received from host communities are channelled to the site community affairs team to facilitate investigations.

The 2020 EMP (K92 Mining Ltd, 2020) states that complaints are processed in a consultative manner whereby status updates of the investigation and the timeframe for a resolution is provided within 24 hours to the complainant. If complaints arise from a contractor (or a subcontractor), they will provide full assistance to K92 Mining to properly investigate nature of the complaint or incidents. There have been no key environmental performance complaints regarding the existing operations that are relevant for the expansion project.

20.7 Closure Plan

The EIS as part of the major amendment to the existing environment permit will be required to demonstrate that technically feasible and economically viable means exist to decommission and rehabilitate the expansion project and present this in a conceptual rehabilitation and mine 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.

Based on the current conceptual mine closure plan, K92 Mining 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:

- Managing AMD issues post closure.
- Maintaining TSF integrity.

- Stability of underground stopes to reduce cave in risk.

20.7.1 Indicative Schedule

Preparation of the EIS to support a major permit amendment is anticipated to take **three to four months** from completion of the technical studies. The CEPA review and approval process typically takes **at least six months**, with another month to grant the permit amendment following approval in principle. Table 20.7.1 provides an indicative schedule that includes proposed timeframes for the technical studies.

Table 20.7.1 Indicative Schedule

Activity	Month																							
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
Initial notification to CEPA																								
Prepare and submit major permit amendment application																								
Prepare and submit EIR																								
Presentation to CEPA																								
CEPA acknowledgement and response to EIR																								
Technical studies																								
Site visit*																								
Scope studies																								
Engage with specialists																								
Surface water (hydrology)																								
Geochemistry																								
Groundwater																								
Aquatic ecology																								
Terrestrial ecology																								
ESIA																								
Prepare impact assessment																								
Submission of EIS																								
CEPA assessment*																								
CEPA review and acceptance of EIS																								
Independent peer review																								
EIS roadshow																								
Provision of additional EIS information																								
Environment Council																								
Draft Environment Permit conditions																								
Ministerial approval in principle																								
Granting of amended Environment Permit																								

***These schedules assume CEPA meets its statutory timeframes for the provision of feedback on the EIR.**

21.0 CAPITAL AND OPERATING COSTS

21.1 Mine Costing Basis of Estimate

Entech was engaged by K92 to assist with the mining cost estimation for the FS mine plan. 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 FS 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 manning requirements. Entech reviewed all K92 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 and dewatering pumps were sourced from manufacturer quotations. The vertical development schedule was provided to a mining contractor, who provided a budget pricing estimate for these activities.

A contractor request for quotation (RFQ) was undertaken to check the owner cost estimation built up by K92 and Entech. Two underground mining contractors were given a preliminary mining schedule with similar physicals to the FS schedule and provided budget pricing based on this schedule. The RFQ process produced a comparable mining cost to the estimate completed by K92 and Entech.

The K92 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 the high voltage (HV) electrical power system (K92 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 Costs

This section provides a summary of the underground mining capital costs.

21.2.1 Capital Decline Development

The decline development costs include allowances for the following:

- Costs associated with jumbo development.
- Costs associated with ground support.
- Costs associated with the loading and hauling of the decline 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.

The associated costs include:

- Costs associated with jumbo development.
- Costs associated with ground support.
- 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 1.3 m – 5.0 m diameter raise-drilled ventilation rises, escapeways, ore passes, 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 21.2.1 shows the breakdown of the infrastructure capital. Paste plant capital costs are captured separately in Section 21.6.

Table 21.2.1 Infrastructure Capital

Description	Unit	Value
Safety		
Refuge Chambers (20-man)	USD (M)	0.1
Refuge Chambers (4x8-man)	USD (M)	0.2
Ventilation		
Primary Fans (MTV)	USD (M)	3.9
Electrical		
WT114	USD (M)	0.5
Quad Booster Pump	USD (M)	0.1
8 x 55kW fans	USD (M)	0.6
Sub-Station Purchases	USD (M)	4.2
Infrastructure		
Escapeway Ladders	USD (M)	2.3
Ventilation Walls	USD (M)	0.6
Total	**USD (M)**	**12.6**

21.2.7 Summary of Capital Costs

The estimated capital costs are summarized in Table 21.2.2.

Table 21.2.2 Mining Capital Cost Totals

Description	Unit	Value
Infrastructure	USD (M)	12.6
Decline	USD (M)	22.1
Cap Access	USD (M)	5.9
Ventilation	USD (M)	30.2
Escapeway	USD (M)	2.2
Other Lateral Development	USD (M)	32.6
Fleet	USD (M)	32.3
Operators and Maintenance	USD (M)	35.5
Capital Mine Services	USD (M)	14.3
Capital Mine Overheads	USD (M)	11.9
Total Capital	**USD (M)**	**199.6**

A breakdown of the capital unit costs is shown in Table 21.2.3.

Table 21.2.3 Mining Capital Unit Costs

Description	Unit	Value
Infrastructure	$ / t ore	2.05
Decline	$ / t ore	3.59
Cap Access	$ / t ore	0.96
Ventilation	$ / t ore	4.91
Escapeway	$ / t ore	0.36
Other Lateral Development	$ / t ore	5.30
Fleet	$ / t ore	5.24
Operators and Maintenance	$ / t ore	5.76
Capital Mine Services	$ / t ore	2.32
Capital Mine Overheads	$ / t ore	1.94
Total Capital Cost	**$ / t ore**	**32.44**

21.3 Mining Operating Costs

This section provides a summary of the underground mining operating costs. Operating paste plant costs are captured separately in Section 21.6.

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 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 21.3.1.

Table 21.3.1 Mining Operating Cost Totals

Description	Unit	Value
Operating Access	USD (M)	8.2
Ore Drive	USD (M)	35.5
Stope	USD (M)	133.4
Operators and Maintenance	USD (M)	67.6
Operating Mine Services	USD (M)	30.2
Operating Mine Overheads	USD (M)	28.4
Surface Haulage	USD (M)	37.1
Grade Control	USD (M)	18.1
Total Operating	**USD (M)**	**358.5**

A breakdown of the operating unit costs is shown in Table 21.3.2.

Table 21.3.2 Mining Operating Unit Costs

Description	Unit	Value
Operating Access	$/t ore	1.33
Ore Drive	$/t ore	5.77
Stope	$/t ore	21.68
Operators and Maintenance	$/t ore	10.98
Operating Mine Services	$/t ore	4.92
Operating Mine Overheads	$/t ore	4.62
Surface Haulage	$/t ore	6.02
Grade Control	$/t ore	2.94
Total Operating	**$/t ore**	**58.27**

21.4 Process Plant and Infrastructure Capital Costs

21.4.1 Capital Cost Estimate Work Breakdown Structure

The capital cost estimate for the Kora Expansion Project has been compiled by Lycopodium Minerals Pty Ltd with input from K92.

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 expressed in United States Dollars (USD) unless otherwise stated and based on 2Q22 pricing. The estimate is deemed to have an accuracy of +15% / -5%.

21.4.2 Capital Estimate Summary

The capital cost estimate is summarized in Table 21.4.1. The estimate is presented for the initial capital cost based on a 1.2 Mtpa throughput.

Table 21.4.1 Capital Estimate Summary (USD, 2Q22, +15/-5%)

Main Area	USD (M)
000 Construction Distributables	17.91
100 Treatment Plant Costs	39.63
200 Reagents and Plant Services	8.78
300 Infrastructure	33.96
500 Management Costs	18.55
600 Owner's Project Costs	3.69
Subtotal	**122.52**
Contingency	14.88
Taxes & Duties	Excl.
Escalation	Excl.
Estimated Total	**137.40**

21.4.3 Estimate Currency and Base Date

The estimate is expressed in USD based on prices and market conditions as of second quarter 2022 (2Q22).

Foreign currency exposure is shown in Table 21.4.2 below.

Table 21.4.2 Foreign Currency Exposure

Currency	Exchange Rates	USD Portion $(000)	Percentage of Capital Estimate
USD	1.00	80,917	59%
USD to AUD	1.32	46,190	34%
USD to EUR	0.82	7,907	6%
USD to PGK	3.57	0	0%
USD to GBP	0.70	581	0%
USD to ZAR	16.00	1,812	1%

21.4.4 Estimate Basis

The capital cost estimate was prepared in accordance with Lycopodium's standard estimating procedures and practices. The estimate basis and methodology are summarized in Table 21.4.3 below.

Table 21.4.3 Capital Cost Estimate Methodology

Description	Basis
Bulk Earthworks	Volume for bulk earthworks engineered by Lycopodium.
Detailed Excavation	Allowances for under pad excavation and backfill to prepare site for concrete works.
Concrete Installation	Quantities based on study engineering, reference projects and estimated structures.
Structural Steel	Quantities based on study engineering and reference projects.
Platework & Small Tanks	Platework items as per the mechanical equipment list.
Tankage Field Erect	Tanks as per the mechanical equipment list.
Mechanical Equipment	Items as per the mechanical equipment list. Formal budget enquiries with datasheets and/or specifications for the major mechanical items. Costs for minor items taken from the Lycopodium (recent) database.
Plant Piping General	Factored from mechanical costs.
Overland Piping	Size and specification based on engineering selection. Quantity based on site layout. Other rates taken from the Lycopodium database.
Electrical General	Quantities derived from engineering design and site layout. Electrical equipment priced for the project. Bulks and installation costs drawn from a combination of recent database and budget pricing.
Electrical HV	Quantities derived from engineering design and site layout. Electrical equipment priced for the project. Bulks and installation costs drawn from a combination of recent database and budget pricing.
Commodity Rates – General	Based on specific contractor enquiries with indicative drawings.
Installation Rates – General	Based on specific contractor enquiries with indicative drawings.
Large Cranage	Hire of a 250t crawler crane for major lifts.
Freight General	Combination of freight tons and percentages.
Contractor Mobilisation / Demobilization	Based on Budget Quotation Requests (BQRs).
EPCM	Resource based estimate for the EPCM controlled scope.
Owner's Costs	
Owner's Project Costs	K92 estimate.
Construction Accommodation	Owners team, EPCM personnel and contractors expats at $25 / day.
Project Insurances and Permits	Part of Owner's project costs.
Plant pre-production expenses	Estimated as part of Owner's costs.
Opening Stocks, First Fill Reagents and Consumables	Estimated from consumption rates and costs as part of Owner's costs.
Spares	% Allowance.
Duties and Taxes	Excluded.
Escalation	Excluded.

21.4.5 General Estimating Methodology

General arrangement drawings and a 3D model have been produced with sufficient detail to permit the assessment of the engineering quantities for earthworks, concrete, steelwork, mechanical and electrical for the crushing plant, processing plant, conveying systems and infrastructure.

Unit rates that reflect the current market conditions have been established for bulk materials, capital equipment and labour from region specific budget quotation requests (BQRs) and benchmarked against projects that are either currently under construction or recently completed.

Budget pricing for equipment and infrastructure facilities were obtained from suitable suppliers and contractors.

21.4.6 Engineering Status

The design status varies on a facility-by-facility basis, from recently completed designs, modified construction, and as-built drawings of current and past project facilities, to initial concept drawings.

The key process and engineering design criteria used for equipment selection in the development of the capital cost estimate is described in Section 21.3.4.

21.4.7 Quantity Development

The Project works were quantified to represent the defined scope of work and to enable the application of rates to determine costs. Allowances for compaction, waste, rolling margin and the like are included in the build-up of unit costs.

Quantity information was derived from a combination of sources and categorized to reflect the maturity of design information as follows:

- Study engineering that includes quantities derived from project specific engineering for the purpose of the study. Includes equipment lists and 3D modelled facilities.
- Reference projects that include quantities drawn from previously constructed projects or detailed designs, and by exception have been adjusted to suit the equipment sizing and layout specific to this project.
- Estimates that include quantities derived from sketches or redline mark-ups of previous project drawings and data, compiled by estimating.
- Factored quantities derived from percentages applied as a factor from previous estimates or projects.

The derivation of quantities within these categories by percentage is provided in Table 21.4.4 weighted by bulk quantity.

Table 21.4.4 Derivation of Quantities

Classification	Quantity	Unit	Study Engineering List / Model %	Reference Projects (Adjusted) %	Estimated %	Factored %
Plant Earthworks	486,580	m^3	100%	-	-	-
Plant Concrete	4,540	m^3	-	95%	5%	-
Structural Steel	559	t	-	90%	10%	-
Platework	233	t	-	90%	10%	-
Field Erected Tanks	161	t	-	100%	-	-
Mechanical Equipment	262	ea.	100%	-	-	-
Conveyors	199	m	100%	-	-	-
Piping – Plant	1	lot	-	-	-	100%
Piping – Overland	4.6	km	100%	-	-	-
E & I Plant & Infrastructure	1	lot	80%	-	20%	-
Steel Buildings	1,600	m^2	100%	-	-	-

21.4.8 Pricing Basis

Estimate pricing was derived from a combination of the following sources:

- Budget Pricing – Market pricing solicited specifically for the project estimate.
- Database – Actual costs from similar projects that have recently been constructed or were under construction at the time of the estimate and are less than six months old.
- Estimated – Historical database pricing older than six months, escalated to the current estimate base date.
- Factored – Factors derived from percentages applied as a factor from previous estimates or projects.

Table 21.4.5 summarizes the source of pricing by major commodity, weighted by value of the direct permanent works (excluding temporary works, construction services, commissioning assistance, EPCM costs and contingency), including supply and installation.

Table 21.4.5 Sources of Pricing

Classification	Subtotal USD (M)	Budget Pricing %	Database %	Estimated %	Factored %
A Architectural	6.72	88%	12%	-	-
B Earthworks	8.78	-	100%	-	-
C Concrete	5.53	100%	-	-	-
E Electrical	27.86	80%	20%	-	-
L Platework	4.11	100%	-	-	-
M Mechanical	25.22	80%	20%	-	-
P Piping	7.99	-	23%	-	77%
S Steelwork	4.83	100%	-	-	-
S1 SMP Indirects	6.53	100%	-	-	-
U Owners Costs	3.92	100%	-	-	-
V EPCM	17.98	100%	-	-	-
Z General	3.05	-	100%	-	-
Pricing Derivation	**122.52**	**76%**	**19%**	**0%**	**5%**

Pricing is categorized by the following cost elements, as applicable, for the development of each estimate item.

Bulk Materials

This component covers all materials normally purchased in bulk form, for installation on the project. Costs include the purchase price ex-works, off-site fabrication (if applicable), and over-supply for anticipated wastage.

Plant Equipment

This component represents prefabricated, pre-assembled, off-the-shelf type of mechanical or electrical equipment items. Pricing is inclusive of all costs necessary to purchase the goods ex-works, generally excluding delivery to site (unless otherwise stated) although does include operating and maintenance manuals. Vendor representation and commissioning spares have been allowed for separately in the estimate.

Installation

This component represents the cost to install the plant equipment and bulk materials on site or to perform site activities. Installation costs are further divided between trade labour, equipment, and contractors' distributables.

The labour component reflects the cost of the trade labour workforce (excluding management, supervision, and other onsite support staff) required to construct the Project scope. The labour cost is the product of the estimated work hours spent on site multiplied by the cost of labour to the contractor inclusive of overtime premiums, statutory overheads and payroll burden, and contractor margin.

The equipment component reflects the cost of the construction equipment and running costs required to construct the Project. The equipment cost includes small tools, consumables, PPE, and the applicable contractor's margin.

Contractors' distributable costs encompass the remaining cost of installation and include items such as off-site management, onsite staff and supervision above trade level, site facilities, cranes (up to 100 t) and crane drivers, mobile equipment, scaffold, mobilization, and demobilization, R&Rs, meals and accommodation costs, durations costs general, and the applicable contractors' margin.

21.4.9 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.

21.4.10 EPCM 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.

The estimate for EPCM services costs has been based on a preliminary manning schedule for the anticipated project deliverables and project schedule. The resulting EPCM cost estimate is consistent with other projects of this nature in terms of the percentage of the plant capital cost.

The engineering design component of the EPCM estimate for the home office is based on a calculation of required manning levels to complete the Project and benchmarked against Lycopodium experience on similar projects.

Engineering, design, and procurement is assumed to commence shortly after Client Board approval.

21.4.11 Owner Costs

As part of the project development, K92 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:

- 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.

21.4.12 Spares

A value for the Project spares has been estimated by Lycopodium and included in the estimate.

21.4.13 Duties, Taxes, and Insurances

Government Duties and Taxes

The capital estimate excludes government taxes and duties.

21.4.14 Contingency

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed project cost. The contingency does not cover scope changes, design growth, etc. or the listed qualifications and exclusions.

Contingency has been applied to the estimate as a deterministic assessment by assessing the level of confidence in each of the defining inputs to the item cost being engineering, estimate basis and vendor or contractor information. It should be noted that contingency is not a function of the specified estimate accuracy and should be measured against the Project total that includes contingency.

A contingency analysis has been applied to the estimate that considers scope definition, materials / equipment pricing and installation costs. Contingency applicable to various Owners inputs have been specified by K92.

The resultant overall contingency for the Project on the capital cost estimate is 12.14%.

21.4.15 Escalation

There is no allowance for project escalation in the capital estimate.

21.4.16 Qualifications and Assumptions

The capital estimate is qualified by the following assumptions:

- The estimate base date regarding project pricing is second quarter 2022 (2Q22).
- Contingency has been allowed based on the quality of the information, however no allowance for escalation has been included.
- Prices of materials and equipment with an imported content have been converted to USD at the rates of exchange stated previously in this document. All pricing received has been entered into the estimate using native currency.
- Contractor rates and distributables include mobilization / demobilization, recurring costs, direct and indirect labour, construction equipment, construction crane (up to 100 t), materials, materials handling and offloading, temporary storage, construction facilities, off site costs, insurances, flights, construction fuel, tools, consumables, meals, and PPE.
- Potable water and construction raw water supply will be provided by the client and available at site for the use by contractors.
- Site construction offices will be containerized units only, with the intention that the permanent administration building construction schedule will be accelerated.
- 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.
- The estimate allows for aggregate and sand for concrete batching to be provided by the concrete contractor and are assumed available locally to the Project site.
- The estimate allows for all reinforced bar and mesh for construction to be provided by the concrete contractor. Free issue of materials would be a project capital opportunity.
- There is no allowance for blasting in the bulk earthworks.

- The estimate allows for supply of structural steel and platework from South East Asia.
- Meals and accommodation for the Owner and EPCM teams and senior contractor management have been allowed in the estimate.
- Domestic flights for EPCM team to be provided by Owner.
- Project spares are a percentage allowance of the mechanical supply cost based on similar size projects.
- PLC programming for the process plant has been allowed for in the E&I estimate.
- Communications network and data for construction facilities to be free issued by the Owner.
- No allowance for security infrastructure or personnel, assumed provided by Owner.

21.4.17 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Owner's Team Project and expenses.
- 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

The process plant operating costs have been compiled with input from K92 and from a variety of sources, including:

- Labour pay rates with on-costs and manning as advised by K92.
- Power cost as advised by K92.
- Consumables prices from supplier budget quotations, K92 advice or the Lycopodium database.

- Modelling by Orway Mineral Consultants (OMC) for crushing and grinding energy and consumables, based on Kora average ore characteristics determined in the metallurgical testwork.

- Reagent consumptions based on metallurgical testwork results and K92 advice.

- First principle estimates based on typical operating data and standard industry practice.

The exchange rates used for the preparation of the operating cost estimate are the same as those used for the capital cost estimate as presented in Table 21.4.2.

The operating cost estimate by cost centre is presented in Table 21.5.1 and is deemed to have an accuracy of +15% / -10% based on pricing as at 1Q22. The process operating cost includes all direct costs to produce gold doré and saleable concentrate for the Project.

Table 21.5.1 Process Plant Operating Cost (USD, 1Q22, +15% / -10%)

Cost Centre	Kora Ore		
	USD/t	Fixed	Variable
		USD'000/y	USD/t
Power	5.92	2,625	3.74
Operating Consumables	3.81	370	3.50
Process and Maintenance Labour	3.29	3,947	-
Maintenance Materials	1.57	1,430	0.38
Total Processing (Variable)	**14.59**	**8,371**	**7.61**
Laboratory	0.34	411,434	-
Total	**14.93**	**8,783**	**7.61**

21.5.1 Summary Battery Limits

The battery limits for the processing operating costs are as follows:

- Ore on the ROM pad.

- Tailings discharge to TSF and/or paste plant.

- Gold doré in safe on site.

- Saleable concentrate on site.

21.5.2 Qualifications

The following items have been excluded from the operating cost estimate:

- All sunk costs.
- Government monitoring and compliance costs.
- All K92 head office costs and corporate overheads.
- Withholding taxes and other taxes, such as GST or VAT.
- Escalation.
- Financing costs.
- Foreign exchange fluctuations.
- Interest charges.
- All costs associated with areas beyond the battery limit of the study.
- Land or crop compensations costs.
- Licence fees.
- Royalties.
- Contingency.
- Labour union fees.
- All mining and exploration costs, including mining services.
- Maintenance costs of all mine, haul and plant access roads.
- Doré transport and insurance costs.
- Gold refining costs.
- Concentrate marketing, transport, and insurance costs.
- Concentrate smelting and refining costs.
- Tailings storage costs, including future lifts and rehabilitation.
- Paste plant operating costs for generating and pumping paste back-fill into the underground workings.
- External government required tailings monitoring and compliance costs.

- Any environmental rehabilitation or closure costs.

The process plant operating cost estimate is based on the following items:

- The power cost estimate has been based on grid power at a unit cost of USD0.170/kWh.
- The plant maintenance cost allowance has been factored from the capital supply cost using factors from the Lycopodium database.

21.5.3 General and Administration Costs

The general and administration (G&A) costs have been compiled by K92 and is based on the following items:

- Mine technical services.
- 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 29.36 /t ore.

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 21.6.1 and is based on the following exchange rates.

- USD1.00 = AUD1.32 (Australian Dollar).
- USD1.00 = EURO0.82 € (Euro).

Table 21.6.1 Estimated Kora Paste Backfill System Capital Cost Estimate (AACE Class 3 Estimate)

Location	Component	PFS Estimate (USD)
Process Plant	Design and Management	$879,304
	Equipment	$4,860,851
	Installation	$2,477,533
	Process – 800 Portal overland piping and Instruments (inc. Supply and Install)	$3,424,171
	Commissioning	$202,614
800 Portal	Design and Management	$726,212
	Equipment	$4,014,545
	Installation	$2,651,130
	800 Portal -1205 level piping and Instruments (inc. Supply and Install)	$3,271,418
	Commissioning	$167,338
1205 Level	Design and Management	$734,684
	Equipment	$4,061,382
	Installation	$2,326,148
	1170 Level – 1st stope Reticulation and Instruments (inc. Supply and Install)	$826,164
	Commissioning	$169,290
EPC Margin (15%)		$4,618,918
Logistics (7% of equipment costs)		$1,315,526
Contingency (10%)		$3,210,831
Grand Total		**$39,938,059**

21.6.3 Scalability Capabilities

It is understood that K92 have some concern around the proposed paste system constraining the ability for the Kainantu operation to expand production rates from the design rate of 1.2 Mtpa (as assumed in this study) to 1.7-1.8 Mtpa. As part of a PEA investigation being carried out in conjunction with this investigation, the higher production rate was considered. It is the opinion of the author of this section that most components of the system can simply be duplicated to achieve the increased production. Analysis presented in the PEA indicates that to achieve the increased production rate, it would be necessary to increase the capacity of the four paste pumps as well as increasing the size of the 1205 Level Paste mixer.

The paste pump increase is expected to include, increasing the Process Plant and 800 Portal paste pumps from TZPM 1200 to TZPM 1600 pumps and upgrades to the DHC26300 pumps (on the 1170 Level). The net cost increase for the larger capacity pumps is USD0.8M.

Should K92 wish to proceed with the paste system, while maintaining the opportunity to expand the system later, its recommended that these larger pumps be specified from the beginning.

21.6.4 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 at 140 tph (dry solids) and ≥59% solids.
- Potable water delivered to each of the paste plant locations, primarily for safety showers.
- Raw water of suitable quality for gland seals delivered to the bund of the paste plant at the various locations.
- Raw water of suitable quality for vacuum pump seals delivered to the bund of the paste plant at the process area.
- The processes have assumed that compressed air (from the mine network) is available at 700 kPa at both the 800 portal and 1170 Level sites. An air dryer is included in the paste plant infrastructure at these locations.
- Flocculant is required to be delivered to the paste plant filter at the Process plant at a rate of 2800 l/hr. Note, a provision to upsize the thickener flocculant plant is included in the CAPEX estimate.
- Fibre communications network connection between the three components in the process and to reticulation system dump valves and instrumentation located throughout the mine.
- Ground Granulated Blast Furnace Slag delivered in 20 ft containers that are fitted with a 'blowout' diaphragm.
- GP Cement delivered in 20t ISO containers suitable for transport underground (Provision for 5 x 20t ISO containers included in costing).

Outgoing:

- Wastewater discharged at the process plant.

- Out of specification paste discharged to the final tailings box (assumed to be located adjacent to the tailings thickener presented in Section 18.4.3).
- Flush paste from the Process Area – 800 Portal transfer line rejected into the final tailings box (assumed to be located adjacent to the tailings thickener presented in Section 18.4.3).
- Paste discharged into a stope, where the capital cost estimate includes provision for 350 m of horizontal piping and 60 m of borehole drilling and casing. All reticulation supply and installation costs beyond this point are allocated to operating costs.

The services required at each plant location is presented in Table 21.6.2

Table 21.6.2 Incoming Battery Limit Inputs

Service	Units	Process Plant	800 Portal	1205 / 1170 Level
Power (installed) (kW)	kW	1434	1031	2286
Process Water Consumed / Peak (m^3/h)	m^3/h	40 / 133	10 / 50	5 / 50
Potable Water (m^3/h)	m^3/h	5	5	5
Compressed Air @ 700 kPa	-	Not Reqd.	Not Reqd.	Mine air assumed
Flocculant (Consumption / Peak demand)	kg/h / kg/day	6 / 120	-	-
Slag (Consumption / Peak demand)	t/h / t/day	-	3.3 / 200	-
GP (Consumption / Peak demand)	t/h / t/day	1.5 / 20	-	-
1205 Level Development*	m^3			15,000
1170 Level Development*	m^3			5,000
Paste Reticulation System Underground Cuddy's*	m^3			3,235

*** Costs associated with these items are included in the underground development, included in studies by others**

Table 21.6.3 Outgoing Battery Limits

Service	Units	Process Plant	800 Portal	1205 Level
Filtrate/waste slurry return to Tailings Thickener area	m^3/h	66	-	-
Waste Paste on Start-up and Shutdown / Flushing	m^3/h	125	-	-
Paste Discharge	m^3/h	-	-	125

21.6.5 Exclusions

- No provision has been made for ground improvement works at the 800 Portal or excavation and support of suitable underground cavities. For ground improvement at the 800 Portal area, it is recommended that K92 make a provision of USD0.5M.
- Provision has been made for thrust blocks and supports for overland and underground pipe supports, but no provision is made for any overland pipe earthworks, trenching, roadway underpasses etc. Provision is made for supply and installation of two pipeline dump sumps, with bog out ramp in compounds between the process area and 800 Portal.
- No Provision has been made for the supply of services outside of battery limits (i.e. no provision is made for supplying the services noted in Table 21.6.2 to the edge of the relevant paste plant bund).
- 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 Process-800 Portal pipeline corridor and dump sumps is excluded.
- Supply of concrete is excluded (i.e. it is assumed that the client would free issue 1,200 m^3 of concrete 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 Introduction – Operating Cost Model

This section presents the results of the operating cost analysis for the Kora paste backfill system. This section considers all backfill system operating costs inclusive of:

- Underground and surface personnel.
- Consumables such as binder, flocculant, water, and power.
- Plant maintenance.
- Vehicles and machinery.
- Reticulation pipe extensions.
- Underground containment bulkheads.
- As well as quality control and training / management costs.

For the purpose of this analysis, the operating costs have been determined for each unit volume of in situ fill. This is understood to be consistent with the projected fill volume requirements, which relate to the in-situ void space requiring fill.

21.7.2 Operating Costs - Operational Fill Activities

When filling operational stopes as an integral part of the Kora mining operations the site model inputs are summarized in Table 21.7.1.

Table 21.7.1 Base Case Assumptions for Estimating Fill Plant Operating Costs

Item	Units	Value
Average Stope Volume	m^3	2,750
In situ void filling rate	m^3/h	120
Annual total Stope void filled	m^3/y	429,000
Average monthly void filled	m^3/m	36,000
Fill plant operator costs	USD/operator/y	10,000
Backfill Superintendent	USD/operator/y	60,000
Backfill coordinator cost + vehicle	USD/operator/y	35,000
Underground service crew	USD/operator/y	10,000
Surface Light Vehicle (inc. hire, maintain, fuel)	USD/operator/y	30,303
Surface Light Vehicle (inc. hire, maintain, fuel)	USD/operator/y	30,303
Underground Scissor Lift (inc. hire, maintain, fuel)	USD/y	75,800
Surface Binder Delivery Truck	USD/y	37,900
Underground Binder Delivery Truck	USD/y	68,220
Power cost	USD/kWh	0.17
Flocculant cost	USD/kg	3.8
GP Cement	USD/t	170
Ground Granulated Blast Furnace Slag	USD/t	327
Raw Water Cost	USD/m^3	0.1
Bulkhead Cost	USD/bulkhead	8,073

Based on past experience with similar systems and the unit rates presented in Table 21.7.1, the expected operating costs for the proposed paste system as summarized on Table 21.7.2.

Table 21.7.2 Estimated Operating Costs for the Proposed Paste System

Item	Units	Operating Cost (USD)
Fixed Operating Costs		
Plant Operating Labour (3 Surface Operators, 1 UG Operator, 1 Surface binder delivery driver, 1 UG binder delivery driver x 3 swings)	/y	$180,000
Fill Superintendent (2 total)	/y	$120,000
Fill Coordinator (3 total)	/y	$105,000
Underground Labour (5 Crew x 3 swings)	/y	$150,000
Surface LV (1 vehicles)	/y	$30,303
Underground Vehicle (2 vehicles)	/y	$60,606
Underground Scissor Lift (or Integrated tool carrier)	/y	$75,758
Binder Delivery Vehicles (1 x Surface 1 x UG)	/y	$106,061
Management Training, Consulting Services	/y	$210,000
Total Annual Fixed Costs (USD)		**$1,037,727**
Variable Costs		
Process Plant		
General Consumables (Thickener, Agitator tank, Centrifugal pumps, Auxiliaries)	/m^3	0.15
Plant repairs and Maintenance (Parts and Labour)	/m^3	0.19
Filter Maintenance (Parts and Labour)	/m^3	0.25
Paste Pump Maintenance (Parts and Labour)	/m^3	0.28
Power (937 kW consumed)	/m^3	1.33
Flocculant (40 g/t avg)	/m^3	0.19
Raw Water	/m^3	0.03
800 Portal		
General Consumables (Agitator tank, Centrifugal pumps, Slag System, Auxiliaries)	/m^3	0.11
Plant repairs and Maintenance (Parts and Labour)	/m^3	0.11
Paste Pump Maintenance (Parts and Labour)	/m^3	0.28
Power (850 kW)	/m^3	1.21
Slag (2.2% Slag addition, w/w)	/m^3	9.00
Slag Delivery	/m^3	0.09
Raw Water	/m^3	0.00
1205 Level		
General Consumables (Mixer, Centrifugal pumps, GP System, Auxiliaries)	/m^3	0.15
Plant repairs and Maintenance (Parts and Labour)	/m^3	0.23
Paste Pump Maintenance (Parts and Labour)	/m^3	0.49
Power (480 kW)	/m^3	0.68
GP Cement (1.0% GP addition, w/w)	/m^3	2.15
GP Delivery	/m^3	0.16
Raw Water	/m^3	0.00
Underground Construction Materials (labour & Equipment inc. in Fixed Operating Costs)		
Underground Reticulation Materials ($140,000/level)	/m^3	1.70
Bulkheads (1 per stope)	/m^3	2.93
Overheads		
Quality Control	/m^3	0.08
Total Variable Costs	**/m^3**	**21.8**
Fixed + Variable Costs		$24.2
Contingency (5%)		$1.2/$m^3$
Total Estimated Operating Cost (USD)		**$25.4/$m^3$**

21.7.3 Upside Opportunity

The major contributor to the Operating cost is that for Ground Granulated Blast Furnace Slag (slag) at $9.0/m^3$. The Opex model assumes a unit slag supply price of $315 USD/t, which in comparison with imported GP Cement (unit cost of $170 USD/t) is very high. The reason for this high unit price is associated with the different logistics strategy. GP Cement is delivered to Lae in bulk (sea) tankers and blown into holding silos. Cement is drawn from these silo's and transported to site using ISO containers. At the time of writing this report slag wasn't commercially available in Lae and as such the assumed supply logistics involves loading shipping containers with the product and transporting these containers to site for unloading. The major contributor to slag pricing is Sea Freight and port costs, which equate to almost 60% of the total cost.

With additional infrastructure in place to facilitate bulk slag deliveries it is expected that the unit slag price may reduce to a value similar to that for GP cement (i.e. $170 USD/t). If slag can be supplied at a cost of $170 USD/t the unit operating cost for K92 paste is expected to reduce to $21.0 USD/m^3$.

21.7.4 Operating Costs - Tailings Disposal (Unexposed Paste)

A critical objective of the Kora paste system is disposal of tailings and as such it's important to understand the incremental operating cost associated with storing (additional) paste in historic workings at the Kainantu site using the Kora paste system.

Assuming that the operational activities account for all fixed costs on the system additional paste placed into remnant workings would not incur any further fixed costs and as such this component could be removed from the operational cost of any additional fill placed. This would reduce the unit operating cost by $2.5 USD/m^3$.

When filling stopes that are not required to be exposed the binder content can be reduced to just 2.0% (1.0% slag:1.0% GP). The reduction in slag content from 2.2% (for the average case) to 1.0% for the non-exposure case has a significant impact on the operating costs. This would reduce the operating cost associated with binder by a total of $5.1 USD/m^3$.

Finally, due to the relatively small stopes in the Kora orebody the bulkhead construction activities constitute a significant portion of the unit operating costs. However, if deposited into historic workings it is expected that larger voids would be selected ensuring that the void size per bulkhead would increase. Assuming that the void size per bulkhead was doubled the unit cost associated with bulkhead construction could be reduced by $1.6 USD/m^3$.

Combining the above, if additional paste is to be placed into historic Kainantu voids it is expected that the incremental unit operating cost of this fill would reduce to $16.0 USD/m^3$.

21.8 Tailings Storage Facility Capital Costs

The total estimated capital cost to increase the existing TSF capacity, ensuring sufficient volume is available based on the Kora Expansion, is calculated as USD$18.80 million, which includes a ± 30% contingency allowance for the embankment construction. Other capital costs (included in the above total) include researching tailings to use as a material to build the dam's embankment and increase the settled density of tailings deposited.

21.9 Tailings Storage Facility Operating Cost

The annual operating cost for the tailings management is estimated at USD$0.47 million (value per year does not consider the rate of discount). The highest components of the annual operating expenses are related to items such as consultants' activities and the cost required to conduct initiatives that can help to improve the settled density of tailings.

21.10 EIS Capital Cost Estimate

Costs for the scope of work are shown in Table 21.9.1. These can be refined following CEPA review of the EIR. Costs for each study have been approximated to the nearest $25,000.

Table 21.10.1 Indicative Cost Estimate for EIS

Activity	Cost Estimate (AUD)
Initial documentation to commence process and engage CEPA	50,000
Technical Studies	
Surface water (hydrology)	50,000
Geochemical and physical characterization of mine waste materials	50,000
Groundwater	50,000
Aquatic ecology	35,000
Terrestrial ecology	25,000
Social Studies	
Socio-economic baseline characterization	50,000
Studies management	30,000
EIS and EMP preparation	120,000
Closure and rehabilitation plan	35,000
Total	**$495,000**

These costs assume that the socio-economic impact assessment is part of the EIS preparation. As discussed under Section 20.2, it is advisable to allow for approximately $200,000 for CEPA's peer review, which is at the lower end of IPRs.

22.0 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 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 2022, with processing through the existing plant. Discounting has been applied 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 are summarized in Table 22.2.1.

Table 22.2.1 Key Model Inputs

Model Inputs	Source	Unit / Value
Base Currency		USD
Base Date		3rd Quarter 2022
Ore Processed over LOM	Entec	6.15 Mt
Metal Prices		
Gold	K92	USD1,600 / oz (fixed)
Copper	K92	USD4.00 / lb (fixed)
Silver	K92	USD20 / oz (fixed)
Recoveries		
Gold (total)	MMS	93.0%
Gold (gravity to dore)	MMS	15.0%
Copper	MMS	95.2%
Silver	MMS	80.0%
Concentrate copper grade	MMS	15.5%
Processing Plant Capacity (dry tonnes of ore)		
Existing Plant	K92	500,000 tpa
New Plant	Lycopodium	1,200,000 tpa
Royalties (deducted from gross revenue)	K92	2.0%
Levies (deducted from gross revenue)	K92	0.5%
Tax Rate	K92	30%
Depreciation		Not considered
NPV Discount Rate	K92	5%

22.2.1 Ore Processed

Total ore processed from 2022 to 2028 is 6.15 Mt.

Average head grade over LOM is:

- Gold 6.65 g/t
- Silver 17.53 g/t
- Copper 0.88%

22.2.2 Gross Revenue

Gross revenue from doré and concentrate (excluding treatment and refining charges and operating costs) is USD2,470M over the LOM, or USD401/t ore.

22.2.3 Treatment and Refining Charges

Total treatment and refining charges, including penalty element deductions equates to USD43/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 22.2.2.

Table 22.2.2 Operating Costs

Area	Source	Unit / Value
Mining (average over LOM)	Entec	USD58 / t ore
Processing Plant – Existing - Total (average over LOM)	K92	USD22 / t ore
Processing Plant – New - Fixed	Lycopodium	USD8.8M /year
Processing Plant – New - Variable	Lycopodium	USD7.61 / t ore
Processing Plant – New - Total (average over LOM)		USD15 / t ore
General & Admin - Total (average over LOM)		USD29 / t ore
Paste Plant - Fixed	Helinski	USD1.1M /year
Past Plant - Variable	Helinski	USD22.87 / m^3 paste
Paste Plant - Total (average over LOM)		USD25 / m^3 paste
TSF (average over LOM)	ATC Williams	USD0.90 / t tails
Transport & Insurance (average over LOM)	K92	USD3.07 / t ore
Total (average over LOM)		**USD116 / t ore**

22.2.5 Capital Costs

Pre-production capital expenditures are defined in Table 22.2.2. Sustaining capital for the Plant, Mining and TSF expansion costs have been phased over the life of the project and detailed in Table 22.2.3.

Table 22.2.3 Pre-Production Capital Expenditure

Item	Source	Total (US$'000)
Mine		Refer sustaining capital
Process Plant	Lycopodium	80,195
Power Station	Lycopodium	9,572
Power Supply	Lycopodium	13,815
Camp Upgrade	K92	5,466
Haul Road Bridges	K92	3,740
EPCM	Lycopodium	20,423
Owner's Costs	K92 / Lycopodium	4,200
Paste Plant	Helinski	39,938
Total		**177,349**

Table 22.2.4 Sustaining Capital Expenditure

Item	Source	Total (US$'000)
Mine	Entec	199,594
TSF	ATC Williams	18,804
TSF Closure Costs	ATC Williams	5,463
Total		**223,860**

22.2.6 Royalties

A royalty of 2.0% has been deducted from gross 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 3Q22 values.

22.3 Financial Model Results

The financial model indicates that the project has a post-tax Net Present Value (NPV) of USD586M at a discount rate of 5%.

Figure 22.3.1 shows the post–tax annual and cumulative cash flow for the project over the LOM.

As the model starts with a positive cash flow in 2022 due to revenue from the existing plant operation, the cumulative cash flow position is never negative.

Due to this initial positive cash flow, an Internal Rate of Return (IRR) is not published.

Figure 22.3.1 Cumulative Cash Flow

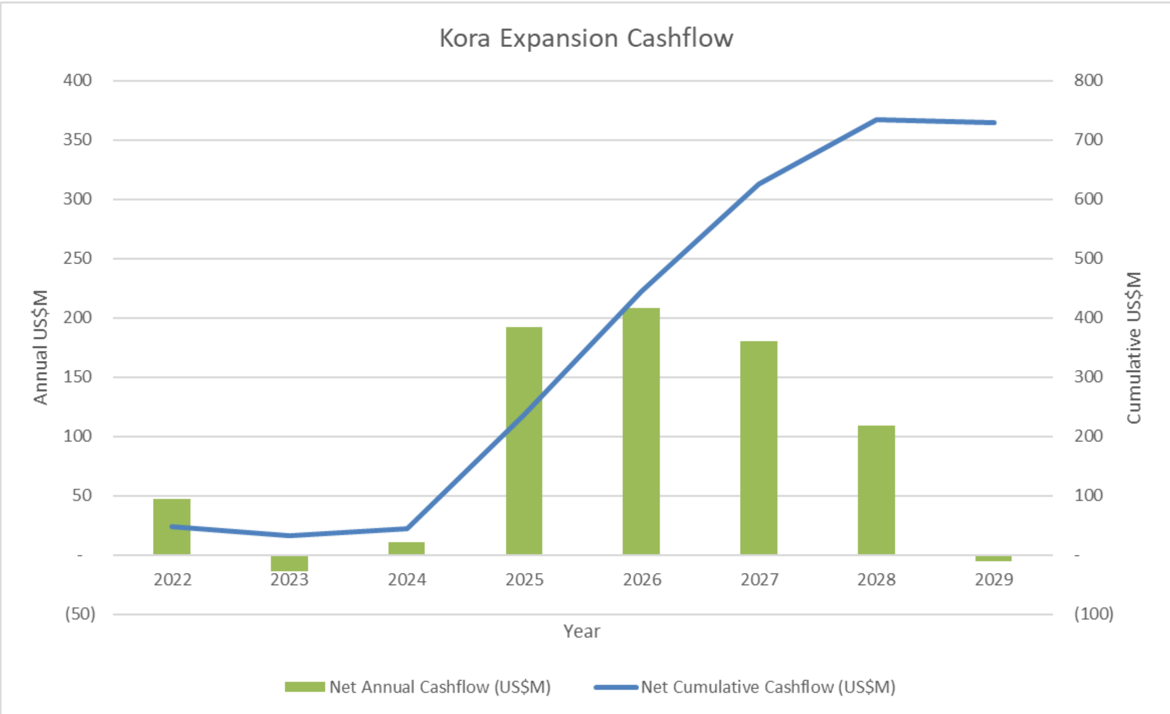


Table 22.3.1 shows a summary of the cash flow model for the project.

Table 22.3.1 Cash Flow Model Summary

	Source	Units	2022	2023	2024	2025	2026	2027	2028	2029	TOTAL
Period Ending			31/12/2022	31/12/2023	31/12/2024	31/12/2025	31/12/2026	31/12/2027	31/12/2028	31/12/2029	
Gross Revenue - After Treatment & Refining Costs		**US$ 000's**	**195,470**	**217,864**	**311,109**	**421,255**	**437,523**	**379,255**	**245,071**	**0**	**2,207,548**
Total Operating Costs		**US$ 000's**	**63,387**	**72,711**	**111,784**	**129,897**	**128,334**	**123,068**	**86,621**	**0**	**715,802**
Total Royalties & Levies		**US$ 000's**	**4,887**	**5,447**	**7,778**	**10,531**	**10,938**	**9,481**	**6,127**	**0**	**55,189**
Total Capital Costs		**US$ 000's**	**46,034**	**125,374**	**148,296**	**31,459**	**27,000**	**10,862**	**7,248**	**5,463**	**401,736**
FINANCIAL SUMMARY											
Total Revenue		US$ 000's	195,470	217,864	311,109	421,255	437,523	379,255	245,071	0	2,207,548
Total Outgoings		US$ 000's	114,308	203,532	267,858	171,887	166,272	143,411	99,996	5,463	1,172,727
Earnings before Interest, Tax, Depreciation and Amortisation (EBITDA)		**US$ 000's**	**81,162**	**14,332**	**43,252**	**249,368**	**271,251**	**235,844**	**145,075**	**-5,463**	**1,034,821**
Depreciation Allowance ignored		US$ 000's									0
Taxable Profit / Loss		US$ 000's	81,162	14,332	43,252	249,368	271,251	235,844	145,075	(5,463)	1,034,821
Tax Payable		US$ 000's	33,806	28,156	32,586	57,011	62,991	55,301	35,934	-	305,785
Net Profit After Tax		US$ 000's	47,356	(13,824)	10,665	192,357	208,260	180,543	109,141	(5,463)	729,036
After Tax Operating Cash Flow		US$ 000's	47,356	(13,824)	10,665	192,357	208,260	180,543	109,141	(5,463)	729,036
Cash Flow											
Cash Flow - After Tax		000's US$	47,356	(13,824)	10,665	192,357	208,260	180,543	109,141	(5,463)	729,036
Accumulated Cash Flow		000's US$	47,356	33,532	44,197	236,554	444,814	625,357	734,498	729,036	
Payback Period - Simple		years			-	-	-	-	-	-	0.00
Discounted Cash Flow - Mid Year Discounting											
Discounted Cash Flow @ 5%	K92 Calc	US$ 000's	46,214	(12,848)	9,440	162,161	167,207	138,051	79,480	(3,789)	585,917
Discounted Cash Flow @ 8%	K92 Calc	US$ 000's	45,568	(12,317)	8,798	146,935	147,299	118,236	66,181	(3,067)	517,634
Discounted Cash Flow @ 10%	K92 Calc	US$ 000's	45,152	(11,982)	8,404	137,795	135,625	106,886	58,740	(2,673)	477,947

Discount Rate		%	**5.0%**	**8.0%**	**10.0%**
Net Present Value (NPV) - After Tax		000's US$	**585,917**	**517,634**	**477,947**

22.4 Sensitivities

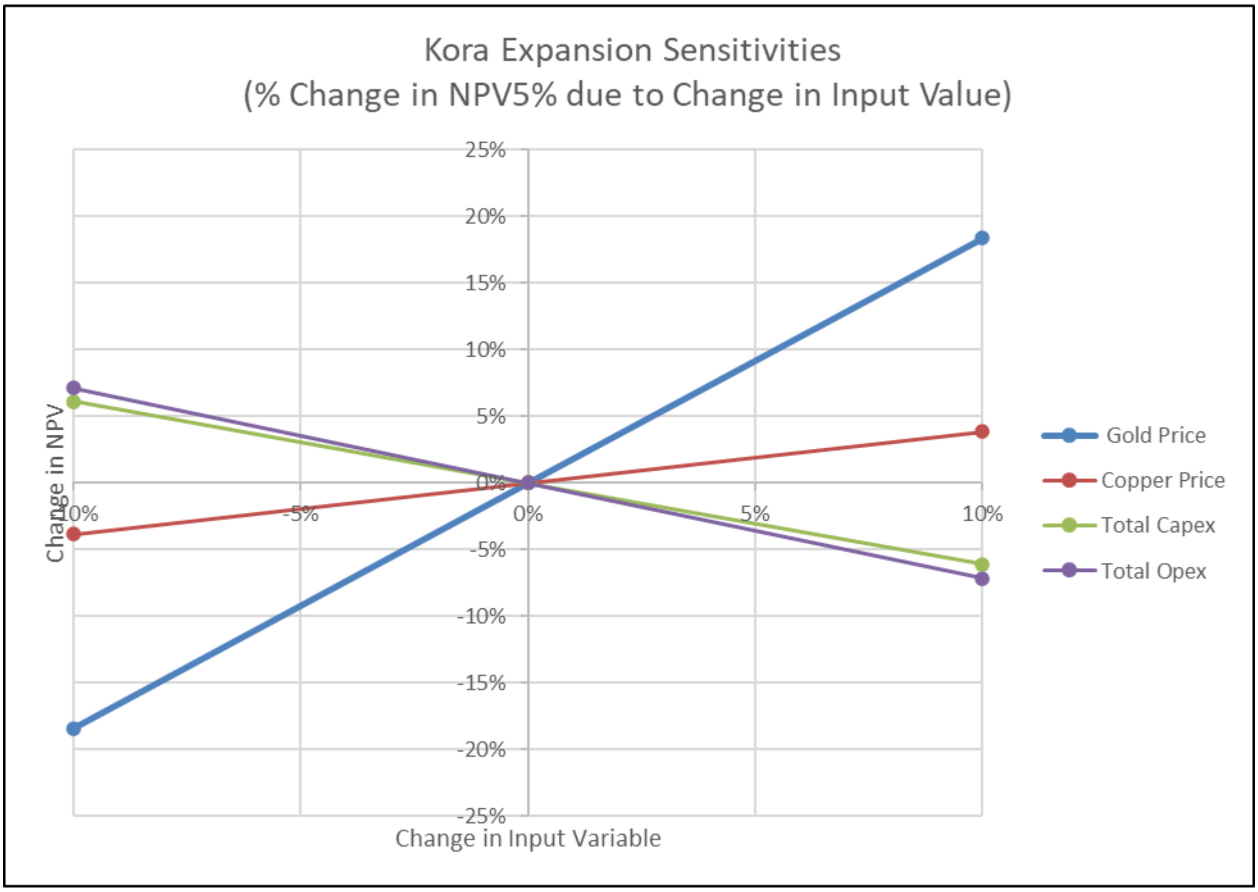
Figure 22.4.1 shows the % change in the project $NPV_{5\%}$ due to a +/- 10% change in input variables.

The project is most sensitive to gold price with a 10% change in input value equating to an 18% change in NPV. NPV is similarly affected by a change in either gold recovery or gold head grade.

The project is somewhat less sensitive to copper price with a 10% change in input value equating to a 4% change in NPV.

The project is slightly more sensitive to total operating cost than total capital cost.

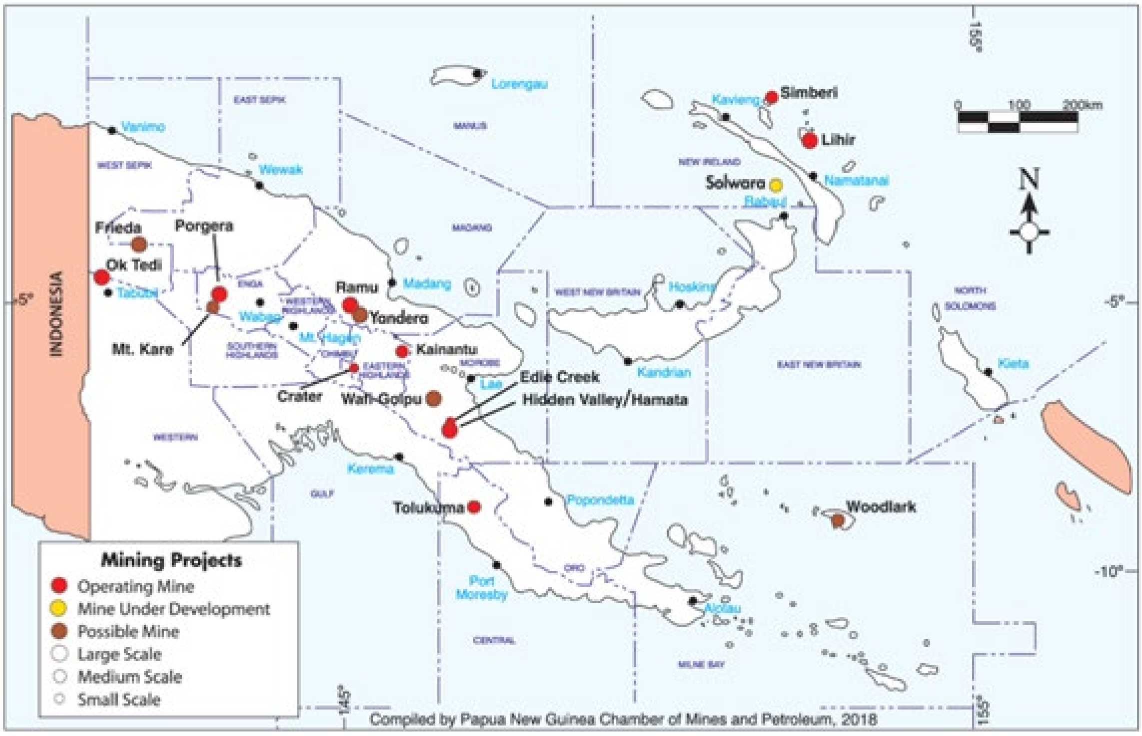
Figure 22.4.1 NPV Sensitivity



23.0 ADJACENT PROPERTIES

Kainantu occurs within a well-endowed belt of epithermal and porphyry style mineralisation that reportedly contains several major deposits Figure 23.1. This information was unable to be verified and the information is not necessarily indicative of the mineralization on the property that is the subject of this technical report.

Figure 23.1 Location of Kainantu Project and Gold Deposits Within Major Mineralised Province



Source: PNG Chamber Mines and Petroleum (2018)

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Kora 2022 PEA

The Kora 2022 PEA analyses a production case with an expansion of the Kora processing facilities, and associated infrastructure to 1.7 Mtpa. This is achieved by upgrading the mines infrastructure, increasing the mining production rate, and bringing the existing processing plant back into production and operating it in parallel to the 1.2 Mtpa plant described in the DFS.

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.

24.2 Kora 2022 PEA Assumptions

The PEA Case involves the construction of a standalone 1.2 Mtpa process plant adjacent to the 500,000 tpa Stage 2A process plant. At the end of 2024, the Stage 2A process plant is idled as the 1.2 Mtpa Stage 3 process plant ramps up, with commissioning of the Stage 3 process plant commencing in Q3 2024. Upon achieving the Stage 3 run-rate throughput in 2025, the Stage 2A plant is recommissioned in mid-2026, ramping up to run-rate throughput of 500,000 tpa by year end, for a combined processing run-rate of 1.7 Mtpa at the beginning of 2027.

To support the higher throughput rate, the underground mining fleet is significantly increased to support expanded mining operations opening multiple mining fronts concurrently: Kora Upper, Lower and Central Zones within the Kora deposit, and the Judd deposit.

24.3 Kora 2022 PEA Mining

24.3.1 Introduction

Entech were engaged by K92 Mining Inc. (K92) to complete a Preliminary Economic Study (PEA) on underground mining of the Kainantu Gold Mine. The PEA targets an expansion to a peak processing rate of 1,700,000 tpa with an 11-year mine life. The PEA assumes a start date of the life of mine schedule, of 1 January 2022.

24.3.2 Geotechnical

A total of 3,662 m of drill core was geotechnically logged from core photographs in detail for the Kora underground mining assessment during Q1 and Q3 2021. The drillholes were selected from existing exploration holes. From this total, 1,946 m, was logged as part of an initial campaign, with a second campaign of 1,716 m completed in October 2021. In addition to the 3,662 m, 472.3 m of core was validated and logged associated with the Judd orebody.

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 K92 staff and provided to Entech.

Rock property testing specific to the orebodies was unable to be sampled and sent for testing again due covid restrictions, resourcing availability in country, and available drill holes for sampling. 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 remain a significant information gap for the project and should be addressed as soon as practical.

The drill core logging data was analysed by Entech and forms the basis of this study. This information has allowed for characterization of the rock mass, assessment of stable stoping span predictions and estimates of dilution factors.

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 utilized for visualization 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 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.

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 and 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 and 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 both the Kora and Judd orebodies can be classed as 'good ground'. Table 24.3.1 outlines the recommended stope spans and dilution estimates for the Kora and Judd orebody.

Table 24.3.1 Summary of Assessed Stoping Parameters for Kora and Judd

Orebody	Parameters	Hangingwall	Footwall
K1	Allowable Strike Length	16 m	20 m
	Dilution	0 - 0.5 m	0 - 0.5 m
K2	Allowable Strike Length	31 m	19 m
	Dilution	0 - 0.5 m	0 - 0.5 m
Judd	Allowable Strike Length	35 m	35 m
	Dilution	0 - 0.5 m	0 - 0.5 m

The presence of the FGZ and pervasive structure throughout the rock units will dominate stope wall behaviour with the possibility of slabbing, sliding and unravelling failure types occurring 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 will be the key to minimising stoping performance issues.

24.3.3 Current Site Mining Methods

Currently Avoca and modified Avoca stoping methods (as described below) are used on site.

24.3.4 Mining Method Selection Process

The previous 2020 PEA, in combination with current site mining practices, are 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 mechanized 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 two 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 optimization parameters for each of the potential mining methods.
- Completed stope optimizations over a range of cut-off grades (COG).
- Estimated mining physicals and mining costs.
- Produced excel based schedules with stope optimization 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.; and
- Selected a cut-off grade and mining method(s) as a basis for the PEA.

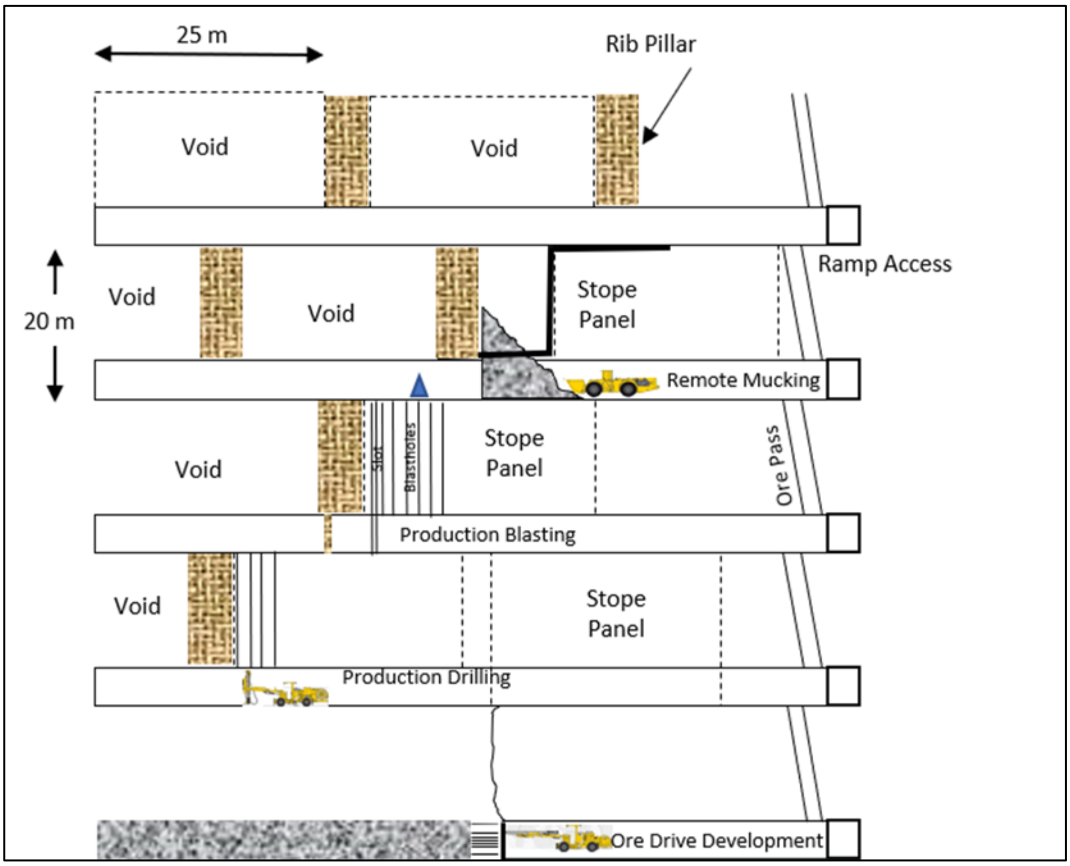
Considered Mining Methods

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. It is estimated that two years is the time required to build a paste fill plant, and so the mining methods have been split around this timeframe.

Longhole Open Stoping with Pillars

LHOS with pillars is a mechanized 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 24.3.1.

Figure 24.3.1 Example Long Section of Longitudinal LHOS Extraction



Avoca

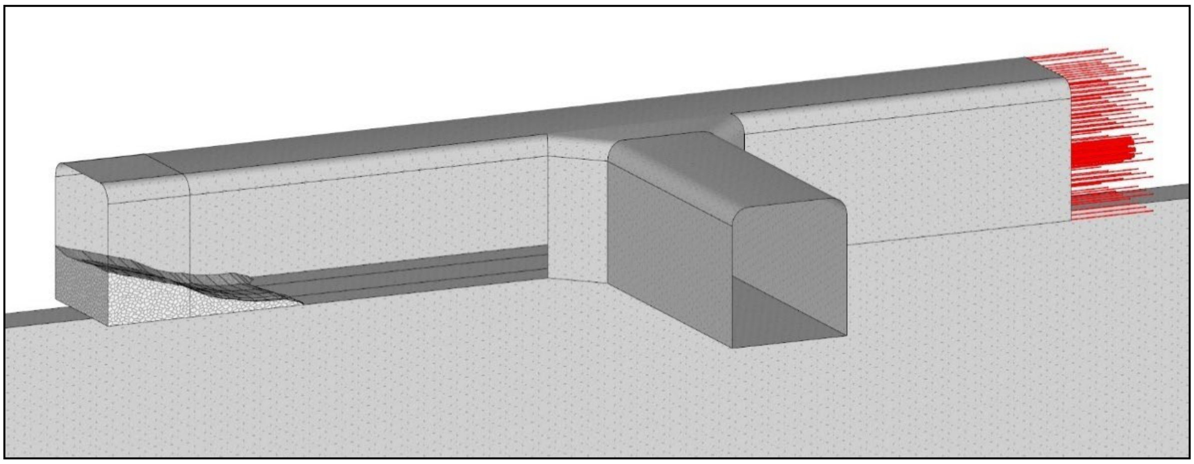
Avoca is a mechanized mining method which is widely used in global underground mining. The proposed Avoca stoping design assumptions includes a 20 m vertical level spacing, a bottom-up extraction sequence, and 76 mm or 89 mm diameter down-hole production drilling. The stoping panels are subdivided into 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 is bogged, and the process is repeated. In Avoca, there is access to the stoping horizon at both ends on 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 24.3.2.

Figure 24.3.2 Step 1 – Ore Development



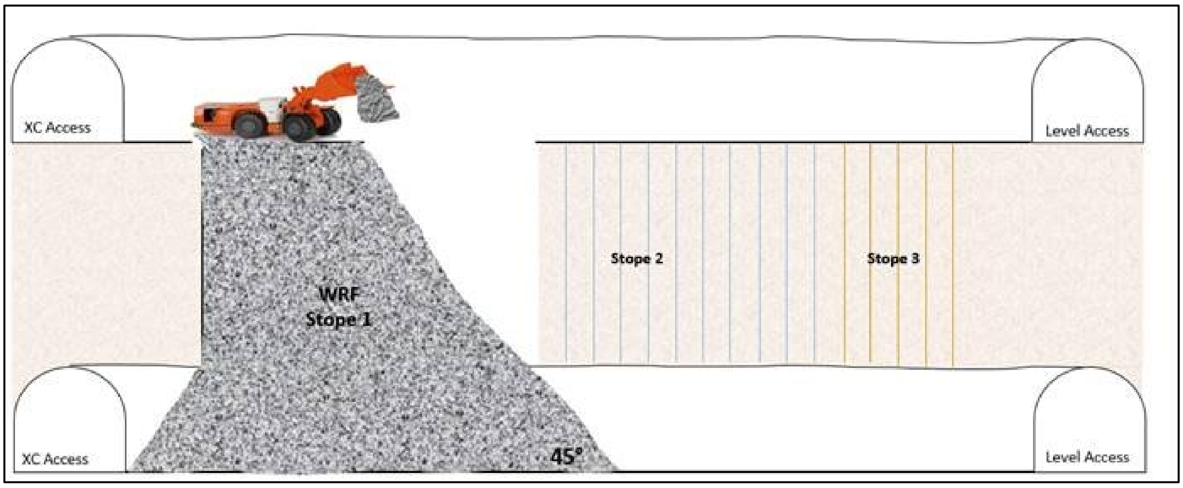
2. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block/panel. A strike length of approximately 6.0 m for each firing is typical.

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 24.3.3.

Figure 24.3.3 Avoca Overall Mining Method Schematic



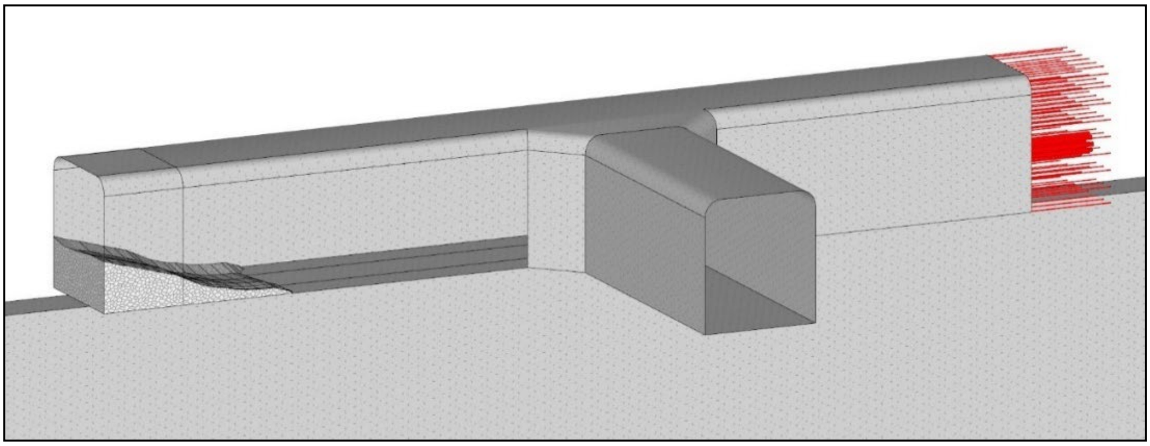
Modified Avoca

Modified Avoca is a mechanized mining method which is well understood in underground mining environments. The proposed modified Avoca stoping design assumptions includes 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:

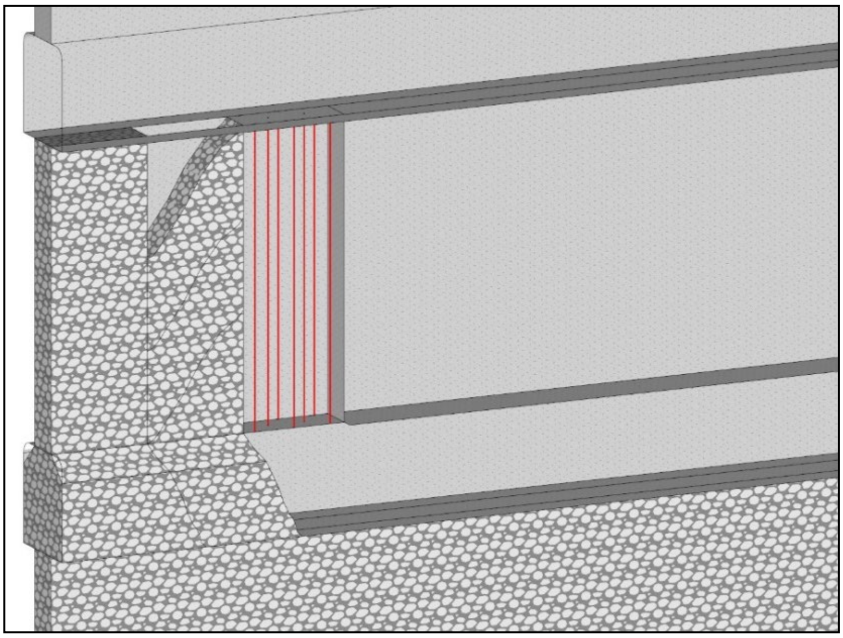
1. Ore drives are developed along strike at the top and bottom of the stoping block or panel using jumbo drills as per Figure 24.3.4.

Figure 24.3.4 Step 1 – Ore Development



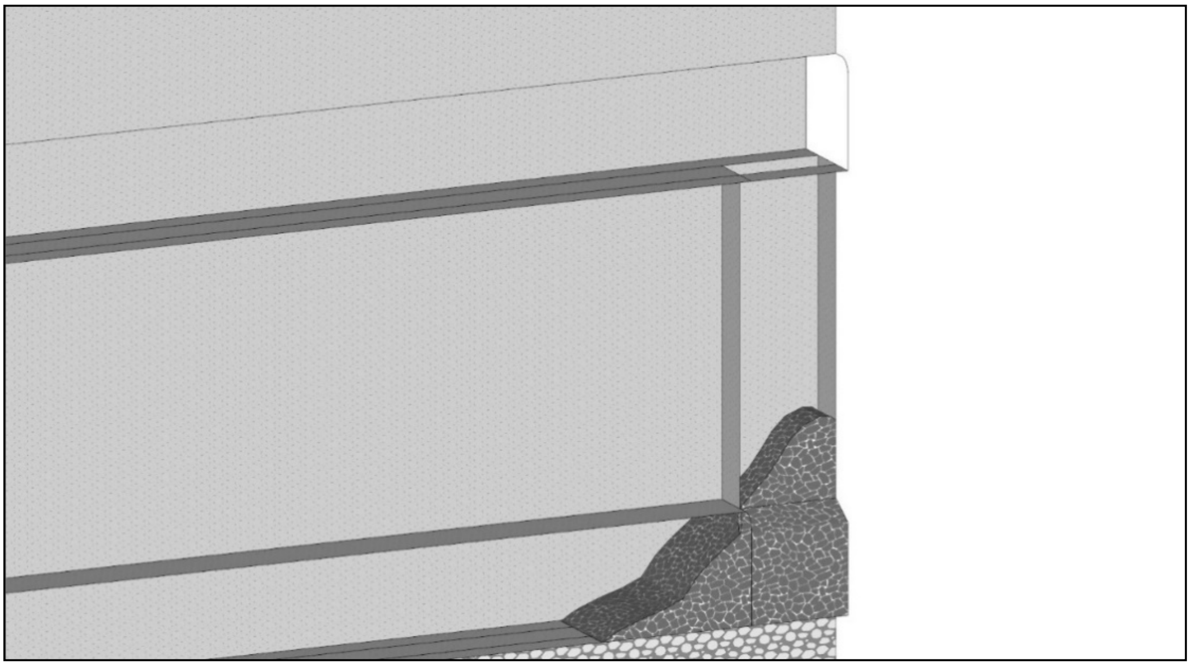
2. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block. A strike length of 6.0 m for each firing is shown in this example as per Figure 24.3.5.

Figure 24.3.5 Step 2 – Stope Drill & Blast



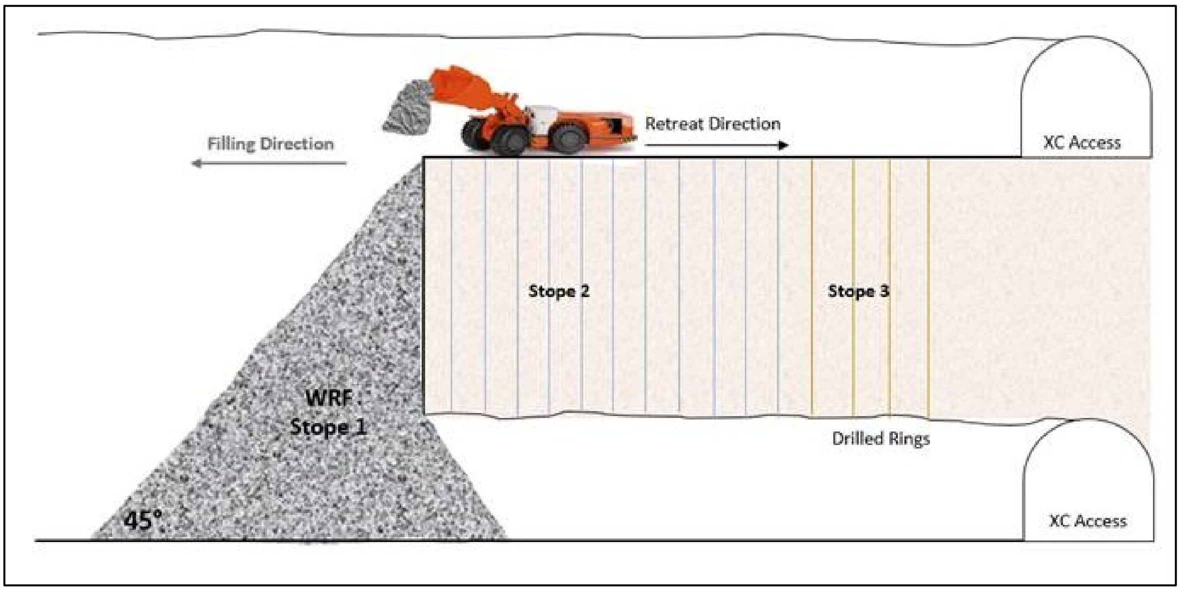
3. Blasted ore is bogged to level stockpiles for truck loading using conventional diesel-powered underground loaders, utilising both conventional (i.e., manual) and tele-remote techniques as in Figure 24.3.6.

Figure 24.3.6 Step 3 – Stope Loading



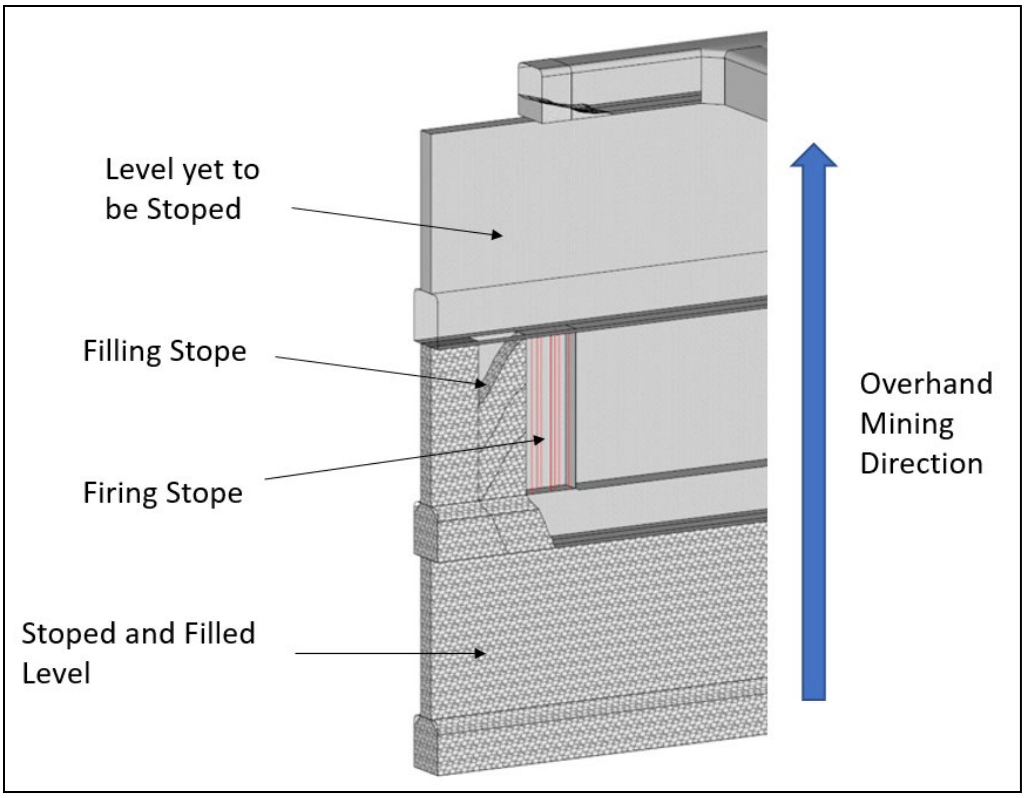
4. Waste material is dumped into the stope void from the level above as per Figure 24.3.7.

Figure 24.3.7 Modified Avoca Waste Tipping and Cycle



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 24.3.8.

Figure 24.3.8 Modified Avoca Overall Mining Method Schematic

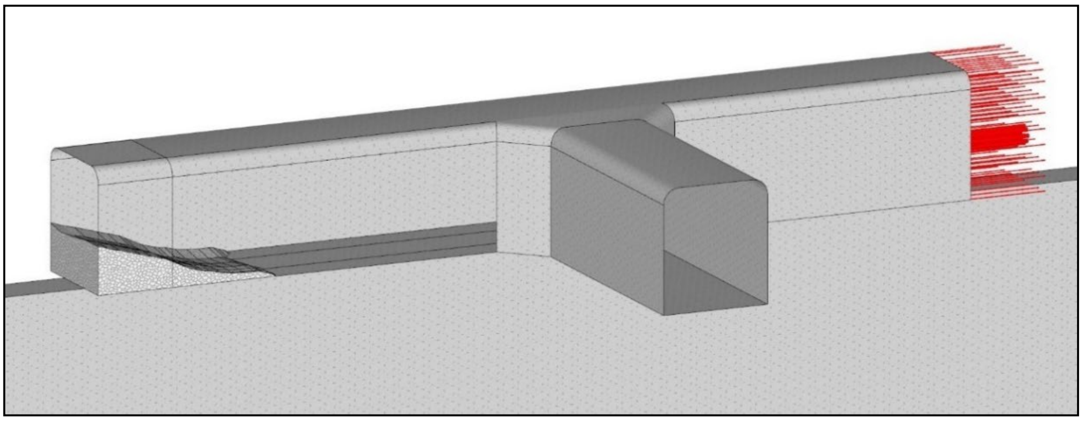


Longhole Stoping with Cemented Rock Fill

The LHS with CRF assumes a continuous fill mining method involving the following steps:

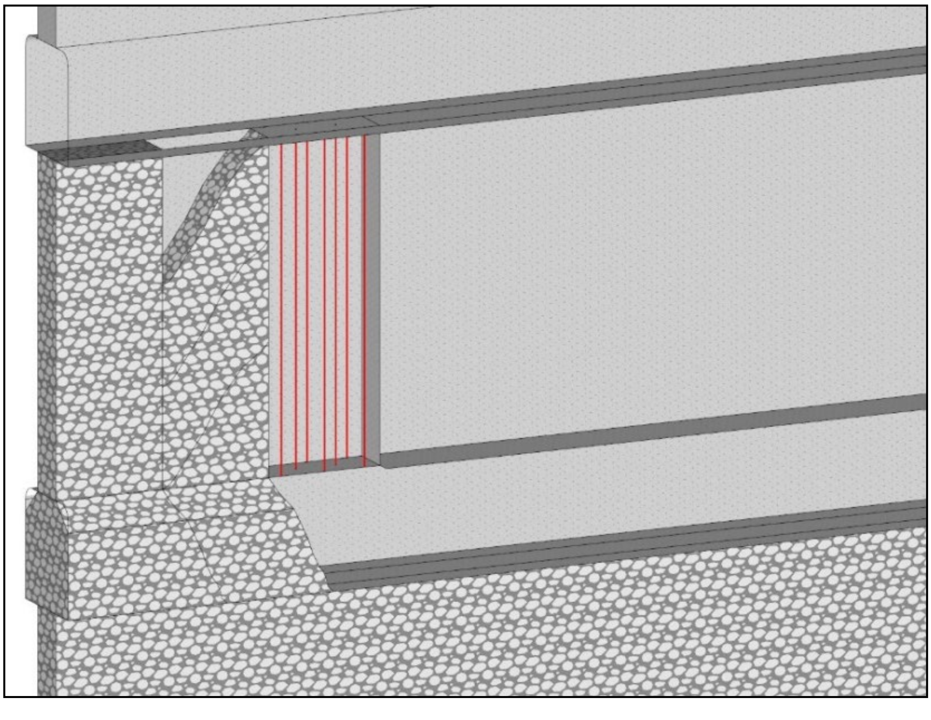
1. Ore drives are developed along strike at the top and bottom of the stoping block or panel using jumbo drills as per Figure 24.3.9.

Figure 24.3.9 Step 1 – Ore Development



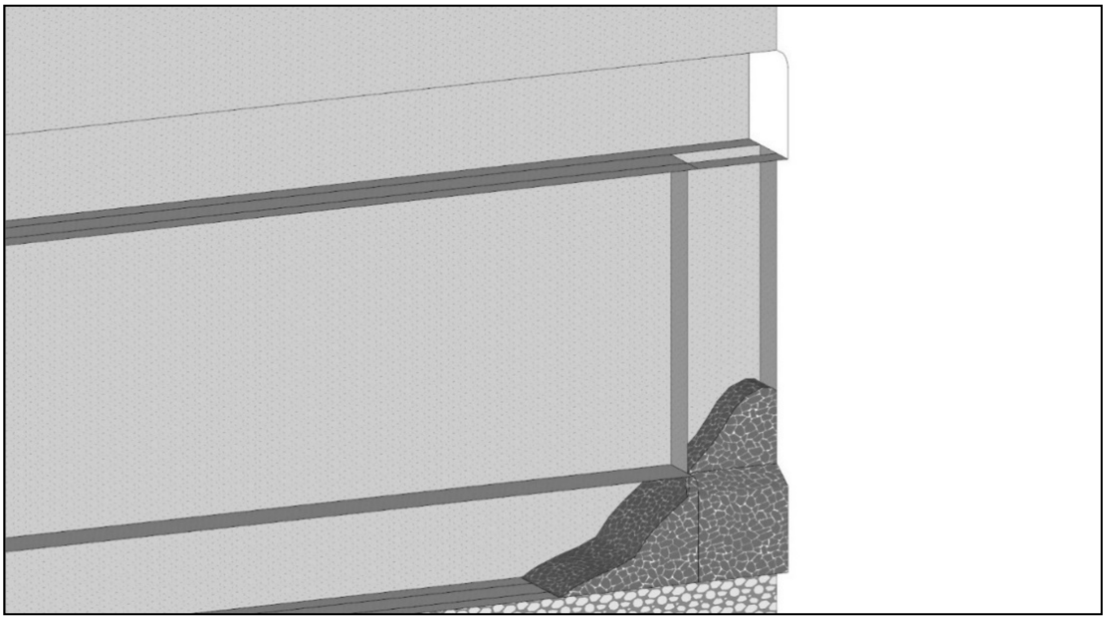
2. Stope ore is drilled and blasted (one firing) using longhole drill techniques. Slot rises are developed when firing the first stope in a block. A strike length of 6.0 m for each firing has been assumed in this example as per Figure 24.3.10.

Figure 24.3.10 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 24.3.11.

Figure 24.3.11 Step 3 – Stope Loading



4. Waste material mixed with a cement slurry is dumped into the stope void from the level above.

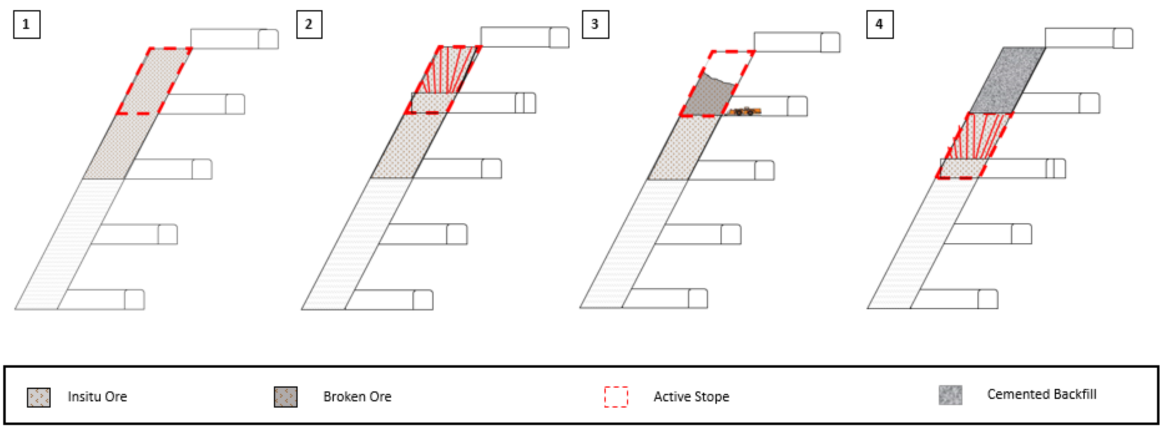
5. Once filled, the next stope along is fired against the fill typically within 12-36 hours which allows for firing against the fill without requiring another slot to be established. Alternatively larger stopes can be excavated, filled with the cement slurry, and a new slot opening will be required to mine the next adjacent panel in the sequence.

Longhole Stoping with Paste fill

LHS with paste fill is proposed to be utilized 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 may also be opportunity to utilize transverse extraction where required, most likely 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 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) utilized in a top-down/underhand mining sequence is shown in Figure 24.3.12.

Figure 24.3.12 Example of Longitudinal Stoping Extraction with Paste Fill Mining Cycle



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 Q2 2024, and longhole stoping with pastefill for the remainder of the mine life. An economic comparison was done 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.

24.3.5 Stope Design

On completion of the trade-off studies, Entech were provided with the PEA geological model. Stope optimizations were completed to compare to the trade-off studies and validate the selected Avoca, and LHS with paste fill mining methods. The stope optimization results aligned with the findings of the trade-off studies and K92 confirmed selection of the Avoca / Modified Avoca and LHS with paste fill mining methods for design and scheduling of the PEA. The PEA mine plan only includes Measured, Indicated, and Inferred Mineral Resources.

Stope Optimization

The stope optimization parameters for the PEA are shown in Table 24.3.2 and Table 24.3.3 below, based on the proposed mining method. The results of the previous trade-off studies meant that a smaller range of cut-off grades was run for the PEA.

Table 24.3.2 LHOS with Pillars / LHS with Paste Fill

Stoping Parameter	Value
Stoping COG (g/t Au Eq.)	3,4,4.5,5,5.5
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 24.3.3 Avoca

Stoping Parameter	Value
Stoping COG (g/t Au Eq.)	3,4,4.5,5,5.5
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 (the initial two years of the mine plan as advised by K92), are planned to be mined with the Avoca/modified Avoca methods.

Cut-Off Grade Selection

K92 selected the 4.5 g/t AuEq stope optimizations to carry forward for the final mine design and PEA 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.

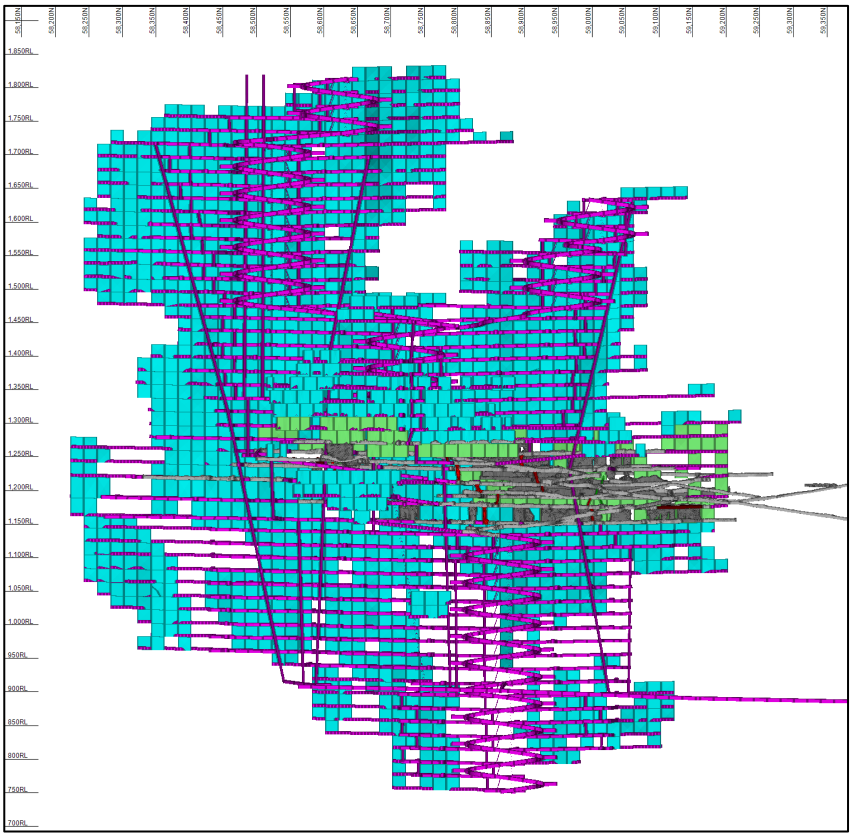
Stoping Methodology

The extraction methods that are proposed for the PEA are:

- Longitudinal LHS with paste fill
- Longitudinal Avoca and Modified Avoca mining

The spatial application of these mining methods is shown in Figure 24.3.13.

Figure 24.3.13 Stope Extraction Methods – Longitudinal LHS with Fill (blue), Avoca (green), As-builts in Grey



Stope Design Parameters

The mining method selection outcomes dictated the stope design parameters as per the MSO parameters previously described. Following the stope optimization 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.

Modifying Factors

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 optimization phase. An additional 0.5m of dilution was applied to stope shapes that were within 2.0 m from the fault gouge zone. These dilution parameters were based on Entech's geotechnical study.

An additional 2.5% dilution from paste fill, and 5% waste rock dilution for Avoca stoping methods were applied in the schedule. Paste fill and Avoca dilution are based on benchmark and site dilution data. The LOM average stope dilution is ~20%.

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.

Drilling, Blasting and Slotting Methodology

The current drill design philosophy for Kainantu is based around 89 mm blast holes, although a 76mm blasthole could also be suitable for some stoping areas. Underground blasting with 2.0 - 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 24.3.4 outlines typical burden and spacing parameters that would be used for this hole size.

Table 24.3.4 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

The drill yield used for the PEA is 5.5 t/dm which aligns with current site parameters and is appropriate for the average stope width.

Table 24.3.5 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 24.3.5 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 'Rhino' (or equivalent) raises for blasting voids as illustrated in Figure 24.3.14. 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 24.3.15 shows the 'Rhino' (or equivalent) rig.

Figure 24.3.14 Example Slot Around 0.76 m Rhino Boxhole

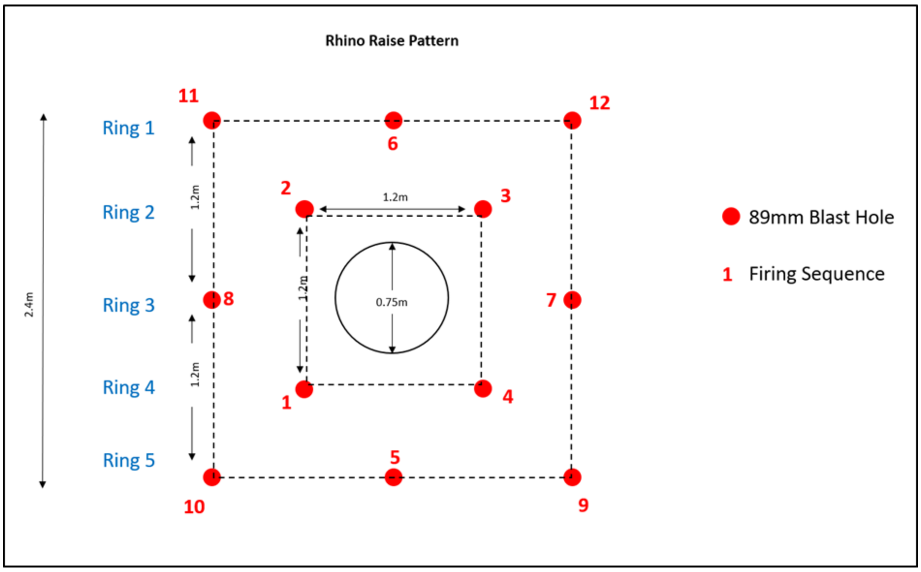
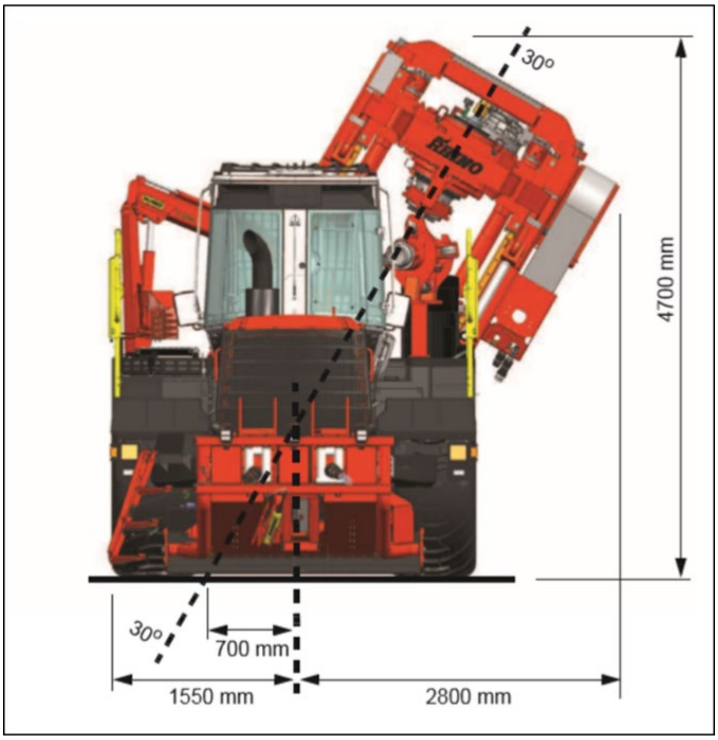


Figure 24.3.15 Rhino 100M Boxhole/Raisebore rig

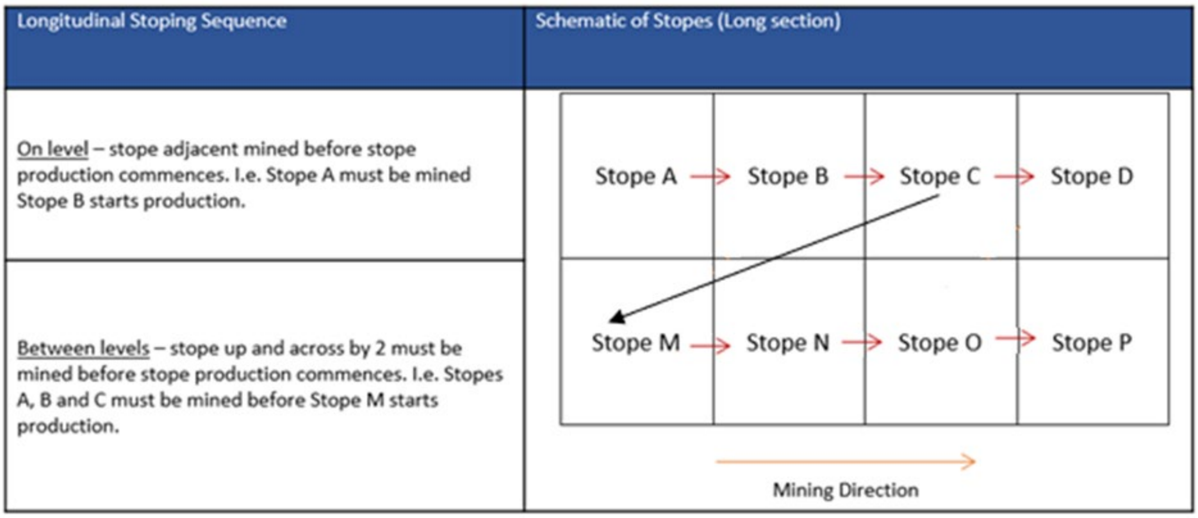


Longitudinal Sublevel Stoping with Paste Fill

Stoping Sequence

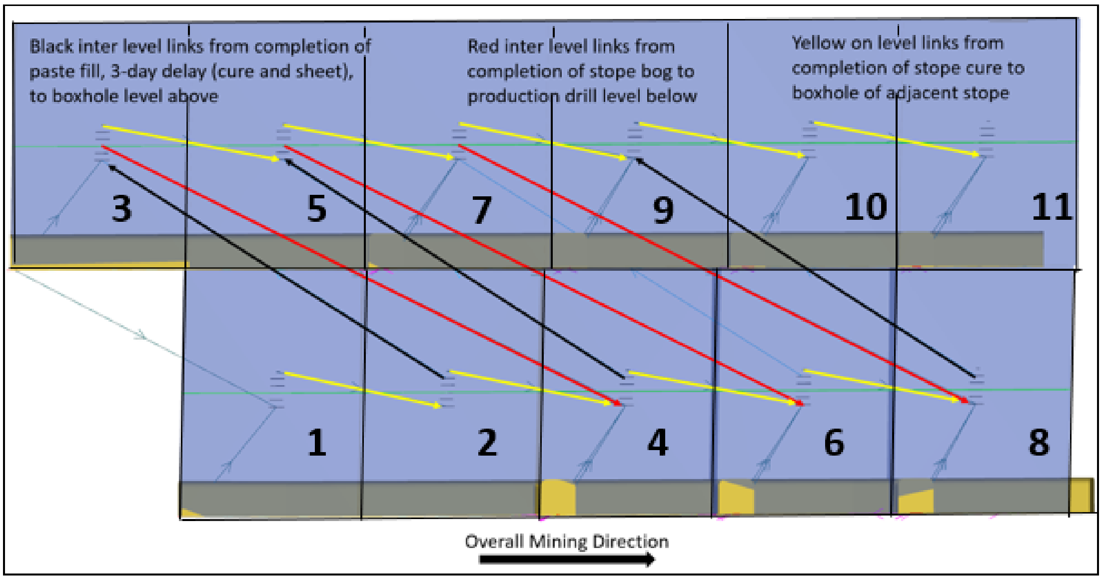
The typical stoping and linking sequence for underhand longitudinal LHS with paste fill is shown in Figure 24.3.16, with overhand sequence in Figure 24.3.17.

Figure 24.3.16 Example of a Longitudinal LHS with Paste Fill Underhand Mining Sequence (Sequence Will Vary in Areas)



Longitudinal Stoping Sequence	Schematic of Stopes (Long section)
On level – stope adjacent mined before stope production commences. I.e. Stope A must be mined Stope B starts production.	Stope A → Stope B → Stope C → Stope D
Between levels – stope up and across by 2 must be mined before stope production commences. I.e. Stopes A, B and C must be mined before Stope M starts production.	Stope M → Stope N → Stope O → Stope P Mining Direction

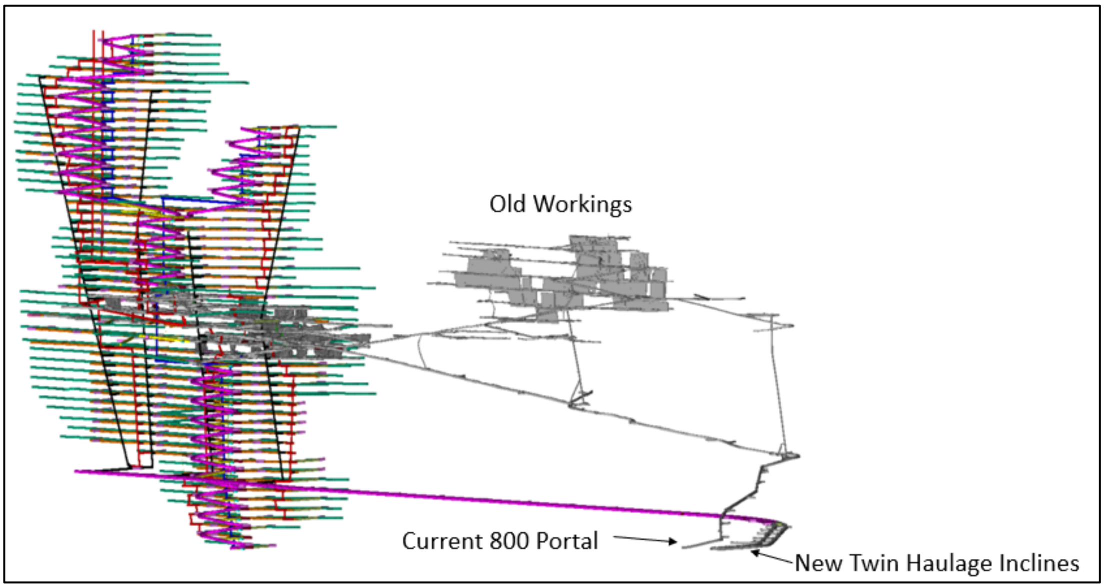
Figure 24.3.17 Example of a Longitudinal LHS with Paste Fill Overhand Mining Sequence (Sequence will Vary in Areas, Example Sequence Shown By #)



24.3.6 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 24.3.18.

Figure 24.3.18 An Isometric View Looking Southwest of the Kainantu Underground Development Layout



Entech has completed development design from the current as-built positions, and followed the current K92 site design standards. Decline gradients (-1:7), decline standoffs to the orebody (~70 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 type is suitable for the proposed mining methods.

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 'Rhino' (or equivalent) 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.

Mine Access

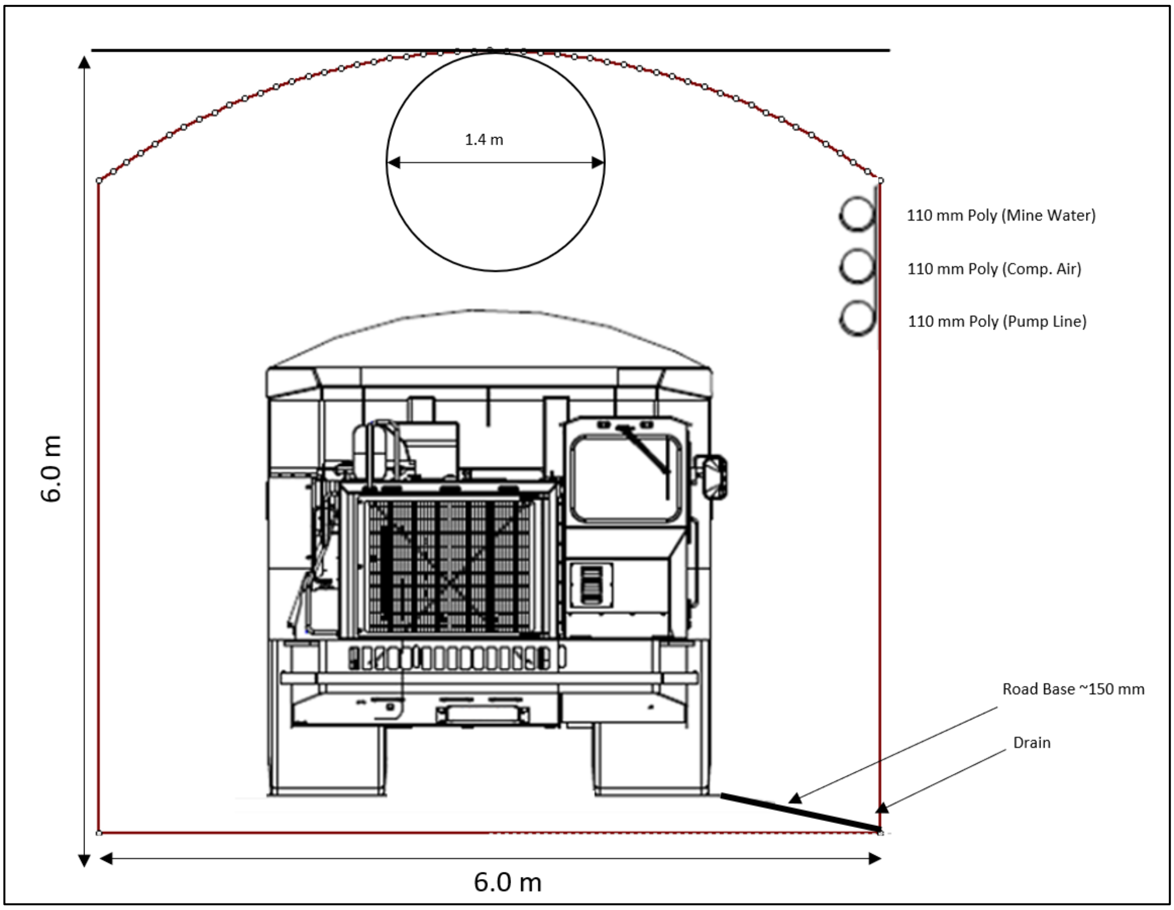
The mine is currently accessed via the 800 Portal which is a 4.5 mW x 4.5 mH access accommodating 45 t payload underground mining trucks. New twin haulage incline drives are being excavated, one at 6.0 mW x 6.0 mH drive size, and the other at 5.0 mW x 5.0 mH. These new drive sizes could allow future 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.

Decline Design

The primary haulage drive will be the 6.0 mW x 6.0 mH which accesses the ore pass network, to allow for future trucking size increase, as per Figure 24.3.19. The other areas of the mine utilize a 5.0 mW x 5.5 mH decline drive size, with the current 800 portal and legacy incline at 4.5 mW x 4.5mH 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 20-25 m to the Judd lode. This distance will minimize 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.

Figure 24.3.19 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.

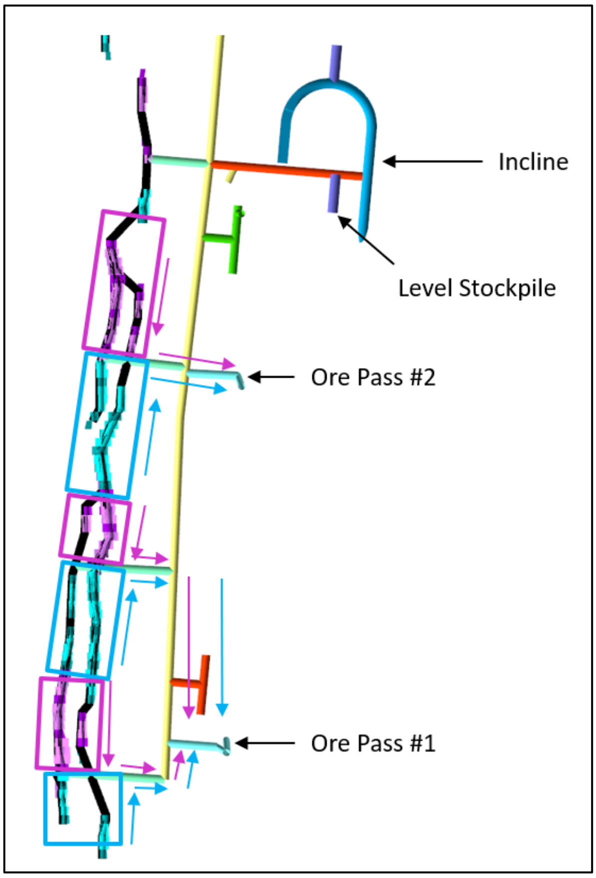
A 'racecourse' style decline configuration with a minimum radius of 25 m has been designed to provide optimal access to levels every 20 vertical metres whilst also allowing trucks to operate at a productive speed. Stockpiles are typically designed every 140 m along the decline, which is sufficient for high-speed development. This configuration allows for numerous locations to pass equipment and position infrastructure such as stockpiles, electrical substations and refuge chambers.

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.

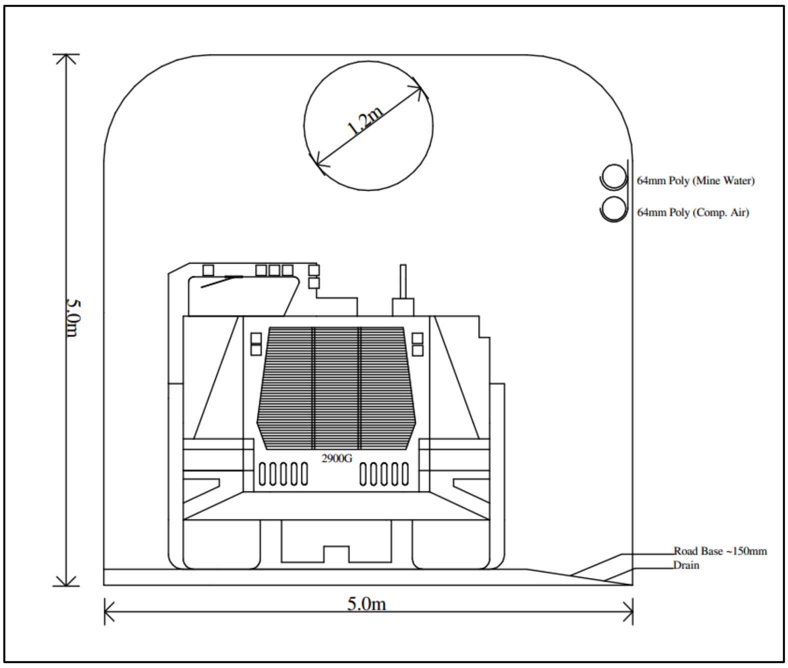
A square profile stockpile has been designed on each level access at 5.0 mW x 5.5 mH and 25 m in length. The level stockpile will be used for the stockpiling of development and stope ore that is produced from the level until it can be loaded into a truck or ore pass. A typical level layout is shown in Figure 24.3.20.

Figure 24.3.20 An Example of the 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 24.3.21.

Figure 24.3.21 An Example of the Ore Drive Profile



Vertical Development

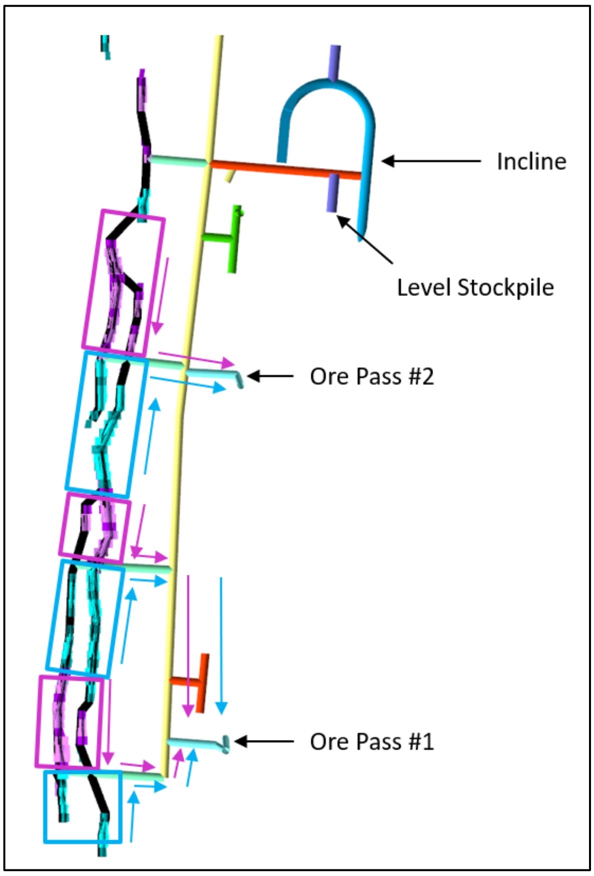
Vertical development is separated into capital and operating development. Capital development consists of ore passes, escapeway rises, ventilation return rises, and fresh air intakes. Escapeway rises are planned at 1.1 m diameter to be excavated with the 'Rhino' (or equivalent) boxhole rig. Ventilation 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 0.76 m diameter boxhole rise excavated with the 'Rhino' (or equivalent).

Materials Handling

Two material handling methods were considered. The first option considered all material to be hauled via trucking from the extraction level that the material was blasted. This material would be loaded onto trucks by a bogger utilizing the nearest stockpile. The second option considered the use of ore pass infrastructure at different locations within the mine - blasted material would be tipped by a bogger into an ore pass and hauled from a dedicated trucking level below.

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 24.3.22.

Figure 24.3.22 Plan View Typical Level Layout



The following points resulted in the selection of utilising an ore pass network versus trucking only.

- 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.
- Better cost profile makinsg the ore pass option more cost effective.
- Trucking congestion could become problematic if a trucking only option was used at a higher production rate.

24.3.7 Mining Inventory

The PEA inventory can be found detailed in the table below, with an annual material movement breakdown in Table 24.3.6 below.

Table 24.3.6 Material Movement Breakdown

		Total	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Waste	t	5,724,586	570,390	791,008	813,006	847,904	778,323	651,210	646,818	434,260	191,667	-	-
Dev Ore (t)	t	2,381,745	113,959	188,891	263,801	194,312	263,652	386,515	409,427	512,674	48,512	-	-
Stope Ore (t)	t	9,773,783	336,608	335,796	571,365	1,031,001	1,254,383	1,315,357	1,297,840	1,156,102	1,406,731	844,924	223,678
Ore	t	12,155,527	450,568	524,687	835,166	1,225,312	1,518,036	1,701,871	1,707,267	1,668,776	1,455,243	844,924	223,678
Ore Tonnes													
Measured	t	2,043,650	269,844	235,914	317,956	552,926	443,459	152,403	30,994	21,159	11,153	2,221	5,620
Indicated	t	3,043,274	167,796	216,916	346,558	479,118	611,624	476,802	345,951	207,933	139,343	47,777	3,456
Inferred	t	7,068,601	19,984	90,124	158,816	198,690	512,340	1,096,199	1,356,055	1,429,953	1,283,636	772,430	150,373
Total	**t**	**12,155,526**	**457,624**	**542,954**	**823,330**	**1,230,734**	**1,567,423**	**1,725,405**	**1,732,999**	**1,659,046**	**1,434,132**	**822,428**	**159,450**
Au Grade	g/t	6.98	9.53	7.90	6.64	6.47	7.61	7.66	6.04	6.51	5.51	7.13	14.02
Cu Grade	%	1.20%	0.76%	0.82%	1.18%	1.29%	1.06%	1.03%	1.16%	1.44%	1.52%	1.20%	0.94%
Ag Grade	g/t	22.85	14.61	14.97	22.69	23.21	18.80	18.77	23.17	29.60	30.45	22.26	15.13
Metal													
Au	Moz	2.73	0.14	0.14	0.18	0.26	0.38	0.43	0.34	0.35	0.25	0.19	0.07
Cu	kt	145.36	3.49	4.47	9.74	15.86	16.64	17.80	20.17	23.82	21.87	9.86	1.51
Ag	Moz	8.93	0.21	0.26	0.60	0.92	0.95	1.04	1.29	1.58	1.40	0.59	0.08

24.3.8 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 peak rate of 1,700,000 t per annum (1.7 Mtpa). The mine life is 11 years with a 6-year production ramp up (inclusive of 2022), sustained production of 1.7 Mtpa for 3 years, and final year production of 0.2 Mtpa.

The major constraints on the underground scheduling were as follows:

- Ensure a smooth ramp-up to steady ore production.
- Minimize variations in development rates and production to avoid additional project costs due to under-utilization 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 (m RL).
- 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 summarized into individual activities that provided the appropriate detail for PEA 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.

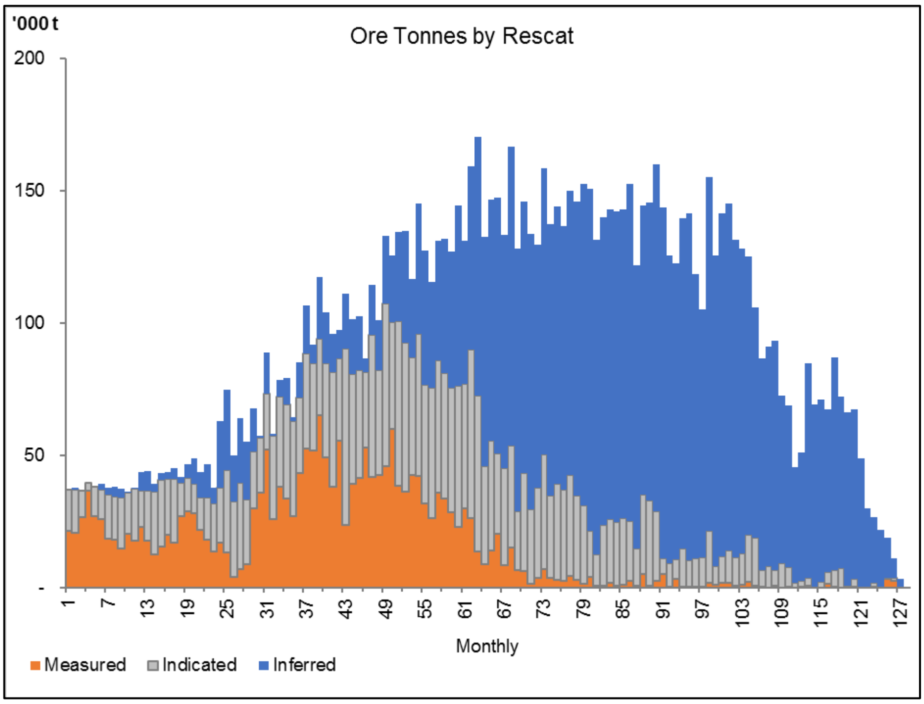
A summary of the key mining physicals and schedule is detailed in Table 24.3.7 below.

Table 24.3.7 Key Physicals Per Annum

		Total/Av	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Lateral Development Total	**m**	**110,945**	**9,020**	**13,192**	**14,292**	**14,433**	**14,435**	**14,436**	**14,430**	**13,420**	**3,285**	**-**	**-**
Vertical Development Total	**m**	**36,695**	**1,328**	**2,050**	**4,233**	**2,914**	**3,615**	**4,678**	**5,355**	**4,175**	**4,835**	**2,777**	**735**
Ore Profile													
Ore Tonnes	t	12,155,527	450,568	524,687	835,166	1,225,312	1,518,036	1,701,871	1,707,267	1,668,776	1,455,243	844,924	223,678
Au Grade	g/t	6.98	9.53	7.90	6.64	6.47	7.61	7.66	6.04	6.51	5.51	7.13	14.02
Au Ounces	oz	2,727,923	138,046	133,213	178,254	254,979	371,205	419,396	331,347	349,033	257,869	193,750	100,830
Cu Grade	%	1.20	0.8	0.8	1.2	1.3	1.1	1.0	1.2	1.4	1.5	1.2	0.9
Cu Tonnes	t	145,361	3,438	4,320	9,882	15,786	16,117	17,558	19,869	23,957	22,192	10,129	2,112
Ag Grade	g/t	22.85	14.61	14.97	22.69	23.21	18.80	18.77	23.17	29.60	30.45	22.26	15.13
Ag Ounces	oz	8,930,020	211,598	252,485	609,194	914,184	917,597	1,027,064	1,272,019	1,587,964	1,424,534	604,562	108,820

A monthly breakdown of mined ore tonnes by resource classification (Rescat) can be found in Figure 24.3.23.

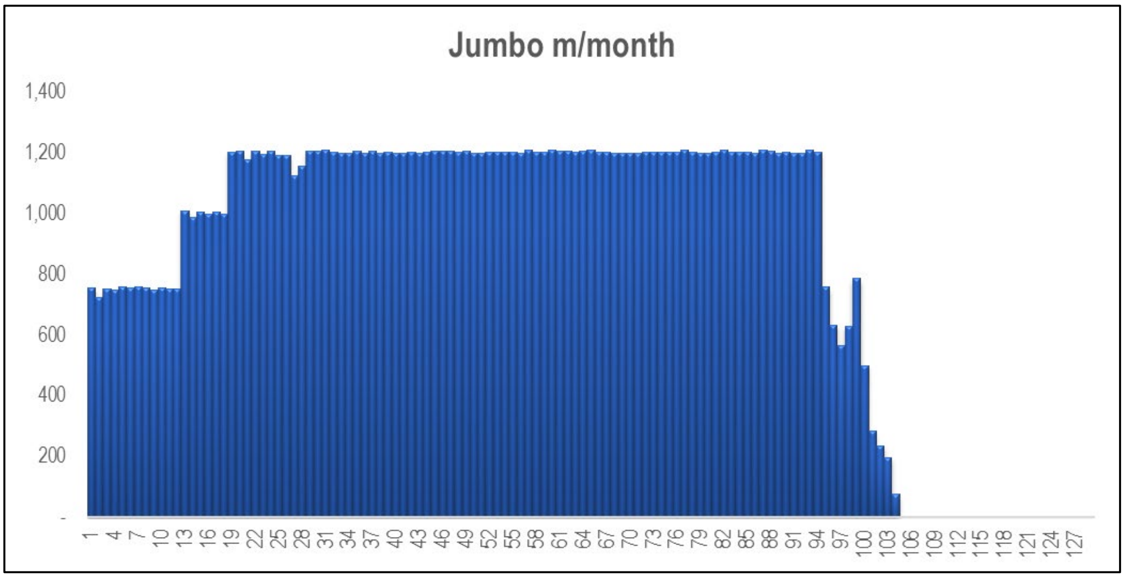
Figure 24.3.23 Monthly Ore Breakdown by Resource Classification



Jumbo Development

The life of mine horizontal development schedule is shown in Figure 24.3.24.

Figure 24.3.24 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.

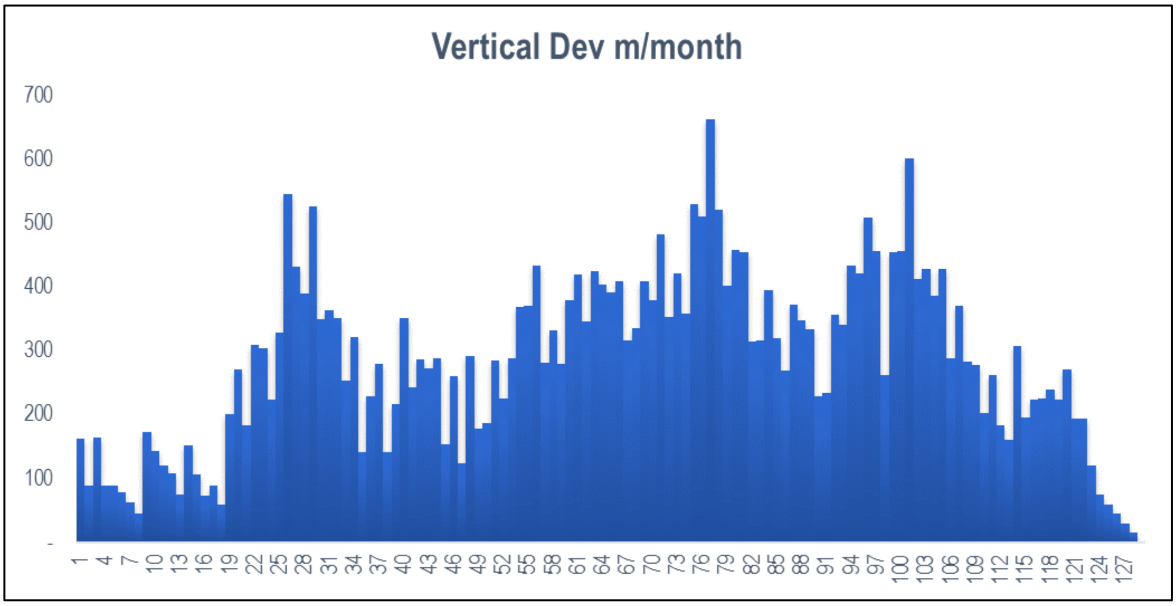
Task priorities followed were:

- Establish primary ventilation drive / escapeway.
- Establish primary ventilation.
- Push decline to establish multiple production horizons.
- Develop ore drives as required for stoping.

Vertical Development

The life of mine vertical development schedule is shown in Figure 24.3.25.

Figure 24.3.25 Vertical Development Metres by Month



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.

Loading

The life of mine loading by month is shown in Figure 24.3.26, and the ore production profile in Figure 24.3.27.

Figure 24.3.26 Bogged Tonnes by Month

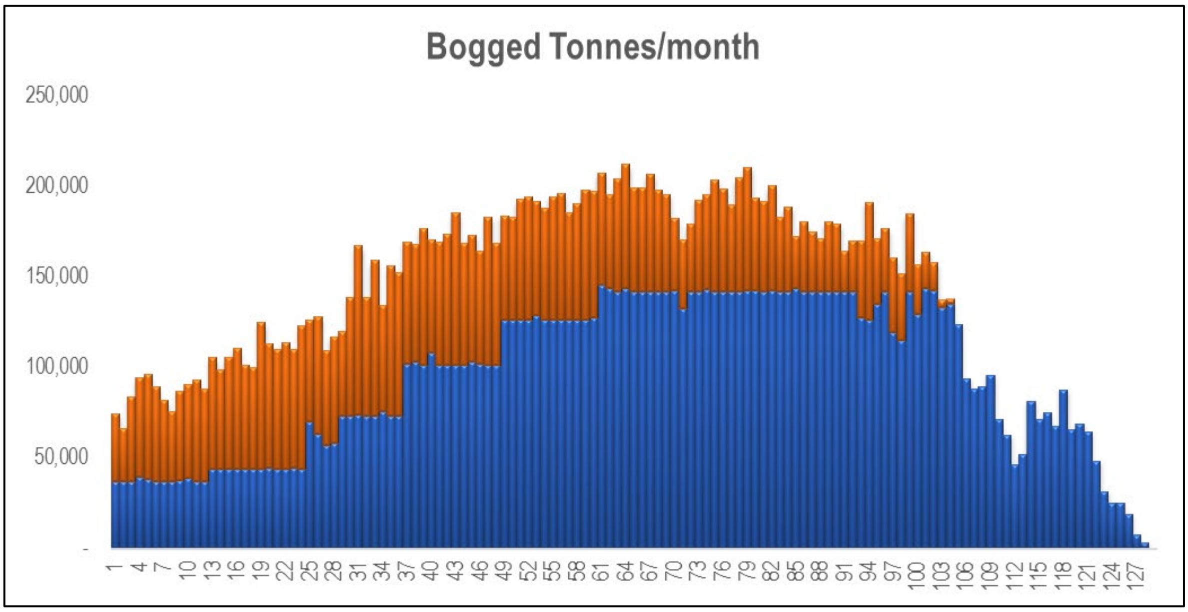
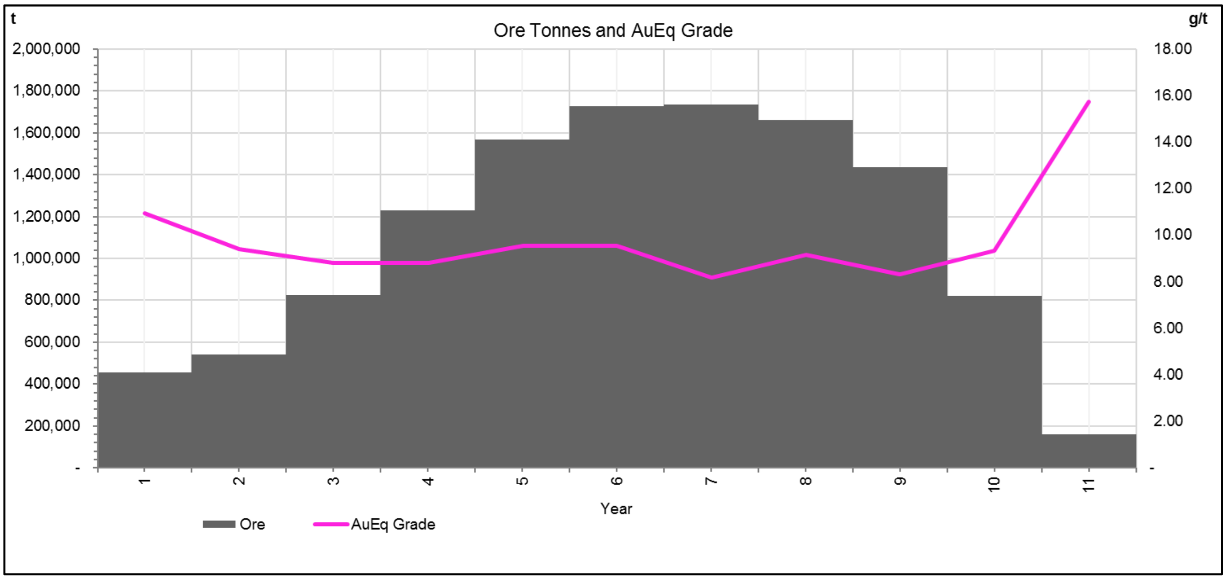


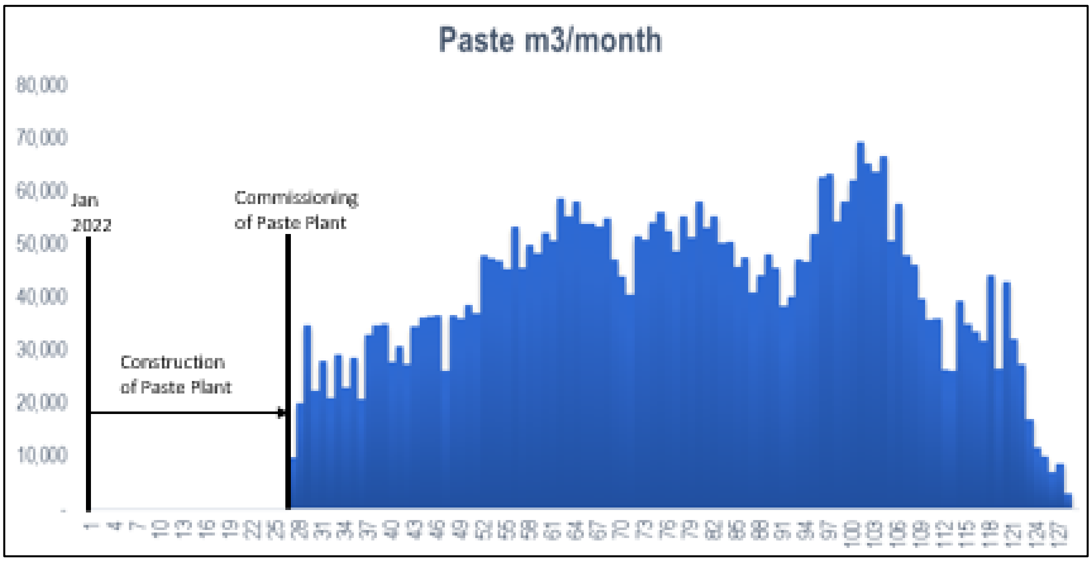
Figure 24.3.27 Ore Tonnes and AuEq Grade Per Annum



Backfilling

Figure 24.3.28 below shows the life of mine paste filling schedule.

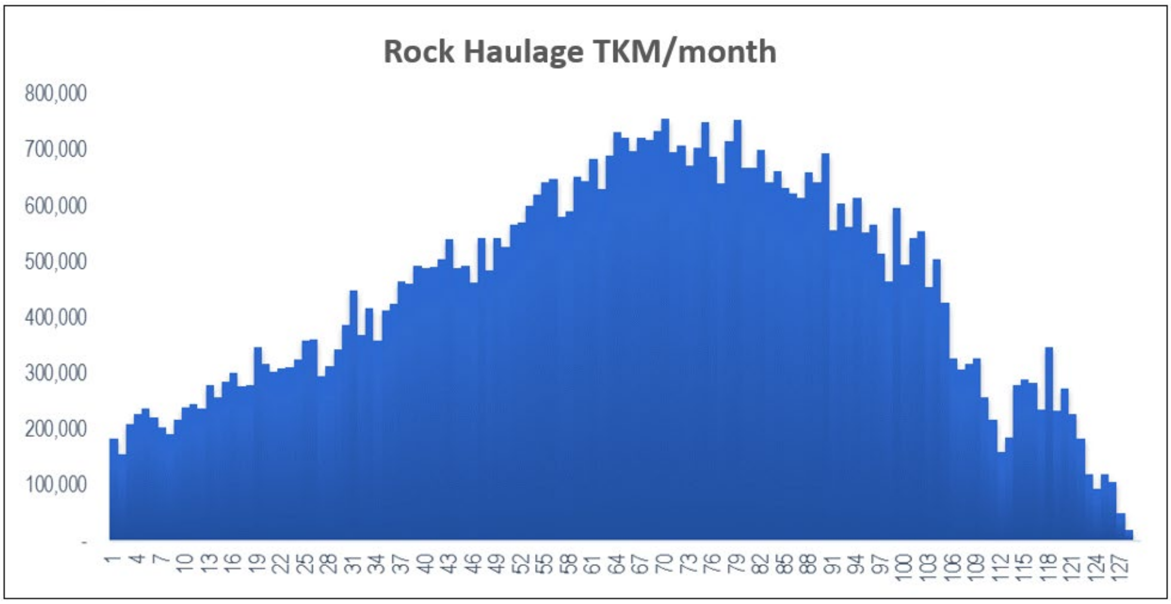
Figure 24.3.28 Paste Fill by Month



Haulage

The haulage requirement from the mine, inclusive of all ore and waste material, is illustrated in Figure 24.3.29 and is stated on a tkm basis. Surface haulage is captured by a separate fleet and has been costed independently of the underground haulage cost and physicals estimates.

Figure 24.3.29 Rock Hauled TKM by Month



Mine Productivity

The mine operates on 12-hour shifts - Papua New Guinean nationals typically work on a 10-day shift, 10-night shift, 10 off roster and expatriate workers typically work a fly in fly out arrangement on a 4-weeks-on and 4-weeks-off roster. Productivity of scheduling tasks have been derived by Entech, with fleet requirements built up by K92 based on site productivities. Entech have reviewed the specified fleet requirements to ensure they correlate with appropriate productivity assumptions.

Jumbo Development

The development productivities are based on using modern electric-over-hydraulic twin boom jumbo drills (e.g. Sandvik DD421-60 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 are based on K92’s site productivities (150 m/mo to 175 m/mo per jumbo drill) and were cross-checked by Entech for appropriateness. The schedule has a maximum total of 1,200 m/mo jumbo development at any point in the schedule.

Vertical Development

A development rate of 3 m/day has been applied to all conventional raisebore vertical development for hole diameters between 3.0m and 5.0m. Boxhole activities completed by the ‘Rhino’ (or equivalent) machine are assumed to occur at a rate of 10 m/day for hole diameters <= 1.1 m. These rates include rig up and rig down, pilot hole drilling, hole reaming and ladderway installation (where relevant).

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 5.5t 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.

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, shift change, meetings, meal breaks, breakdowns, maintenance, services installation, and geology / survey control delays.

Backfill

Paste fill has been scheduled using a filling rate of 2,000 cubic metres of paste per day.

Materials Handling

All ore and waste are currently planned to be hauled using conventional 45-tonne underground haulage trucks to the portal transfer point. At peak production, a total of nine trucks will be required. Productivity for a Sandvik TH545 underground truck in this scenario would average 75,000 tkm/mo (tonne-kilometres per month) per unit. The increased twin haulage drive size allows for future increase to the truck size.

24.3.9 Ventilation

Introduction

Entech Pty Ltd (Entech) was commissioned to construct a working model of the ventilation plan for Kainantu underground mine (K92), located in the Eastern highlands of Papua New Guinea (PNG), as part of the PEA study. When a country, like PNG, has no specific regulations or guidelines for mine ventilation, Entech chooses to base design parameters on those specified by DMIRS, Western Australia, and industry best practice.

The scope of work covered by Entech in this report includes:

- Construct a working model for the life of mine (LOM) based on ventilation design by Entech.
- Supply a 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. Not included in the analysis is the potential for mine expansion.

Model Design Parameters & 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 24.3.8.

Table 24.3.8 Ventilation Design Parameters

	Parameter Description	Design Parameter	Remarks
1.	Recommended air velocity in declines to prevent dust generation.	< 6.0 m/s.	Industry best practice for preventing dust downwind of moving equipment.
2.	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.
3.	Intake shaft velocity.	< 20 m/s (unequipped).	Guide for maintaining safe air velocities in underground workings.
4.	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.
5.	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.
6.	Heat modelling parameters set to the 90^{th} 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*
7.	Use of the K92 mine's elevation above sea level for setting barometric pressure.	Elevation set to 800 mRL for the portal.	Elevation estimated for the main haulage portals. Air density is set to this elevation.
8.	WB design limit before applying safety measures.	28°C.	Industry best practice for the prevention of heat stress in workers.
9.	WB stop-work upper limit.	32°C.	Industry best practice for the prevention of heat stress in workers.
10.	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.
11.	K92 airway sizes for modelling purposes.	Main incline = 26 m^2 (arched airway). Dual haulage drives = 34 m^2 & 23 m^2 (arched airways). Return adits = 23 m^2 (arched airway). Return LHR = Ø4.0 m or 12.6 m^2 (RB airway). Return shaft to surface = Ø5.0 m or 19.6 m^2 (RB airway). Internal fresh air intake shaft = Ø3.0 m or 7.1 m^2 (RB airway). Footwall drives for primary vent = 23 m^2 (arched airway).	
12.	Exposure standard for diesel particulates.	< 0.1 mg/m^3 TWA (measured as sub-micron elemental carbon).	Set by Work Safe Australia.
13.	Exposure standard for respirable silica dust.	< 0.05 mg/m^3 TWA.	Set by Work Safe Australia.

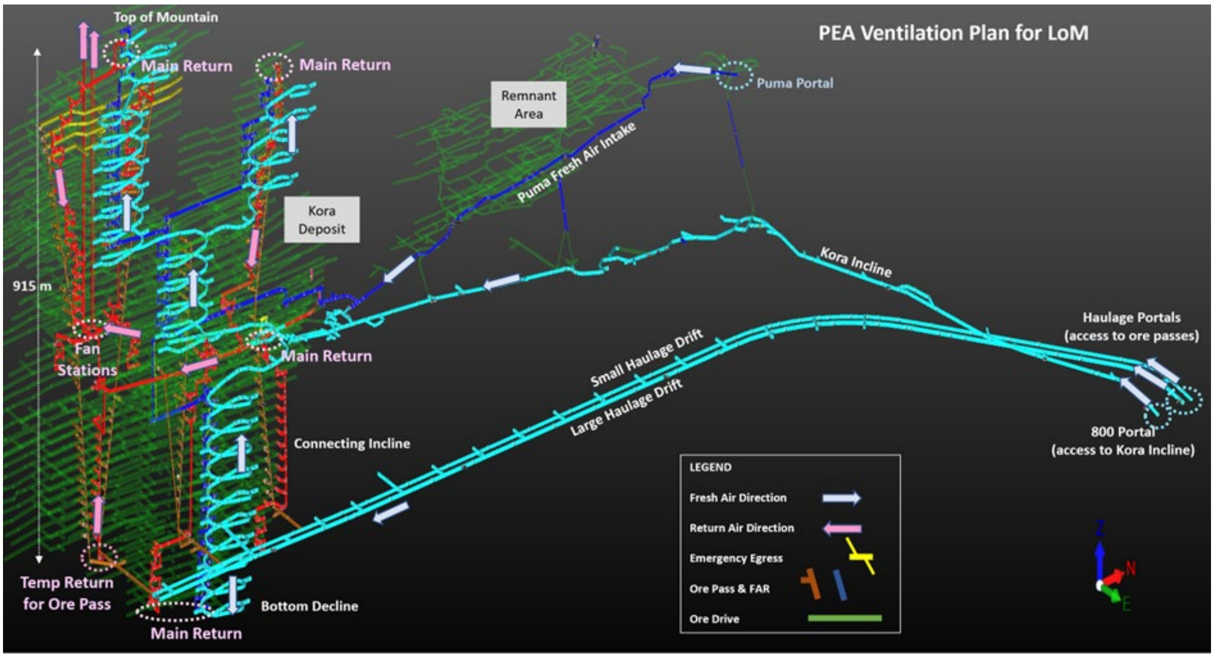
Design assumptions affecting modelling outputs:

- No more than seven production levels operating simultaneously, with those attached to ore passes requiring primary ventilation in footwall drives for loader access to the ore pass.
- No more than one operating level per ore pass at any one time. Primary airflow will be directed to the ore pass draw point on the truck haulage level.
- System leakage and the short-circuiting of air will be minimized as much as possible. Once a level has ceased production the regulator louvres and ore pass accesses will be replaced with bulkheads.
- All remnant connections to surface are blocked except for the Puma portal.

Primary Ventilation

The PEA plan to ventilate K92 underground sees fresh air entering the mine through four portals in the side of the mountain hosting the Kora deposit, with air exhausting via the twin exhaust return air raises daylighting at the top of the mountain. The Puma portal will be repurposed as an intake, which is currently the main exhaust for the existing workings. Figure 24.3.30 illustrates the plan.

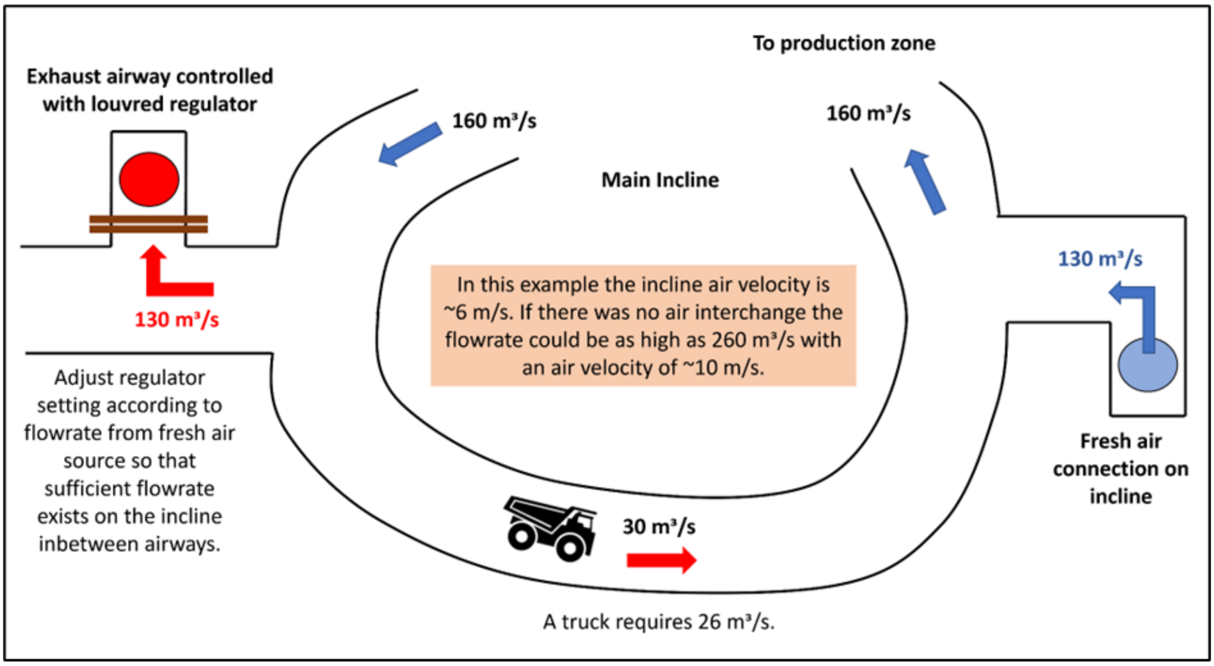
Figure 24.3.30 LOM Ventilation Plan Profile Facing Northwest



Fresh air enters all portals and converges at the mid-point, where production is currently occurring on the Kora Incline. From here a spiral incline continues upward, branching into two inclines near the top of the deposit. Exhausting air links all production zones to two underground fan stations, located together at the base of the two 480 m raises to surface.

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 24.3.31 below illustrates the concept. The Puma intake will link up with the Kora Incline, beyond the mid-point intersection to the bottom of the deposit, through the old workings.

Figure 24.3.31 How an Air Interchange Works



The connecting incline, between the horizontal haulage drifts and the Kora Incline, will be under the influence of an open return at the top of this incline. The two circuits will be separated with a vehicle airlock to prevent air loss and maintain a consistent flow.

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 on haulage routes.

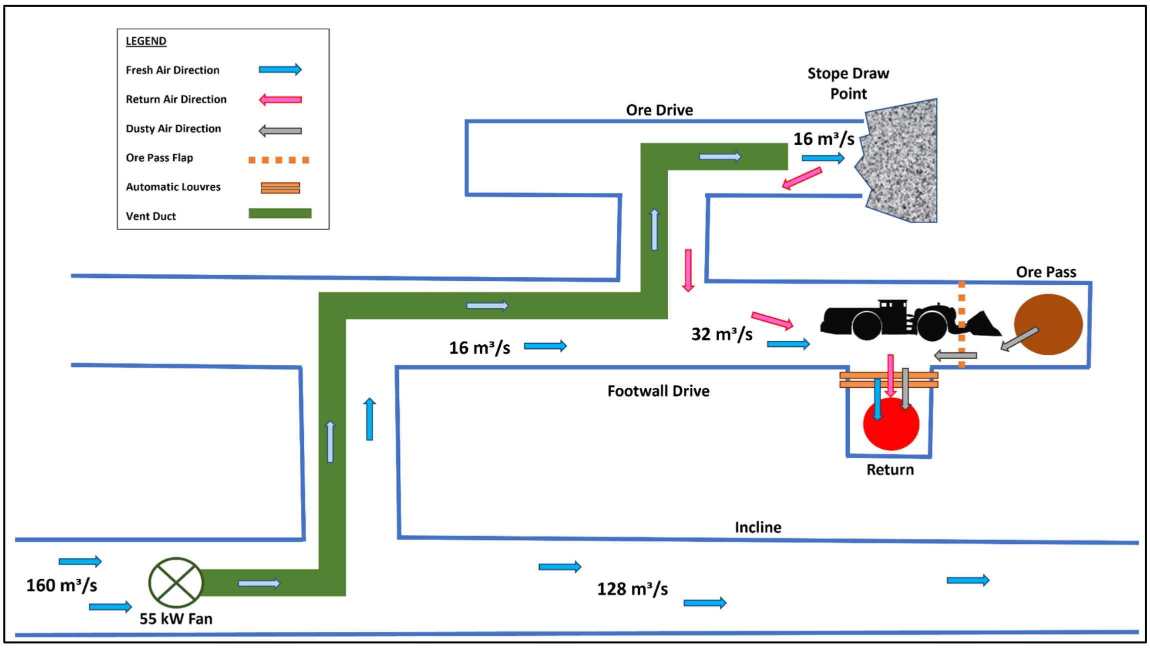
Secondary Ventilation

Production level design will combine secondary air with primary air depending on mining activity.

Development headings will be forced ventilated with 55 kW fans 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 at the end of the drive where ore will be tipped into ore pass. Any dust generated from the ore pass will directly report to the return without polluting the incline. See Figure 24.3.32 for a typical ventilation plan.

Figure 24.3.32 Example of a Production Level Ventilation Plan



Levels without ore pass access, such as the ones below the haulage drift level, will use forced ventilation for the entire level.

Ventilation Analysis

Flowrate Definition by Diesel Exhaust Dilution

Table 24.3.9 contains a list of the mobile equipment intended for use at the mine and shows the peak number of vehicles anticipated with the subsequent flowrate total. According to WA regulations, utilization is not a factor in calculating flowrate, however vehicles which mainly operate under electrical power, i.e., drill rigs, were excluded from the overall calculation. These vehicles, however, will need to be accommodated for when tramming between work sites. A single operator will alternate between the grader and the water cart, hence the grader was omitted from the totals.

Table 24.3.9 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	9	232
Loader	Sandvik LH517	310	16	10	155
Charge-up	Getman	120	6	4	24
Development Drill **	Sandvik DD421-60C	110	6	8	0
Production Drill **	Sandvik DL431-7C	119	6	4	0
Cable Bolter**	Sandvik DS421C	119	6	1	0
Grader	12K Grader	134	7	1	7
Water Cart	Volvo A30D	242	12	1	12
Fibrecrete Sprayer	Normet Spraymec SF 050 D	90	5	2	9
Agitator	Normet LF 700 transmixer	170	9	3	26
IT	Liugong	203	10	5	52
LV	Toyota Landcruiser	151	8	16	124
Total Flowrate for Diesel (m^3/s)					**640**
Leakage @ 15%					96
Total Flowrate Including Leakage (m^3/s)					**736**
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**

***Flowrates calculated for diesel equipment at 0.05 m^3/s per kW of rated engine power.**

****Vehicles 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 peak flowrate of 640 m^3/s for diesel exhaust dilution can be delivered to the working parts of the mine if the fan intakes draw no less than 740 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, if limited by design parameter 1 (see Table 24.3.8) for velocity, will allow a maximum of 160 m^3/s in a travel way before velocities exceed 6 m/s. For this reason, K92 will need to manage the peak number of diesel units between multiple production zones to meet the total flowrate requirement. Table 24.3.10 offers an example of a diesel unit count for a production zone with this incline velocity limit.

Table 24.3.10 Peak Flowrate Calculation According to Expected Mobile Fleet for a Production Area with Single Incline.

Per Production Zone with 160 m³/s Limit.	Assumed Model	Engine Power Rating (kW)	Flowrate Requirement (m^3/s) *	Count	Total Flowrate (m^3/s)
Development Truck	Sandvik TH551	515	26	2	52
loader	Sandvik LH517i	310	16	2	31
Charge-up	Normet MC 605D	120	6	1	6
Development Drill **	Sandvik DD421-60C	110	6	0	0
Production Drill **	Sandvik DL432i-9C	119	6	0	0
Cable Bolter **	Sandvik DS422i	119	6	0	0
Grader	John Deer 670G	134	7	1	7
Water Cart	Volvo A30D	242	12	0	0
Fibrecrete Sprayer	Normet Spraymec SF 050 D	90	5	0	0
Agitator	Normet LF 700 transmixer	170	9	0	0
IT	Liugong	203	10	1	10
LV	Toyota Landcruiser	151	8	4	30
Total Flowrate for Diesel (m^3/s)					**136**
Leakage @ 15%					20
Total Flowrate Including Leakage (m^3/s)					**156**
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**

***Flowrates calculated for diesel equipment at 0.05 m³/s per kW of rated engine power.**

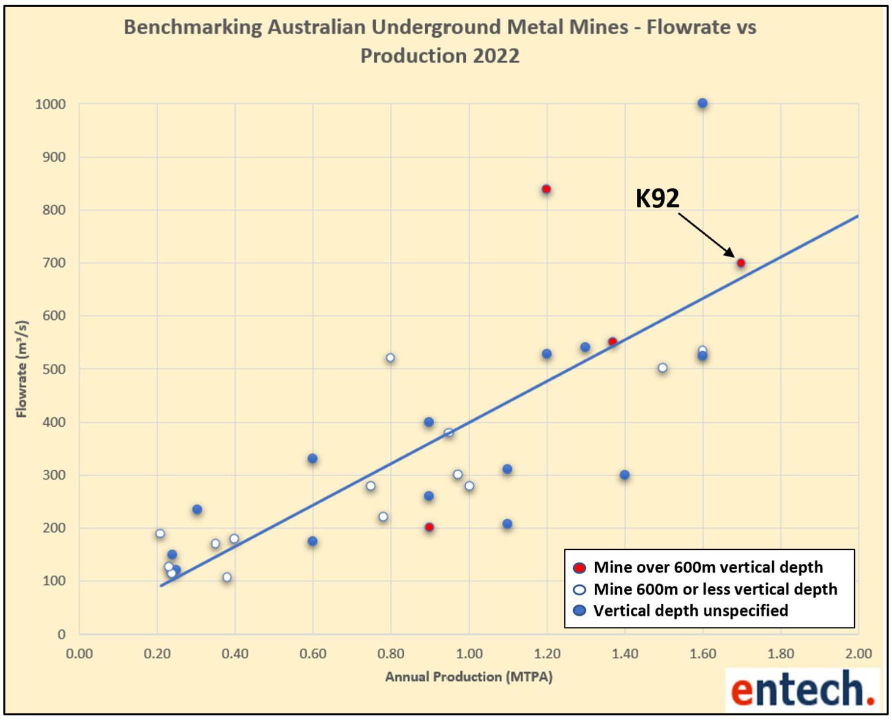
****Vehicles primarily operating under electric power. Flowrate allocation is given when tramming under diesel power, however, unit is omitted in the total count.**

*****Minimum flowrate for activities relying on secondary air. The combination of two fans refers to activities in parallel.**

The truck count in Table 24.3.10 relates to mainly development haulage as most production trucking is isolated to the haulage declines at the base of the deposit.

Benchmarking indicates that the Kainantu mine, with an annual production rate of 1.7 Mt, is representational of other hard rock mines with equivalent flowrate (see Figure 24.3.33).

Figure 24.3.33 Benchmarking Kainantu with Australian Metal Mines



Heat Modelling

A high-level heat analysis was carried out using the following heat settings as per Table 24.3.11.

Table 24.3.11 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	83.8%
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 an average wet bulb temperature ~27°C in the haulage routes, with a reject wet bulb temperature of ~28°C at the bottom most return. The top half of the mine will see an average of ~25°C with a reject wet bulb temperature of ~24°C at the topmost return.

Temperatures at the bottom of the mine are reaching the lower limit for heat stress conditions and consideration should be made to keep personnel safe during the wettest months, when relative humidity is high. Any unscheduled increase in production rate in this area will see the reject temperature move up into control conditions for parameter 8 (see Table 24.3.8).

Ventilation Infrastructure

Airways

The twin exhaust raises to surface, at Ø5.0 m, are sufficient to accommodate the peak flowrate requirements. Reducing the diameter of the RBs is an option considering the shaft air velocity is below the design limit specified at point 5 in Table 24.3.8.

Table 24.3.12 provides a comparison of capital versus operating costs to show how airway size can optimize power costs over the LOM. System pressure reflects the LOM resistance and includes representational electrical fan power in overcoming this resistance.

Table 24.3.12 Capital vs Operating Cost Using Shaft Size

Airway Category	**K92 (11 Year Mine Life) Raise Bore Shaft**			
Diameter (m)	4.0	4.5	5.0	5.5
Drill Cost per Metre ($/m)	6000	6500	7000	7500
Cross-sectional Area (m^2)	12.6	15.9	19.6	23.8
Shaft Length (m)	485	485	485	485
Airflow Quantity Peak Production (m^3/s)	740	740	740	740
Velocity at Peak Production (m/s)	29.6	23.4	18.3	15.7
System Pressure (kPa)	5.7	4.8	4.5	4.1
Power Input Peak Production (kW)	5,510	4,680	4,340	4,000
Estimated RB Cost (AUD)	$2,910,000	$3,152,500	$3,395,000	$3,637,500
Annual Power Cost 0.20/kWh)	$8,205,492	$6,969,456	$6,463,128	$5,956,800
Total Owing Cost (TOC) *	**$45,631,074**	**$39,438,276**	**$37,044,630**	**$34,650,984**

***Eleven-year mine life @8% compound interest**

Having two surface shafts at Ø5.5 m offers a 7% power saving on the Ø5.0 m shaft, however the power savings between the Ø5.0 m and the Ø4.0 m is more substantial across the eleven-year mine life at almost 20%.

Primary Fans

Entech used the following system duties in Table 24.3.13 to select suitable primary fans for the LOM flowrate requirements. 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 24.3.13 System Duties for Meeting Flowrate Requirement

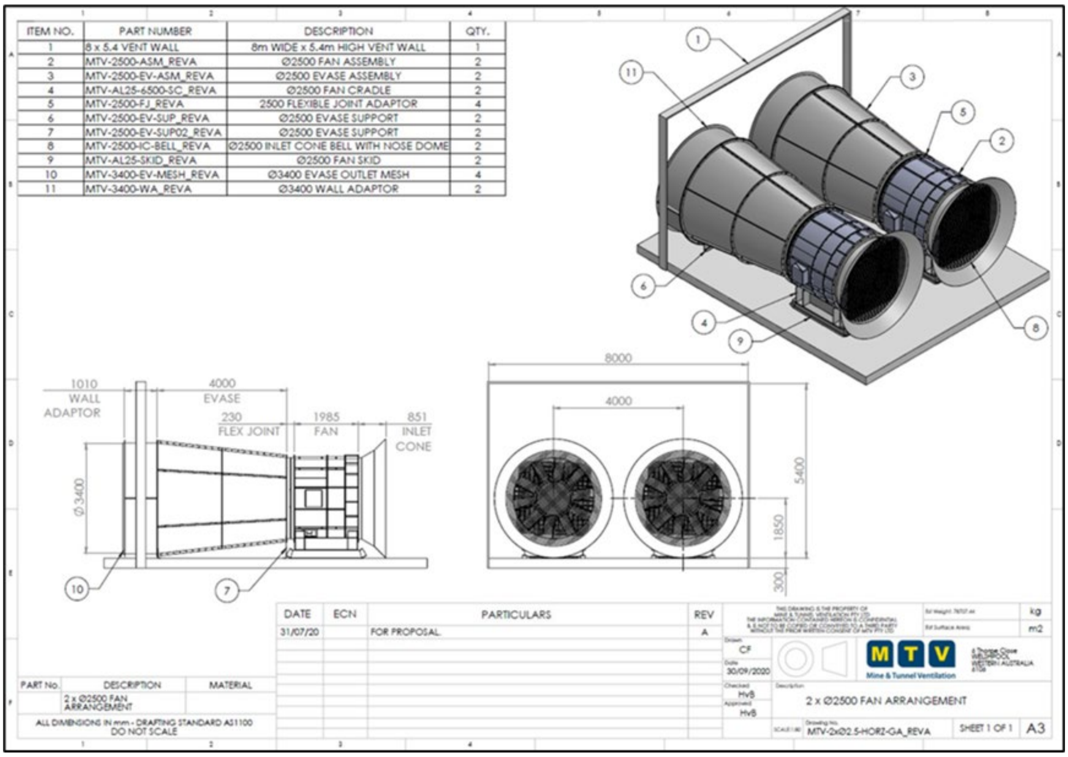
K92 System Duties for the LOM							
Staging	**Mine Resistance (Ns^2/m^8)**	**Electrical Input (kW)**	**System Total Pressure (Pa)**	**System Static Pressure (Pa)**	**Inlet Density (kg/m^3)**	**Flow (kg/s)**	**Flow (m^3/s)**
Development Milestone	0.0089	4,700	4,900	4,792	0.95	703	740
Peak Flowrate @ LOM	0.0082	4,340	4,500	4,389	0.95	703	740
Min Flowrate @ LOM	0.0089	320	800	780	0.99	297	300

The ‘Development Milestone’ in Table 24.3.13 represents the stage in development when the RAR is commissioned, and Puma exhaust becomes the Puma intake. The mine resistance is considerably higher at this point as the connecting incline to the haulage drifts has not yet broken through to the haulage drifts.

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 1.5 MW axial flow fans in parallel would suit each fan station, but this will need to be verified by the supplier. Figure 24.3.34 offers an example of what an underground fan station with two 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-representational of the Kainantu mine.

Figure 24.3.34 Example of Underground Fan Station Care of MTV



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 minimizing short circuiting between levels and allows the loader free access 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. A ventilation on demand (VOD) system can be used to ensure the incline airflow is consistently maintained at the desired flowrate. Examples of ventilation control devices are shown in Figure 24.3.35 and Figure 24.3.36.

Figure 24.3.35 Example of Flap, or Strip Curtain That Can Be Used at Ore Pass Access

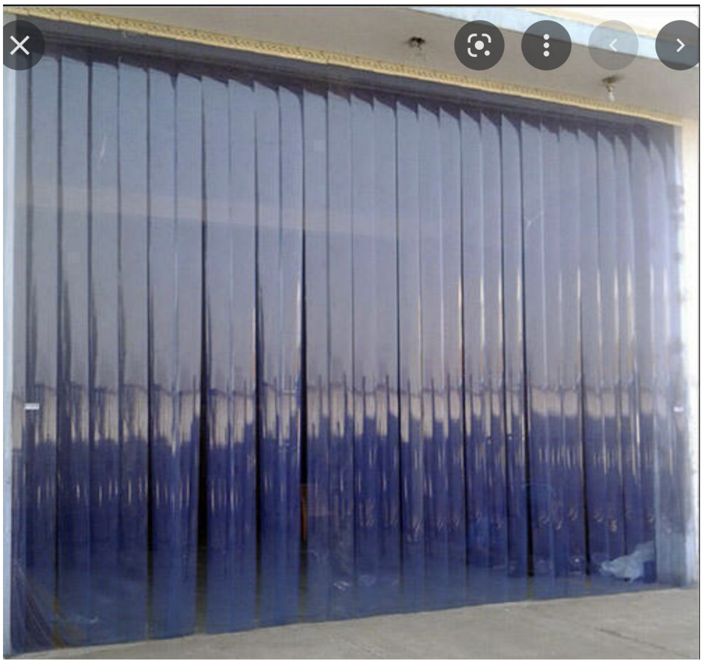
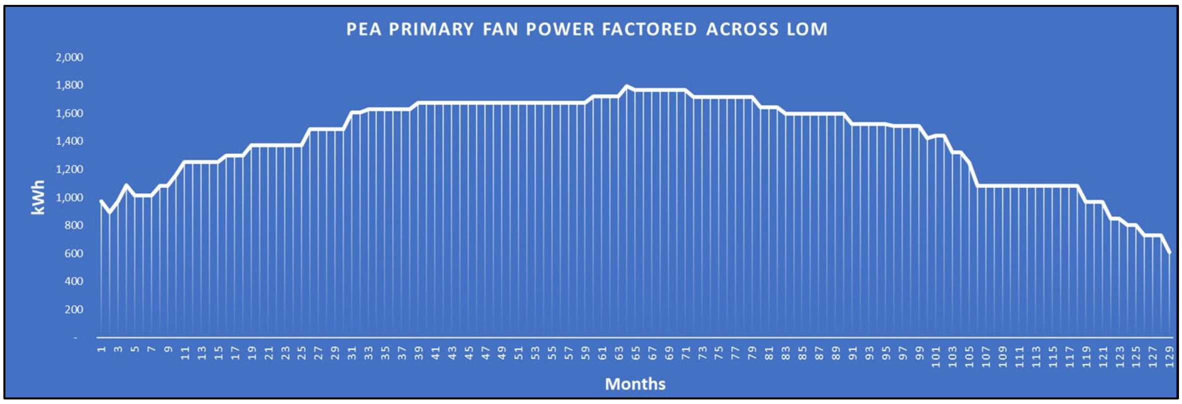


Figure 24.3.36 Example of Ventilation Louvres Care of Clemcorp and Wilshaw

Power

The primary fan power across the LOM is represented in Figure 24.3.37 and is influenced by changes in the mine's resistance. Power output is determined according to a power factor determined from the LOM resistance and applied to the flowrate according to diesel fleet count.

Figure 24.3.37 Primary Fan Power as Average kWh per Scheduled Month

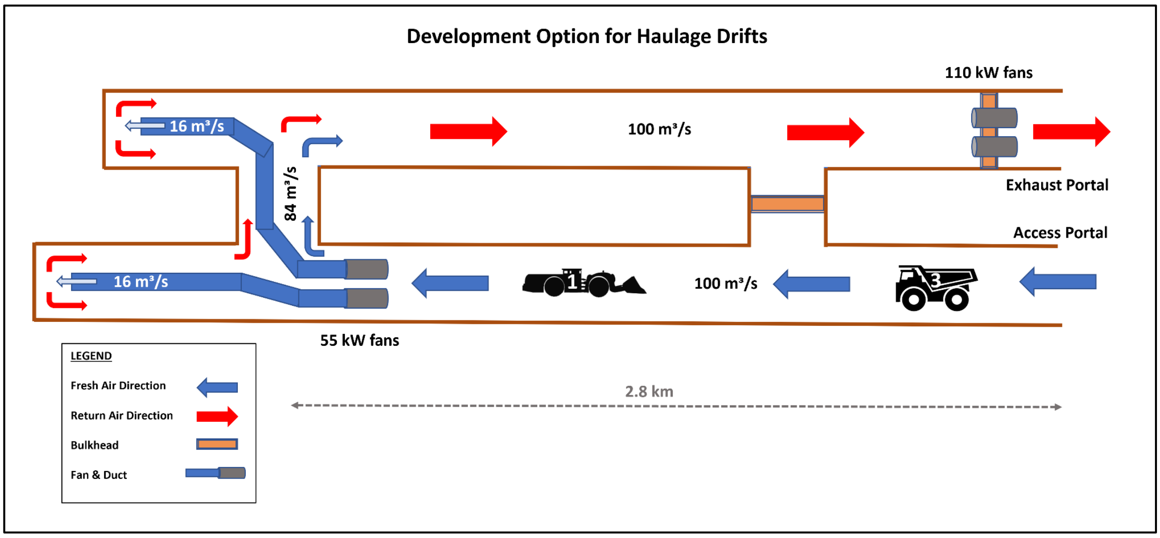


Recommendations and Conclusions

- Use speed controls, like VSDs, on primary fans to change the available flowrate and optimize power. This will prevent overventilation when peak flow is not required.

- Developing the 2.8 km long twin haulage drifts will require two 132 kW auxiliary fans per drift with twin lengths of Ø1.5 m ducting per drift. A simpler option draws primary air into the drifts, as close to the development advance, as the last stockpile connection between the two drifts affords (see Figure 24.3.38). This will create a ventilation circuit loop, providing all other stockpile connections are sealed off along the route, although vehicle access in and out of the workings can only be done via a single access portal. Two 110 kW fans in parallel, plugged into a temporary bulkhead, will provide the flowrate for three trucks and one loader.

Figure 24.3.38 Initial Development Option for Haulage Drifts



24.3.10 Underground Infrastructure

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 all personnel are able to reach a decline connection drive, ladderways are installed in the raise-bored escapeway rises which are fully contained in fresh air.

In certain cases, escaping the mine via a second means of egress in an emergency situation 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.

To provide refuge for underground personnel in such circumstances, re-locatable refuge chambers are installed in the mine. The chambers vary in size from six to 20 -man capacity. Six-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.

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 along existing 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.

The number of substations has been estimated using an average of range of 100 m – 150 m vertical distance between sub stations, with most 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 site power upgrade is planned to meet the anticipated peak power demand of approximately 5.2MW.

Service Water

Water captured by the dewatering system is settled and a portion is recycled for use in the underground mine. It is managed internally where it is captured and distributed to working areas.

Hydrogeology and Dewatering

The following is an excerpt from the 'Kora Hydrogeology Assessment' completed by EMM[1]:

"Hydrogeology conditions are highly heterogeneous and primarily controlled by local topography, local recharge zones and mineralization-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 which facilitates 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 60-110 L/s. The majority of the 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 300 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 backs. Additional dewatering infrastructure will be required for mining operations below 1,185 m RL. 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 incline will be completed in 2023, changing the nature of dewatering. The twin incline will intersect the Kora resource area at approximately 900 m RL, this will become the new locus of mine-discharge collection and transfer to the 800 Portal (largely negating the need for pumping to the 1,185 m RL sump / dam). Pumping infrastructure will be required for Kora mine-development below the 900 m RL. Risks associated with uncontrolled groundwater discharge will increase as underground mining depths increase.

[1] Kora Hydrogeology Assessment – EMM Dec 2021

K92's 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 2024. Paste backfilling will reduce mine-throughflow, and therefore, slow the effects of ARD. Paste backfilling is planned for Kora mining regions; however, paste backfilling may also be required in historical (Irumafimpa) underground workings to reduce 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 K92 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 summarized 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 characterize 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."

Primary Pumping

The underground mine dewatering strategy utilizes 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 1185 m RL. From here the water is discharged via the pipes as shown in Figure 24.3.39 where it reaches the portal and discharge to a weir system where it is treated and subjected to water monitoring.

Figure 24.3.39 Central Dewatering System 1185 m RL

Prior to the completion of excavation of the twin haulage drives, dewatering will utilize the current site methodology and once the twin haulage drives have been excavated, mine water will report to the haulage level where it will be discharged to the twin haulage incline portals.

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 1185 mRL, until the final system is established at the completion of twin haulage drives. For pumping up to the 1185 mRL there is also an allowance for a Triple WT114 mono pump, and a Triplex Vertical Multi-Stage Booster pump.

Compressed Air

There are two compressors in place that supply compressed air to the underground.

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 is capable of communicating with users of the surface UHF system (e.g. mill control, surface operations) via a dedicated surface-underground channel.

24.3.11 Underground Operations

Mining Fleet

Assumptions for mine equipment requirements have been derived by K92 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 24.3.14 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 24.3.14 Peak Fleet Numbers

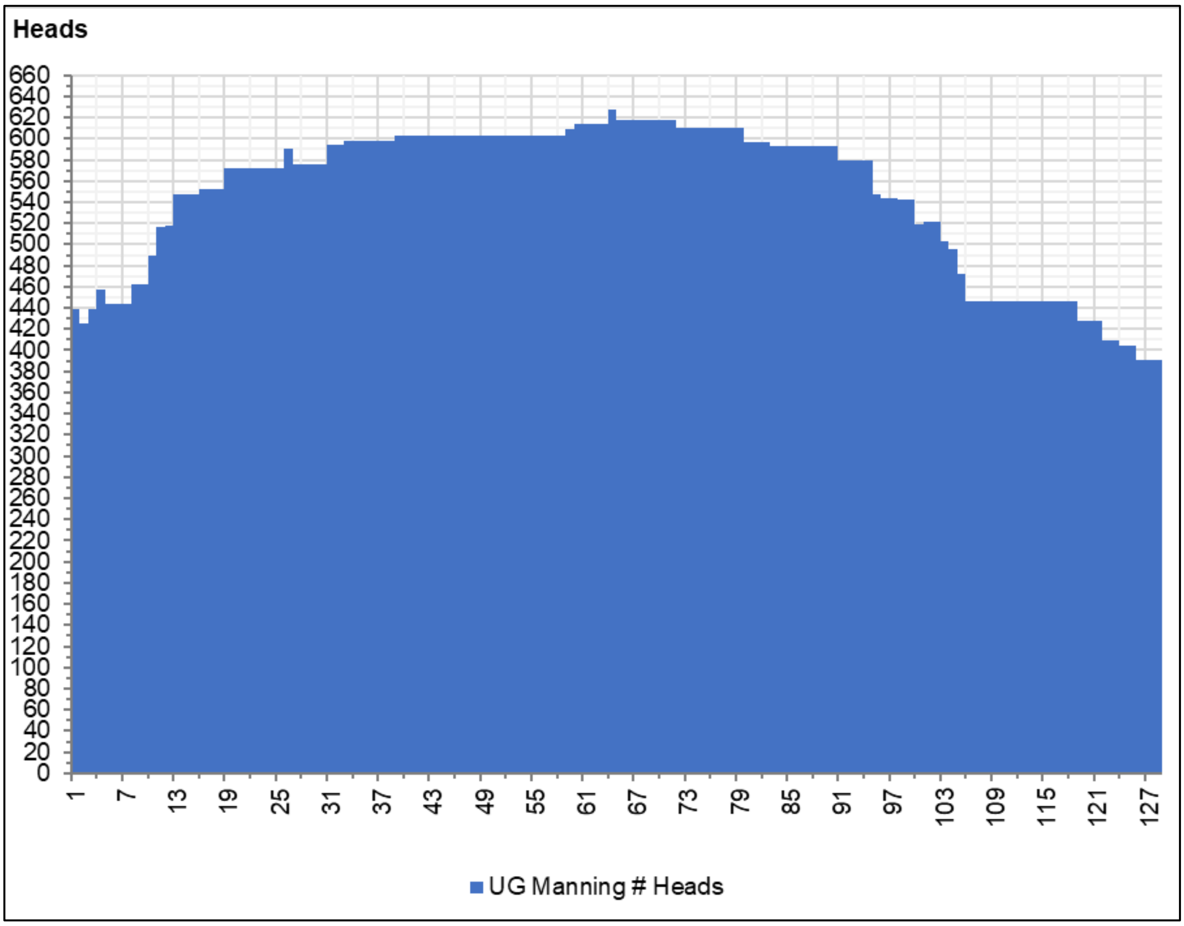
Equipment	Quantity
Primary	
Development Drills (DD421-60C)	8
Loaders (LH517)	10
Trucks (TH545)	9
Production Drills (DL421-7C/DL431-7C/DS 421C Bolter)	5
Slot Drill – 'Rhino' (or equivalent)	1
Production Charge-up (Getman)	2
Development Charge-up (Getman)	2
Ancillary	
Spraymec (6050WP)	2
Agitator	3
IT 856H Development	3
IT 856H Production	2
12K Grader	1
Scissor Lift (Ultimec)	1
Light Vehicles	16
Raisebore	1

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 for expatriate employees.

Peak underground personnel requirements (total employed personnel, not all on site simultaneously) by month are found in Figure 24.3.40.

Figure 24.3.40 Total Underground Mining Personnel Per Month



Mine Facilities

The site layout is illustrated in Figure 24.3.41.

Figure 24.3.41 Overall Site Layout (~6km from Mine Portal to Processing Plant)

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 which is in close proximity to the processing plant approximately 6 km from the mine portal.

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.

Fuel Storage

Diesel is stored on surface in existing licensed facilities.

Explosives Storage

There is an existing surface magazine compound.

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 was provided to a mining contractor who provided a pricing estimate for these activities.

Mining Capital Costs

The estimated capital costs are summarized in Table 24.3.15.

Table 24.3.15 Mining Capital Cost Totals

Description	Unit	Value
Infrastructure	USD (M)	14.4
Decline	USD (M)	28.5
Cap Access	USD (M)	11.4
Ventilation	USD (M)	46.4
Escapeway	USD (M)	3.7
Other Lateral Development	USD (M)	66.8
Fleet	USD (M)	77.8
Operators and Maintenance	USD (M)	92.0
Capital Mine Services	USD (M)	24.3
Capital Mine Overheads	USD (M)	18.1
Total Capital	**USD (M)**	**383.5**

A breakdown of the capital unit costs is shown in Table 24.3.16.

Table 24.3.16 Mining Capital Unit Costs

Description	Unit	Value
Infrastructure	$ / t ore	1.18
Decline	$ / t ore	2.34
Capital Access	$ / t ore	0.94
Ventilation	$ / t ore	3.82
Escapeway	$ / t ore	0.31
Other Lateral Development	$ / t ore	5.50
Fleet	$ / t ore	6.40
Operators and Maintenance	$ / t ore	7.57
Capital Mine Services	$ / t ore	2.00
Capital Mine Overheads	$ / t ore	1.49
Total Capital Cost	**$ / t ore**	**31.55**

Mining Operating Costs

The estimated operating costs are summarized in Table 24.3.17.

Table 24.3.17 Mining Operating Cost Totals

Description	Unit	Value
Operating Access	USD (M)	18.7
Ore Drive	USD (M)	88.0
Stope	USD (M)	246.1
Operators and Maintenance	USD (M)	207.7
Operating Mine Services	USD (M)	59.2
Operating Mine Overheads	USD (M)	48.2
Surface Haulage	USD (M)	68.7
Grade Control	USD (M)	35.7
Total Operating	**USD (M)**	**772.4**

A breakdown of the operating unit costs is shown in Table 24.3.18.

Table 24.3.18 Mining Operating Unit Costs

Description	Unit	Value
Operating Access	$/t ore	1.54
Ore Drive	$/t ore	7.24
Stope	$/t ore	20.25
Operators and Maintenance	$/t ore	17.09
Operating Mine Services	$/t ore	4.87
Operating Mine Overheads	$/t ore	3.97
Surface Haulage	$/t ore	5.65
Grade Control	$/t ore	2.94
Total Operating	**$/t ore**	**63.54**

24.4 Kora 2022 PEA Processing and Infrastructure

24.4.1 Kora 2022 PEA Processing Facility

The processing facility for the PEA case remains unchanged from what is described in Chapter 17 of this report. The only change to processing being that the existing plant (0.5 Mtpa) is run in parallel with the DFS plant (1.2 Mtpa) which achieves a combined throughput of 1.7 Mtpa.

24.4.2 Kora 2022 PEA Non-Process Infrastructure

Power

For the PEA case, further expansion is required for the back-up power.

Table 24.4.1 Electric Power Consumption for the PEA Case

Area	Install Power kW	Average Demand	Peak Power	Comments
Camp	1,650	1,100	1,200	
0.5Mtpa Process Plant	5,000	1,600	2,500	
1.2Mtpa Process Plant	7,973	5,606	6,378	
Paste Plant	4,602	2,778	3,018	
Mine	3,226	1,191	2,581	
Total	**16,295**	**10,064**	**12,484**	
		Total Peak Power	**15,677**	
		Genset Peak Power	**14,477**	Only Paste plant infrastructure and 1.2 Mtpa Mill on the Gen-set upgrade
		New Gen-sets Required	**10**	1600kVA/11kV gen-sets

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 18.9 in this report.

24.4.3 Kora 2022 PEA Tailings Storage

The results of the PEA suggest that it is feasible to consider replacing the existing processing plant with a capacity of 500,000 tpa to treat an increase in ore production of 1,700,000 tpa from the Kora deposit. However, the production expansion will consequently increase the total tailings generated, and preliminary assessments advise that insufficient capacity to store those additional tailings is available in the existing TSF.

The tailings generated due to the Kora Expansion PEA scenario will account for 12.3 million tonnes during the 2022 – 2032 period, as presented in Table 24.4.2. From 2022 to mid-2024, the entire production of tailings will be stored in the existing TSF. Once the paste backfill plant commences operation by mid-2024, an average of 46% of the tailings generated will be processed through the paste backfill plant and the remaining portion stored in the TSF. The tonnes of tailings processed through the paste backfill plant and tonnes stored in the TSF are presented in Table 24.4.3.

Table 24.4.2 Kora Expansion PEA Estimated Production

Year	Mill Processed Tonnage (t)	Tailings Tonnage (t) (*)	Cumulative Tailings Tonnage (t)
2022	435,000	413,250	413,250
2023	500,000	475,000	888,250
2024	850,000	807,500	1,695,750
2025	1,200,000	1,140,000	2,835,750
2026	1,500,000	1,425,000	4,260,750
2027	1,700,000	1,615,000	5,875,750
2028	1,700,000	1,615,000	7,490,750
2029	1,700,000	1,615,000	9,105,750
2030	1,700,000	1,615,000	10,720,750
2031	839,557	797,579	11,518,329
2032	160,788	152,749	11,671,078
Total	**12,285,345**	**11,671,078**	

Table 24.4.3 Tailings Distribution Over the Life of Mine

Year	Tailings generated (t)	Tailings to TSF (t)	Tailings to Paste Backfill (t)
2022	413,250	413,250	-
2023	475,000	475,000	-
2024	807,500	616,047	191,453
2025	1,140,000	642,636	497,364
2026	1,425,000	745,432	679,568
2027	1,615,000	861,139	753,861
2028	1,615,000	865,579	749,421
2029	1,615,000	804,774	810,226
2030	1,615,000	605,216	1,009,784
2031	797,579	334,655	462,924
2032	152,749	43,504	109,245
Total	**11,671,078**	**6,407,232**	**5,263,845**

Since the start of the K92 operations, approximately 1,073,916 tonnes of dry tailings have been generated and stored within the TSF. The volume of tailings stored is expected to be around 894,930 m^3, assuming the average settled density of tailings deposited at 1.2 t/m^3. Therefore, the total tailings generated from 2022 to 2032, as a part of the PEA production scenario, is expected to be 11.6 million tonnes, with 5.2 million tonnes sent to the underground through the paste backfill plant and the remaining 6.4 million tonnes stored in a TSF. Assuming that tailings would be stored at an average settled density of 1.4 t/m3, approximately 4.6 Mm^3 of volume capacity is required. Therefore, at the end of the PEA production plan scenario, the total volume of tailings deposited since the commencement of the operation would be approximately 5.5 Mm^3.

Future raises of the embankment for the existing TSF with their corresponding volume, and total remaining capacity for tailings storage are illustrated in Table 24.4.4. For instance, a storage capacity of 2.15 Mm^3 would be available by raising the dam to Stage 2 (RL 520m); on the other hand, raising the dam to Stage 3 (RL 530m) will provide a maximum storage capacity of 3.54 Mm^3.

The construction activity for an embankment at the site is very due to near continuous rainfall and the limited availability of suitable borrow material. Likewise, the construction of the current downstream embankment lift has a very low efficiency based on the ratio of storage volume increase over the required rockfill material to build the embankment. Table 24.4.5 compares the volume of rockfill material needed to build the dam with the increment in the volume capacity and associated earthworks efficiency for each raise. The results suggest that limiting the rise of the embankment for the existing TSF to Stage 3 would be recommended from an earthwork efficiency point of view.

Comparing the volume of tailings required to be stored with the maximum volume capacity of the existing TSF, 5.5 Mm^3 and 3.54 Mm^3, respectively. Empirical calculations have confirmed that there is insufficient capacity in the existing TSF to cope with the full PEA production. Further, the impact on the efficiency of rockfill required for additional volume capacity suggests that raising the existing TSF beyond the current planned stages would not be efficient.

Exploring a new TSF option to commence operation after stage 3 of the existing TSF exhaust by 2027 is the preferred option for the PEA production plan scenario. Figure 24.4.1shows the progressive sequence of embankment construction for the existing TSF and the growing line of a new facility starting operation by 2028.

Table 24.4.4 Tailings Volume Capacity for TSF Embankment Raises

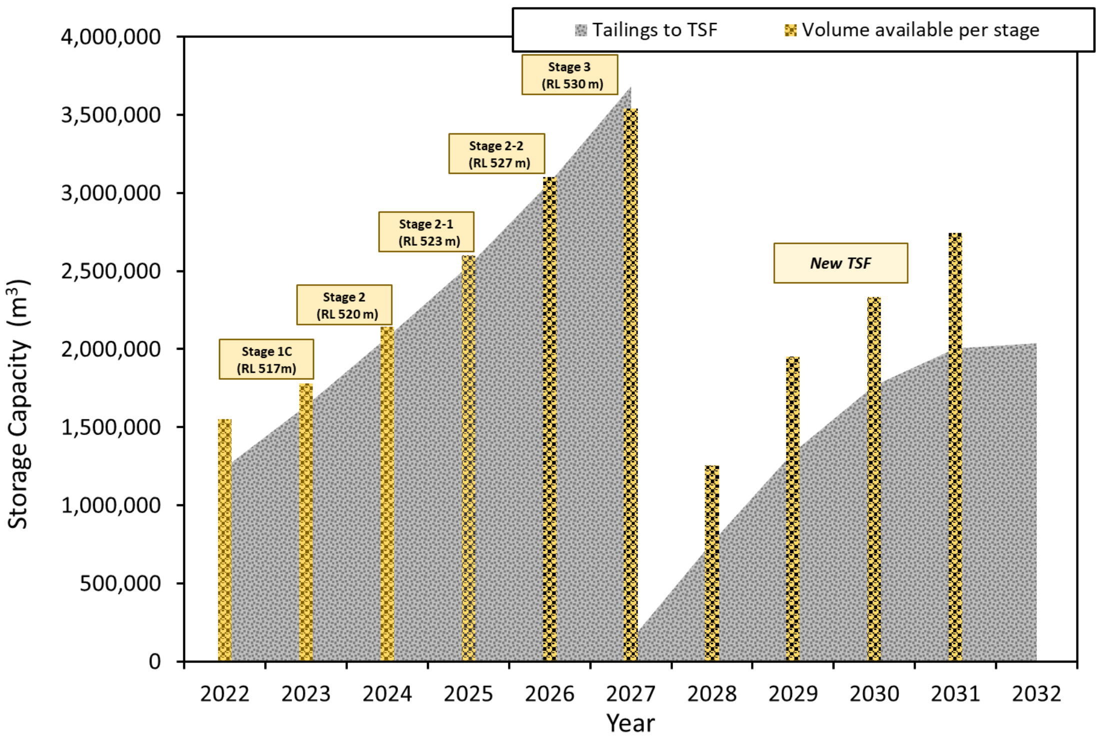
TSF Raise Stage	Crest Level (RL m)	Total volume (m^3)	Remaining capacity (m^3)
1A	512	1,227,000	332,070
1B	515	1,552,000	657,070
1C	517	1,782,000	887,070
2	520	2,145,000	1,250,070
3 (Conceptual)	530	3,540,000	2,645,070

Table 24.4.5 Volume Capacity and Embankment Rockfill Ratio

TSF Raise Stage	Embankment Crest Level (RL m)	Additional Storage Volume (m^3)	Required Rockfill Volume (m^3)	Earthworks Efficiency (*)
1B	515	325,000	158,000	2.1
1C	517	230,000	116,600	2.0
2	520	364,000	247,200	1.5
3 (Conceptual)	530	1,395,000	1,237,500	1.1

(*) Earthworks efficiency equals to the ratio of storage volume over the required rockfill embankment volume. Higher values mean more efficiency

Figure 24.4.1 Embankment Raising Sequence During PEA Scenario



24.5 Criteria to Evaluate the Options of a New TSF

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.

The main criteria considered for adopting solutions for a new tailings facility should include the following aspects:

- The solution should minimize the overall footprint.
- The solution shall increase the in-situ density of the tailings.
- Compatible with the complex site-specific geological and geotechnical conditions.
- Adequate to the high seismicity.
- Minimize the requirements of borrow material.
- Reduce the amount of water released from the tailings impoundment.
- Maintain the 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 a progressive reclamation.

The site is currently licensed to release up to 619,200 m^3 of water per year after being processed to the environment, as outlined in the terms and conditions of the environmental permit issued under *Section 65 of the Environment Act 2000, Condition 50*.

The Definitive Feasibility Study for the 1,200,000 tonnes per annum plant indicates a neutral water balance. As a result, the amount of water released to the environment will be as per the current levels, which may require the following actions:

- Update the site license to increase the limit of water release.
- Undertake water balance modelling to estimate the volume of water that will be stored within the TSF.
- Undertake additional water storage design options based on the water balance modelling result.

24.6 Evaluation of TSF Options

Evaluating different solutions to store the PEA's tailings, including the increased capacity in the existing TSF or in a newly constructed tailings facility. The critical constraints were identified from the several options evaluated; for instance, duration of implementation, understanding the foundation conditions, tailings characterizations, availability of suitable borrow material, and site limitations which are discussed further below.

First, the time required to implement a new facility for the PEA Kora Expansion shall be considered at least three years from now, considering the site investigation, options assessments, and the development of the engineering, construction, procurement activities and regulatory aspects. Second, the currently poor understanding of the foundation conditions for a new TSF location due to the geological and geotechnical complexities typical of PNG would not necessarily ensure that selected sites would be suitable for installing a new facility. Third, there are additional learnings to be understood of the characteristics of the tailings, and the expected variation for those generated from Kora in the coming years create uncertainties for suitable mechanical dewatering solutions. Further, PNG's highly weathered soils do not generally provide optimal conditions for earthwork, so borrowing material to build a new dam is currently considered uncertain. Finally, the site limitation due to the remote location, limited availability of skilled labour, and logistics difficulties may constrain most of the best available technologies from being implemented.

The evaluation outcome for TSF options suggests two potential solutions. First, a conventional TSF with a 25m height rockfill dam embankment built with borrowed material and thickened tailings discharge subaerial into the TSF. Second, the classification of tailings coarse and fine particles through hydrocyclones, coupled with the construction of a 15m dam using geotubes filled with the fine fraction of classified tailings and stacked on top of each other; the coarse fraction of tailings will be stored as a sand stockpile within the impoundment created by the geotubes dam. The high permeable coarse fraction would quickly drain by gravity and is expected to reach a higher settled density than the current tailings stored in the TSF. While the fine fraction is considered the most complex tailings, the deposition of that fraction within the geotubes would allow their encapsulation and ability to build the dam to create the impoundment to store the coarse fraction.

Both solutions will provide enough volume to store the tailings generated for the PEA Kora Expansion after the capacity of the existing TSF at stage 3 exhausts by 2027. For the purpose of this assessment, K92 have adopted the implementation of conventional TSF with a rockfill dam as the preferred solution.

The total CAPEX and OPEX cost (without considering closure costs) for the adopted solution of a conventional TSF with a rockfill dam is US$ 52.33 million. The total estimated CAPEX to increase the existing TSF capacity and implement the new TSF is US$ 45.49 million. Other capital costs (not included in the above total) include researching tailings to use as a material to build the dam's embankment and increase the settled density of tailings deposited. The annual operating cost for tailings management ranges from US$ 0.22 to 0.58 million (value per year does not consider discount rate). The highest components of the annual operating expenses are related to consultants' activities ,consumables and costs of spares / replacement parts.

24.7 Kora 2022 PEA Paste System

Introduction

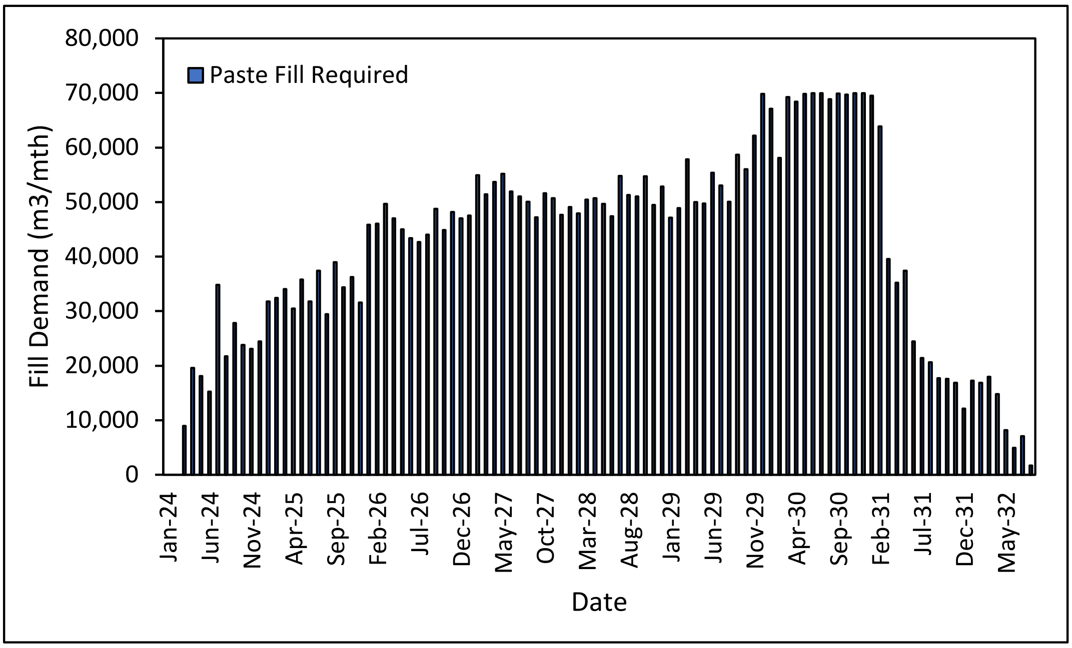
As part of future expansion considerations, K92 are investigating the possibility of increasing total mine production rate to 1.7 Mtpa. As backfill is integral to sustainable mine production at K92, an increase in instantaneous paste production rate is necessary to support the increased productivity. This section presents analysis of the backfill demands (for the 1.7 Mtpa production rate) and to determine the infrastructure requirements necessary to satisfy this demand.

Backfill Demand

In the proposed 1.7 Mtpa mining plan, Kora stope geometries and the extraction sequence remain similar to that described in Section 13.14 , therefore the required physical requirements of the fill remain unchanged. Furthermore, stope sublevel intervals are to remain at 20 m and as such the bulkhead design and filling philosophy described in Section 13.14 are expected to remain valid.

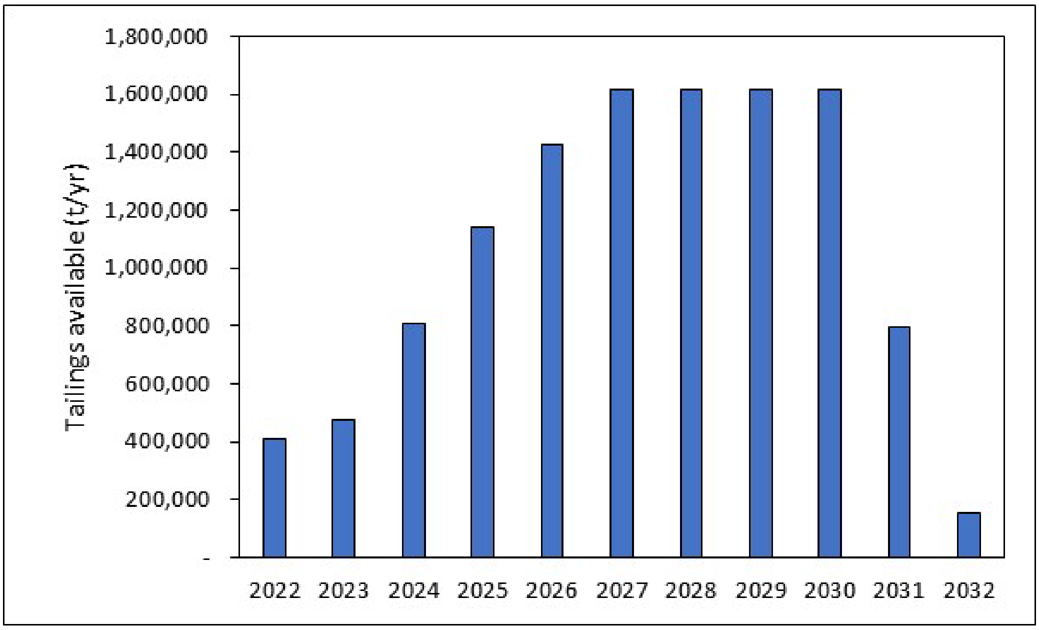
The only tangible change associated with an increase in fill demand is an increase in instantaneous fill production rate. For the proposed mining sequence, the scheduled monthly backfill demand is presented in Figure 24.5.1. This figure shows that the monthly backfill demand ramping up to a peak of 70,000 m^3/month, which is sustained throughout 2030 before reducing thereafter.

Figure 24.5.1 Monthly Backfill Demand for 1.7 Mtpa Production



The estimated mass of tailings available for backfill is plotted in Figure 24.5.2. This figure shows that, during the period where the highest backfill demand is required, a total of 1,615,000 tpa of tailings is expected to be available.

Figure 24.5.2 Annual Mass of Tailings Available for Backfill Operations



Assuming a process plant availability of 92%, an annual tailings production of 1.62 Mtpa is expected to equate to an instantaneous production rate of 200 t/hr. With an in situ dry density of 1.23 t/m^3, this tailings mass is capable of filling an in-situ volume of 163 m^3/hr. At this fill rate a system utilization of 60% is necessary to achieve the required fill rate of 70,000 t/mth. Given this utilization we recommend that the paste system be designed to utilize all available tailings.

At the proposed paste solids content of 67% and average binder content of 3.2% each unit volume of paste (pulp) is expected to contain 1.15 t of tailings. On this basis, if utilizing the entire (200 t/hr) tailings stream the paste system is to be design for flows of 174 m^3/hr.

Paste System Process Design Criteria

To achieve the PEA mine backfill requirements, the system design criteria presented in Table 24.7.1 must be satisfied.

Table 24.7.1 Kora Paste System General Design Criteria

General		
Description	**Units**	**Quantity**
Site Conditions		
Altitude (Process Plant)	mAMSL	507
Altitude (800 Portal)	mAMSL	904
Climate Type		
Min Temperature	deg C	13
Max Temperature	deg C	25
Min Montly Average Rainfall	mm/mth	102
Min Montly Average Rainfall	mm/mth	317
Yearly Average Rainfall	mm/yr	2,410
Relative Humidity		
Average Relative Humidity	%	87%
Design Relative Humidity	%	90%
Tailings Characteristics		
Tailings SG	t/m^3	2.86
Target Grind P80	micron	110
Tailings % solids (by mass)	%	35
Tailings pH		7
Tailings Temperature	deg C	25
Raw Water Characteristics		
pH Range		6.5-8.0
Temperature Range	deg C	15-25
Specific Gravity	t/m^3	1.000
Total Suspended Solids	ppm	5
Chloride	mg/L	6
Sulphate	mg/L	1600
Binder Material Properties		
Binder Product #1		Granulated Ground Blaste Furnace Slag
Material SG	t/m^3	3.0
Bulk Density (nominal)	t/m^3	1.1
Binder Product #2		GP Cement Blend
Material SG	t/m^3	3.2
Bulk Density (nominal)	t/m^3	1.2
Backfill Plant Operating Duty		
Operating hours per day	h	24
Operating days per year	days	219
Availability	%	85%
Operating hours per year	h	5256
Paste Fill Plant Dry Tails Feed	dry t/a	1052200
Paste Fill Plant Dry Tails Feed	dry t/hr	200
Paste Production		
Mass	tph	307
Flow	m^3/hr	175
Paste Density (excluding binder)	t/m^3	1.76
Tailings Feed		Final Tailings
Tailings Source		Thickener Underflow
Mining Detail		
Sub Level interval	m	20
Length (along strike)	m	20
Stope width		
Average	m	7
Minimum	m	3
Maximum	m	30
Stope Fill Time		
Average	hrs	16
Minimum	hrs	7
Maximum	hrs	53
Stength Requirements		
Average	kPa	550
Minimum	kPa	275
Maximum	kPa	2800
Slag Requirements		
Weighted Average	t/m3	4.8
Minimum	t/m3	4.0
Maximum	t/m3	19.0
GP Requirements		
Weighted Average	t/m3	2.0
Minimum	t/m3	1.0
Maximum	t/m3	3.1

Paste Plant Process Design

Introduction

Figure 24.5.2 shows that, during the period 2024-2026, the available tailings remain similar to flows adopted for the DFS before increasing significantly beyond 2026. This transition in available tailings would require the paste system to have the ability to operate over a design range of 122-174 m^3/hr.

At a mine production rate of 1.7 Mtpa, the processing plan involves operating both the existing processing plant and the new processing facility in parallel. A paste system that can be operated across the full range of production rates will increase operational flexibility by allowing paste to be placed during periods that one part of the processing operation is disrupted (i.e. a lesser quantity of tailings is available).

To address this design requirement, the approach taken is to maintain the same process as that defined in the DFS (Section 13.17, 18.4) and utilize as much of the DFS infrastructure as possible, whilst ensuring all major equipment is sized to achieve peak production rates.

To meet the increase in backfill demand the following changes to the DFS process previously detailed in Section 13.14 and 18.4 are required. The following sections present the process changes at each of the process area, 800 portal area, 1205/1170 (underground) area, and distribution system, independently.

Paste Plant Process Area

The major infrastructure changes required at the Process Area site are associated with increased flows and the need for additional filtration capacity.

To increase filter cake production the following modifications were implemented:

- A new filter feed pump added to the thickener boot, with direct piping to the new filter.
- Increased flocculant batching plant capacity.
- Duplication of the filtration process including:
  - Filter
  - Filtrate receiver
  - Vacuum pump
- Expansion of the vacuum pump cooling tower capacity.

To accommodate the increase in hydraulic paste transport requirements (from 122 m^3/hr to 174 m^3/hr) all pipelines are maintained at 200 NB and:

- The capacity (centrifugal) piston diaphragm feed pumps is to be increased.
- The piston diaphragm pump capacity is increased.

The updated equipment list for the Process Plant Area is presented in Table 24.7.2, where equipment that is duplicated is shaded in yellow and equipment that is upsized (relative to the DFS design) is shaded in grey. The final column shows the equipment that should be upgraded in the DFS (122 m^3/hr capacity) to allow the system to be easily upgraded to the higher capacity (174 m^3/hr) at a later date. Items denoted 'A' recommended to be incorporated into the DFS design if an upgrade (to 174 m^3/hr) is possible, while equipment denoted 'B' should be incorporated if an upgrade is likely.

Table 24.7.2 Equipment List for Paste Plant Process Area

Qty	EQ description	Operating Volume / Capacity m3/h	Total Volume m3	Installed Power (total) kW	VSD	Emergency Power Requirement (kW)	Upgrade upfront if PEA Likely
1	Thickener U/F - Filter Feed Pump	150m3/h @35m TBC		37	VSD		
1	Thickener U/F - Filter Feed Pump	90m3/h @35m TBC		22	VSD		
2	Floc Dilution pump			3.0	VSD		
1	LV Electrical Switchroom						
1	Tailings Mixing Tank Agitator			37.0	DOL	37.0	
2	Tailings Filter			7.5	VSD		
2	Tailings Filter Vacuum Pump	10000 m3/hr		355.0	S/S		
2	Filtrate & Spillage Return Pump (Duty)	70m3/h @10m TBC		15	VSD		
2	Floc Dosing Pump			0.8	VSD		
2	Tailings Filter Filtrate Receiver						
2	Tailings Filter Snap Air Receiver						
2	Plant Air Compressor			110	DOL		
1	Tailings Storage/ Mixing Tank	200.00					
1	Filtrate & Spillage Return Tank	4 m3 Live					
1	Seal Water Recycle Tank	4000L					B
1	Seal Water Recycle Pump						
1	Cooling Tower	600 l/s		3.0	DOL		B
1	Filter Area Sump Pump	33m3/hr		5.5	DOL		
1	Cooling Water Circulation Pump	36 m3/hr		7.5	VSD		
1	Hoseup water Pump			0.6	DOL		
1	Filter Area Safety Shower	4.44					
1	Paste Pump*	175 m3/h @ 20 MPa		1,200.0	DOL		A
1	Gearbox Oil Cooling fan			3.0	DOL		
1	Lub/Propelling liquid pump			0.8	DOL		
1	PD Feed Pump (Duty)*	175 m3/h @15m		37.0	VSD	37.0	B
10	Cement ISO Container	20T					

The Process Area Process Flow Diagram is presented in Figure 24.5.3, where equipment that has been upsized (from the DFS design) is ballooned. An isometric view of the Process Area infrastructure is presented in Figure 24.5.4.

Figure 24.5.3 Process Flow Diagram at the Paste Plant Process Area Site

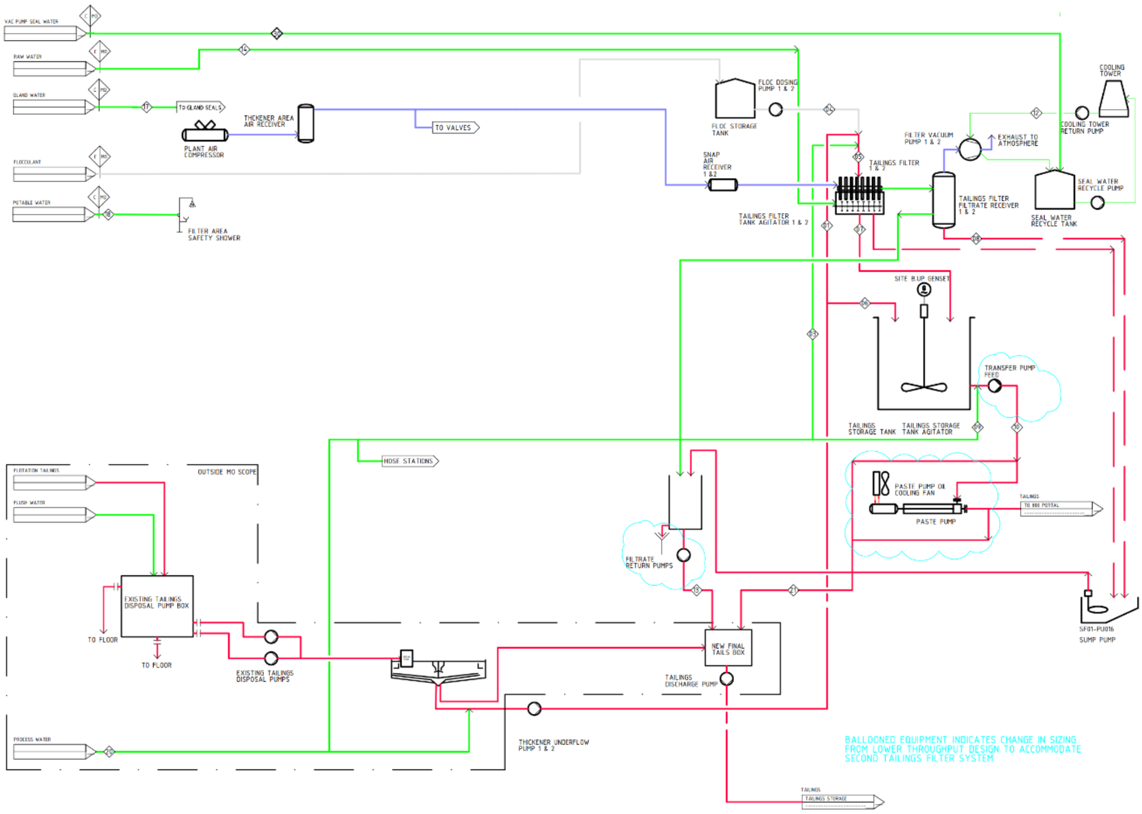
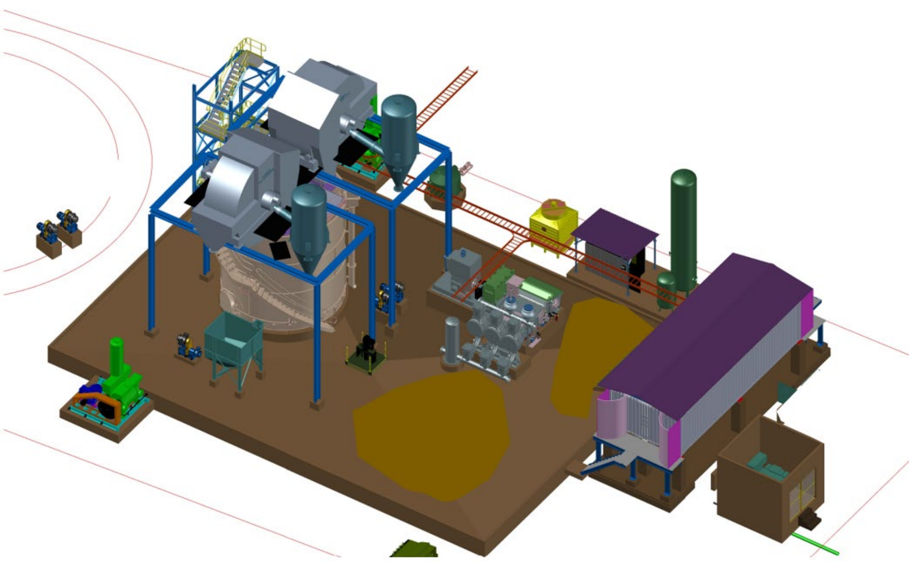


Figure 24.5.4 General Arrangement of Paste Plant Process Area Infrastructure



800 Portal Area

At the 800 Portal, site pumping infrastructure must be upgraded to accommodate the increased paste flows and a minor increase in vortex mixer capacity is also required.

To satisfy paste flows of 174 m^3/hr at the 800 Portal the following modifications to the DFS design are required:

- Increased size of vortex mixer feed pump.
- Increased size of piston diaphragm feed pump.
- Increased size of piston diaphragm pump.

To accommodate the required increase in binder and paste mixing, a slight increase in vortex mixer size is necessary.

As the binder silo size is constrained by real estate availability, it is proposed to maintain the 300t slag silo that was selected for the lower throughput DFS study. At average slag consumption rates this silo is expected to hold sufficient capacity for 2.5 days of production.

The ultimate capacity of the binder metering screw augers and metering system selected for the DFS study is 20 t/hr, which remains adequate for the increased capacity required for a production rate of 174 m^3/hr. No change is required to this component of the process.

The volume of the tailings and wastewater storage tanks at the 800 portal are dictated by the distribution pipeline volume no change to these tank sizes are required.

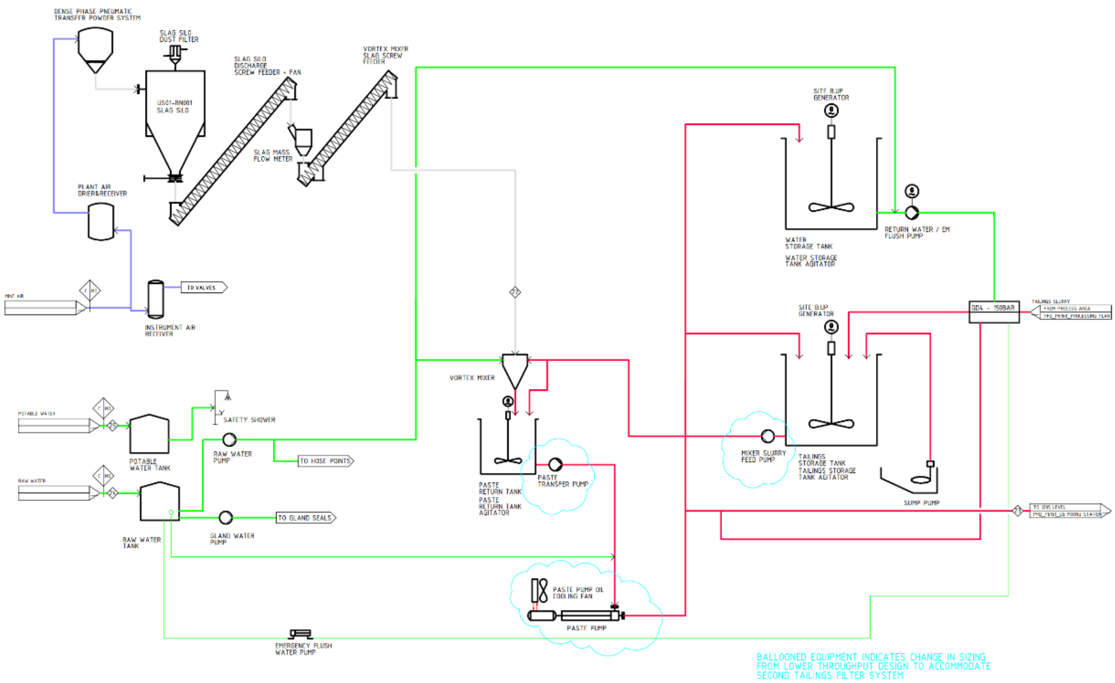
The updated equipment list for the 800 Portal Area is presented in Table 24.7.3, where equipment that is upsized (relative to the DFS design) is shaded in grey. The final column shows the equipment that should be upgraded from the DFS design (122 m^3/hr design capacity) if it is likely that the system is to be upgraded to the PEA capacity (174 m^3/hr) at a later date. Items denoted ‘A’ recommended to be adopted initially if an upgrade (to 174 m^3/hr) is possible, while those denoted ‘B’ should be adopted if an upgrade is likely.

Table 24.7.3 Equipment List for 800 Portal Area

Qty	EQ description	Operating Volume / Capacity m3/h	Total Volume m3	Installed Power (total) kW	VSD	Emergency Power Requirement (kW)	Upgrade upfront if PEA Likely
1	LV Electrical Switchroom						
1	Sump Pump			5.5			
1	Raw Water Pump	Max Flowrate - 48 m3/hr		11	VSD		
1	Potable Water Pump	Max Flowrate - 15 m3/hr		5.5	VSD		
1	Gland Water Pump	5 m3/hr		0.6	DOL		
1	Instrument Air Receiver		1.00				
1	Slag System Air Receiver		1.00				
1	Slag System Drier						
1	Binder Silo Fill Area Safety Shower	4.44					
1	Uninterruptible Power Supply						
5	Raw Water Tank		30.00				
1	Potable Water Tank		2.00				
1	Tailings Storage Tank Agitator	6.5 m x 6.5 m	200.00	18.5	DOL	18.5	
1	Flush Water Return Pump (Duty)	Max Flowrate - 100 m3/hr @ 100 m head		110.0	VSD		
1	Tailings Storage Tank	6.5 m x 6.5 m	200.00				
1	Return Water Tank	6.5 m x 6.5 m	200.00				
1	Return Water Tank Agitator	6.5 m x 6.5 m	200.00	18.5	DOL	18.5	
1	Vortex Mixer Feed PUMP	175 m3/hr @ 8m		32.0	VSD		B
1	Slag Mixing Tank Agitator	2 m x 2 m	4.00	2.2	DOL		
1	Paste Pump	175 m3/h @ 20 MPa		900.0	VSD		A
1	Gearbox Oil Cooling fan			3.0	DOL		
1	Lub/Propelling liquid pump			0.8	DOL		
1	Emergency Flush Water Pump	20.0		45.0	DOL	45.0	
1	Emergency Flush Water Pump Oil Circulation Pump			1.5	DOL	1.5	
1	PD Feed Pump	Max Flowrate - 175 m3/hr @ 15 m head		32.0	VSD		B
1	Slag Binder Silo Bulk Storage	300 m3					
1	SlagSilo Bulk Storage Unlaod POD						
1	Binder Mass Flow Meter	20 t/hr		1.1	DOL		
1	Binder Silo Discharge Screw Feeder Drive Cooling Fan			0.3	DOL		
1	Binder Silo Discharge Screw Feeder			11.0	VSD		
1	Vortex Mixer Binder Screw Feeder			11.0	VSD		
1	Binder Silo Dust Filter				-		
1	Vortex Mixer	1.5 m diameter					
1	Binder Silo Fill Area Safety Shower	4.44					
1	Shipping Container Tilter						

The proposed Process Area Process Flow Diagram is presented in Figure 24.5.5. The ballooned equipment has been upsized from the DFS design. The general arrangement of infrastructure at the 800 Portal remains effectively the same as that previously presented in Section 18.4.

Figure 24.5.5 PFD for the 800 Portal Site Component of the Paste System



1205/1170 Portal Area

At the 1205/1170 underground plant location it is necessary to increase the paste mixer size to maintain the same mixing residence time with the increased production rate. It is necessary to increase the paste pump size to accommodate the increased paste production rate of 174 m3/hr.

While cement addition rates are relatively low, at the 1205 level, to permit thorough mixing and enable consistent quality control samples at this location, the paste mixer volume is increased from 10 to 15 m^3. While this is desirable and has been incorporated into the capital budget, should the smaller (10 m^3) mixer be maintained, the outcome would be increased quality control requirements and the need for paste sampling further along the reticulation system (i.e. where additional mixing had been permitted).

To facilitate transport of the higher production rates it is necessary to upgrade the duty and standby DHC pumps.

Binder storage capacity at the 1205 is sized to permit the comfortable unloading of 20 t ISO containers - as such, it's not considered necessary to increase the capacity of this silo above the 50 t silo allocated in the DFS study. At average addition rates, this silo provides storage capacity for 25 hours, which is considered adequate.

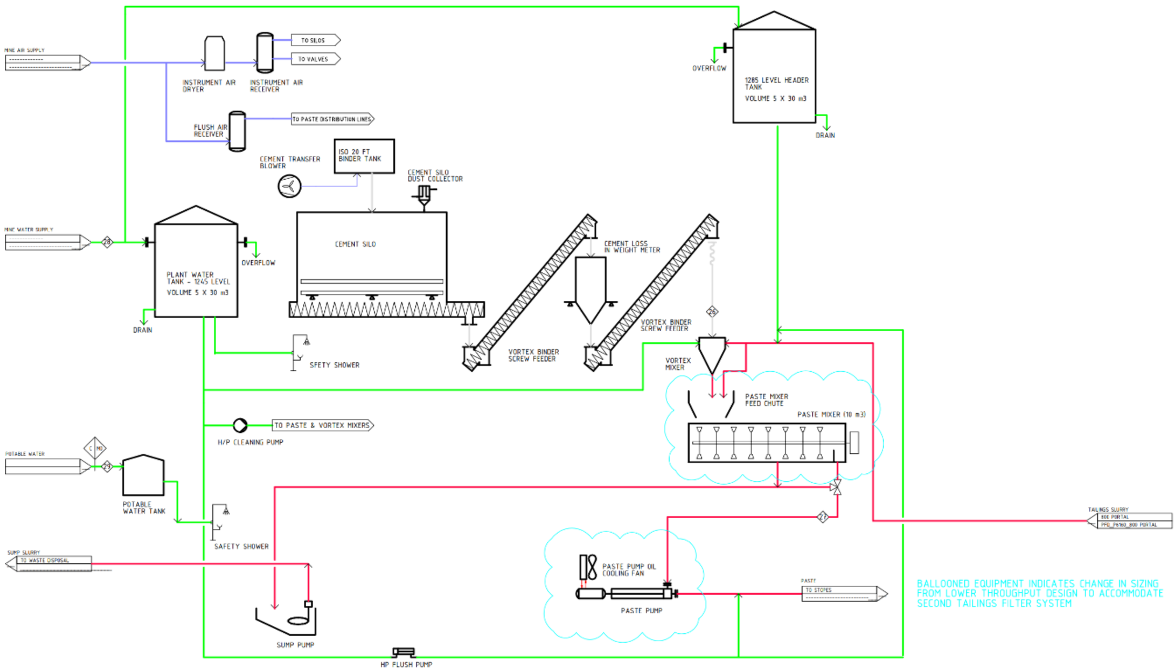
The updated equipment list for the 1205/1170 area is presented in Table 24.7.424.5.4, where equipment that is upsized (relative to the DFS design) is shaded in grey. Iitems denoted 'A' are recommended to be incorporated into the lower capacity (DFS) design if an upgrade (to 174 m3/hr) is possible, while those denoted 'B' should be incorporated if an upgrade is likely.

Table 24.7.4 Equipment List for 1205 / 1170 Levels

Qty	EQ description	Operating Volume / Capacity m3/h	Total Volume m3	Installed Power (total) kW	VSD	Emergency Power Requirement (kW)	Upgrade upfront if PEA Likely
1	Control Room / Ablution						
1	Paste Laboratory						
1	LV Electrical Switchroom						
1	Plant Air Dryer						
1	Essential Power Generator						
1	Instrument Air Receiver		1.00				
1	Binder Silo Fill Area Safety Shower	4.44					
1	Paste Mixer Eyewash	0.09					
5	1245 Lv - Raw Water Tank		30.00				
5	1285 Lv - Raw Water Tank		30.00				
1	Potable Water Tank		2.00				
1	Uninterruptible Power Supply						
1	Flush Air Reciever		14.00				
1	Paste Mixer			200.0	SS		B
1	Vortex Mixer						
1	Binder Silo Unloading Operator Panel						
1	Binder Silo		50.00				
1	Binder LIW Hopper						
1	Binder transfer Blower			75.0	DOL		
1	Vortex Mixer Binder Screw Feeder Drive Cooling Fan			0.3	DOL		
1	Binder Silo Discharge Screw Feeder			22.0	DOL		
1	Vortex Mixer Binder Screw Feeder			4.0	VSD		
1	Binder Silo Dust Filter			-	-		
1	Binder LIW / Surge Hopper Dust Filter						
1	LIW Hopper Binder Screw Feeder			4.0	DOL		
1	Paste Pump (duty)	192m3/h @ 16 MPa		1,300.0	SS		A
1	Hydraulic Booster Pump			15.00	DOL		
1	Hydraulic Auxiliary Booster Pump			11.00	DOL		
1	Oil Cooling Fan			11.00	DOL		
1	Emergency Flush Water Pump	20.0		45.0	DOL	45.0	
1	Emergency Flush Water Pump Oil Circulation Pump			1.5	DOL	1.5	
1	1205 Lv Monorail	22 m travel		5.5	DOL		
1	1170 Lv Monorail	55 m travel		5.5	DOL		

The proposed 1205/1170 Site Process Flow Diagram is presented in Figure 24.5.6. Ballooned equipment in Figure 24.5.6 is the equipment that has been upsized from the DFS design. The general arrangement of infrastructure at the 1205/1170 is effectively the same as that previously presented in Section 18.4.

Figure 24.5.6 PFD for the 1205/1170 Level Site Component of the Paste System



Hydraulic Transport (Reticulation System)

With a paste production rate of 175 m^3/hr, the line velocity in the 200NB schedule 120 pipeline is 1.8 m/s. Whilst towards the upper end of line velocities adopted for paste fill, this velocity is considered reasonable for paste transport. On this basis the pipe system, valve and instrument arrangement proposed for the DFS production rate is to be maintained for the PEA production rate.

Operating Philosophy

For all intents and purposes, the operating philosophy detailed in Section 18.4 is to be adopted for the PEA paste system design. This is inclusive of all startup-shutdown sequences as well as emergency flushing and risk management strategies.

Capital Cost Estimate

Introduction

This section provides capital cost estimates for the proposed paste backfill system to AACE Class 5 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.

Capex Estimate

The estimated capital cost is presented in Table 24.7.5. This table assumes a USD:AUD exchange rate of 1:1.32 and a USD:EURO exchange rate of 1:0.82.

Table 24.7.5 Estimated Kora Paste Backfill System Capital Cost Estimate (AACE Class 5 Est.)

Location	Component	PEA Estimate (USD)
Process Plant	Design and Management	$949,638
	Equipment	$6,471,424
	Installation	$3,370,403
	Process – 800 Portal overland piping and Instruments (inc. Supply and Install)	$3,424,171
	Commissioning	$212,614
800 Portal	Design and Management	$726,211
	Equipment	$4,284,363
	Installation	$2,898,898
	800 Portal -1205 level piping and Instruments (inc. Supply and Install)	$3,271,418
	Commissioning	$167,338
1205 Level	Design and Management	$734,684
	Equipment	$4,353,578
	Installation	$2,836,200
	1170 Level – 1St stope Reticulation and Instruments (inc. Supply and Install)	$826,164
	Commissioning	$169,290
EPC Margin (15%)		$5,204,459
Logistics (5% of equipment costs)		$1,467,607
Contingency (10%)		$3,616,400
Grand Total		**$44,987,861**

Capex Battery Limit

The battery limits for the proposed design are as described in Section 21.6.4 with flow quantities in accordance with Table 24.7.6.

Table 24.7.6 Incoming Battery Limit Inputs

Service	Units	Process Plant	800 Portal	1205 / 1170 Level
Power (installed) (kW)	kW	2430	1350	3050
Process water Consumed / Peak (m^3/hr)	m^3/hr	54 / 190	10 / 50	5 / 50
Potable water (m^3/hr)	m^3/hr	5	5	5
Compressed air @ 700 kPa	-	Not Reqd.	Not Reqd.	Mine air assumed
Flocculant (Consumption / Peak demand)	kg/hr / kg/day	7 / 250	-	-
Slag (Consumption / Peak demand)	t/hr / t/day	-	4.8 / 380	-
GP (Consumption / Peak demand)	t/hr / t/day	2.0 / 62	-	-
1205 Level Development*	m^3			20,000
1170 Level Development*	m^3			5,000
Paste Reticulation System Underground Cuddy's*	m^3			3,235

Table 24.7.7 Outgoing Battery Limits

Service	Units	Process Plant	800 Portal	1205 Level
Filtrate / waste slurry return to Tailings Thickener area	m^3/hr	82	-	-
Waste Paste on Start-up and Shutdown / Flushing	m^3/hr	246	-	-
Paste Discharge	m^3/hr	-	-	175

Capex Exclusions

- No provision has been made for ground improvement works at the 800 Portal or excavation and support of suitable underground cavities.
- Provision has been made for thrust blocks and supports for overland and underground pipe supports, but no provision is made for any overland pipe earthworks, trenching, roadway underpasses, etc. Provision is made for supply and installation of 2 pipeline dump sumps, with bog out ramp in compounds between the Process area and 800 Portal.

- No provision has been made for the supply of services outside of battery limits. i.e. for supplying the services noted in Table 24.7.6 to the edge of the relevant paste plant bund.
- Supply of all water sources to the paste plant sites including supply of gland water, vacuum pump cooling water and process water.
- Supply of HV power and transformers as well as cabling and termination LV power onto the switchroom busbar.
- Excavation and support of underground excavations is excluded from this section of the report, however, it has been included in the relevant mining section.
- Supply of mine compressed air to receivers at the 800 portal and 1170 level.
- Supply of power and communications to all nodes in the project, including pipeline instrumentation and valves.
- Permitting for construction and operation of the paste system, including acquisition of land for the Process-800 Portal pipeline corridor and dump sumps is excluded.
- Supply of concrete - it is assumed that the client would free issue 1 400 m^3 of concrete with strength of 32 MPa.
- Supply of fuel to site mobile equipment and gensets - 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.

Operating Cost Estimate

Operating Costs - Operational Fill Activities

Paste fill operating cost inputs are summarized in Table 24.7.88.

Table 24.7.8 Inputs for Paste Fill Plant Operating Costs

Item	Units	Value
Average Stope Volume	m^3	2,750
In situ void filling rate	m^3/hr	163
Annual total Stope void filled	m^3/yr	576,400
Average monthly void filled	m^3/mth	48,033
Fill plant operator costs	USD/operator/yr	10,000
Backfill Superintendent	USD/operator/yr	60,000
Backfill coordinator cost + vehicle	USD/operator/yr	35,000
Underground service crew	USD/operator/yr	10,000
Surface Light Vehicle (inc. hire, maintain, fuel)	USD/operator/yr	30,303
Surface Light Vehicle (inc. hire, maintain, fuel)	USD/operator/yr	30,303
Underground Scissor Lift (inc. hire, maintain, fuel)	USD/ yr	75,800
Surface Binder Delivery Truck	USD/ yr	37,900
Underground Binder Delivery Truck	USD/ yr	68,220
Power cost	USD/kWhr	0.17
Flocculant cost	USD/kg	3.8
GP Cement	USD/t	170
Ground Granulated Blast Furnace Slag	USD/t	327
Raw Water Cost	USD/m^3	0.1
Bulkhead Cost	USD/bulkhead	8,073

Based on past experience with similar systems and the unit rates presented in Table 24.7.88, the expected operating costs for the proposed paste system as summarized on Table 24.7.99.

Table 24.7.9 Estimated Operating Costs for the Proposed Paste System

Item	Units	Operating Cost (USD)
Fixed Operating Costs		
Plant Operating Labour (3 Surface Operators, 1 UG Operator, 1 Surface binder delivery driver, 1 UG binder delivery driver x 3 swings)	/yr	$180,000
Fill Superintendent (2 total)	/yr	$120,000
Fill Coordinator (3 total)	/yr	$105,000
Underground Labour (5 Crew x 3 swings)	/yr	$150,000
Surface LV (1 vehicles)	/yr	$30,303
Underground Vehicle (2 vehicles)	/yr	$60,606
Underground Scissor Lift (or Integrated tool carrier)	/yr	$75,758
Binder Delivery Vehicles (1 x Surface 1 x UG)	/yr	$127,273
Management Training, Consulting Services	/yr	$210,000
Total Annual Fixed Costs (USD)		**$1,058,939**
Variable Costs		
Process Plant		
General Consumables (Thickener, Agitator tank, Centrifugal pumps, Auxiliaries)	/m^3	0.15
Plant repairs and Maintenance (Parts and Labour)	/m^3	0.19
Filter Maintenance (Parts and Labour)	/m^3	0.25
Paste Pump Maintenance (Parts and Labour)	/m^3	0.21
Power (1150 kW consumed)	/m^3	1.20
Flocculant (40 g/t avg)	/m^3	0.19
Raw Water	/m^3	0.02
800 Portal		
General Consumables (Agitator tank, Centrifugal pumps, Slag System, Auxiliaries)	/m^3	0.11
Plant repairs and Maintenance (Parts and Labour)	/m^3	0.11
Paste Pump Maintenance (Parts and Labour)	/m^3	0.21
Power (1042 kW)	/m^3	1.09
Slag (2.2% Slag addition, w/w)	/m^3	9.00
Slag Delivery	/m^3	0.08
Raw Water	/m^3	0.00
205 Level		
General Consumables (Mixer, Centrifugal pumps, GP System, Auxiliaries)	/m^3	0.15
Plant repairs and Maintenance (Parts and Labour)	/m^3	0.23
Paste Pump Maintenance (Parts and Labour)	/m^3	0.37
Power (588 kW)	/m^3	0.61
GP Cement (1.0% GP addition, w/w)	/m^3	2.15
GP Delivery	/m^3	0.14
Raw Water	/m^3	0.00
Underground Construction Materials (labour & Equipment inc. in Fixed Operating Costs)		
Underground Reticulation Materials ($140,000/level)	/m^3	1.70
Bulkheads (1 per stope)	/m^3	2.93
Overheads		
Quality Control	/m^3	0.08
Total Variable Costs	**/m^3**	**21.2**
Fixed + Variable Costs		$23.0
Contingency (5%)		$1.1/m^3
Total Estimated Operating Cost (USD)		**$24.1/m^3**

24.8 Kora 2022 PEA Economic Analysis

24.8.1 Introduction

An economic analysis has been carried out for the expanded 1.7 Mtpa project using a cash flow model, similar to that carried out for the 1.2 Mtpa project.

24.8.2 Model Inputs and Assumptions

The key model inputs used in the economic analysis are summarized in Table 24.8.1.

Table 24.8.1 PEA Key Model Inputs

Model Inputs	Source	Unit / Value
Base Currency		USD
Base Date		3rd Quarter 2022
Ore Processed over LOM	Entech	12.16 Mt
Metal Prices		
Gold	K92	USD1,600 / oz (fixed)
Copper	K92	USD4.00 / lb (fixed)
Silver	K92	USD20 / oz (fixed)
Recoveries		
Gold (total)	MMS	93.0%
Gold (gravity to dore)	MMS	15.0%
Copper	MMS	95.2%
Silver	MMS	80.0%
Concentrate copper grade	MMS	15.5%
Processing Plant Capacity (dry tonnes of ore)		
Existing Plant	K92	500,000 tpa
New Plant	Lycopodium	1,200,000 tpa
Royalties (deducted from gross revenue)	K92	2.0%
Levies (deducted from gross revenue)	K92	0.5%
Tax Rate	K92	30%
Depreciation		Not considered
NPV Discount Rate	K92	5%

Ore Processed

Total ore processed from 2022 to 2032 is 12.16 Mt.

Average head grade over LOM is:

- Gold 7.03 g/t
- Silver 22.52 g/t
- Copper 1.18%

Gross Revenue

Gross revenue from doré and concentrate (excluding treatment and refining charges and operating costs) is USD5,436M over the LOM, or USD447/t ore.

Treatment and Refining Charges

Total treatment and refining charges, including penalty element deductions equates to USD53/t ore.

Operating Costs

Annual fixed and variable costs are included in the cash flow and summarized in Table 24.8.2.

Table 24.8.2 PEA Operating Costs

Area	**Source**	**Unit / Value**
Mining (average over LOM)	Entec	USD64 / t ore
Processing Plant – Existing - Total (average over LOM)	K92	USD23 / t ore
Processing Plant – New - Fixed	Lycopodium	USD8.8M /year
Processing Plant – New - Variable	Lycopodium	USD7.61 / t ore
Processing Plant – New - Total (average over LOM)		USD15 / t ore
General & Admin - Total (average over LOM)		USD24 / t ore
Paste Plant - Fixed	Helinski	USD1.1M /year
Past Plant - Variable	Helinski	USD22.22 / m^3 paste
Paste Plant - Total (average over LOM)		USD24 / m^3 paste
TSF (average over LOM)	ATC Williams	USD0.71 / t tails
Transport & Insurance (average over LOM)	K92	USD3.99 / t ore
Total (average over LOM)		**USD118 / t ore**

Capital Costs

Pre-production capital expenditures are defined in Table 24.8.3. Sustaining capital for the Plant, Mining and TSF expansion costs have been phased over the life of the project and detailed in Table 24.8.4.

Table 24.8.3 PEA Pre-Production Capital Expenditure

Item	Source	Total (US$'000)
Mine		Refer sustaining capital
Process Plant	Lycopodium	80,195
Power Station	Lycopodium	12,467
Power Supply	Lycopodium	13,815
Camp Upgrade	K92	7,285
Haul Road Bridges	K92	3,740
EPCM	Lycopodium	20,423
Owner's Costs	K92 / Lycopodium	4,200
Paste Plant	Helinski	44,985
Total		**187,105**

Table 24.8.4 PEA Sustaining Capital Expenditure

Item	Source	Total (US$'000)
Mine	Entech	383,457
TSF	ATC Williams	45,494
TSF Closure Costs	ATC Williams	10,800
Total		**439,750**

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 3Q22 values.

24.8.3 Financial Model Results

The financial model indicates that the PEA project has a post-tax Net Present Value (NPV) of USD1,325M 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 24.8.1 shows the post–tax annual and cumulative cash flow for the project over the LOM.

As the model starts with a positive cash flow in 2022 due to revenue from the existing plant operation, the cumulative cash flow position is never negative.

Due to this initial positive cash flow, an Internal Rate of Return (IRR) is not published.

Figure 24.8.1 PEA Cumulative Cash Flow

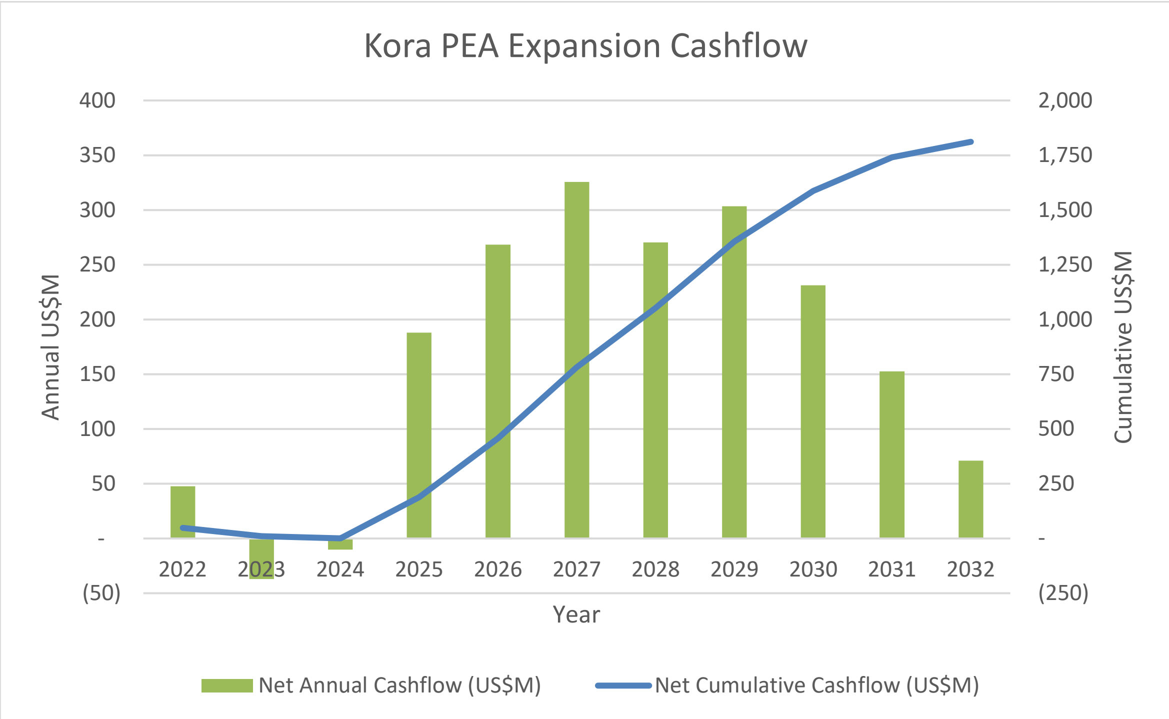


Table 24.8.5 shows a summary of the cash flow model for the project.

Table 24.8.5 PEA Cash Flow Model Summary

	Source	Units	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	TOTAL
Period Ending			31/12/2022	31/12/2023	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	
Gross Revenue - After Treatment & Refining Costs		**US$ 000's**	**209,588**	**209,715**	**318,538**	**450,666**	**604,413**	**717,078**	**619,371**	**652,665**	**514,785**	**334,096**	**156,355**	**4,787,271**
Total Operating Costs		**US$ 000's**	**76,662**	**89,822**	**122,136**	**141,550**	**163,293**	**183,970**	**187,301**	**179,964**	**154,311**	**101,460**	**36,997**	**1,437,466**
Total Royalties & Levies		**US$ 000's**	**5,240**	**5,243**	**7,963**	**11,267**	**15,110**	**17,927**	**15,484**	**16,317**	**12,870**	**8,352**	**3,909**	**119,682**
Total Capital Costs		**US$ 000's**	**45,920**	**131,375**	**169,043**	**51,171**	**61,835**	**61,789**	**38,050**	**31,586**	**17,331**	**6,068**	**13,788**	**627,956**
FINANCIAL SUMMARY														
Total Revenue		US$ 000's	209,588	209,715	318,538	450,666	604,413	717,078	619,371	652,665	514,785	334,096	156,355	4,787,271
Total Outgoings		US$ 000's	127,822	226,439	299,142	203,988	240,239	263,686	240,836	227,867	184,511	115,880	54,693	2,185,103
Earnings before Interest, Tax, Depreciation and Amortisation (EBITDA)		**US$ 000's**	**81,766**	**-16,724**	**19,396**	**246,678**	**364,175**	**453,392**	**378,535**	**424,798**	**330,273**	**218,216**	**101,662**	**2,602,168**
Depreciation Allowance ignored		US$ 000's												0
Taxable Profit / Loss		US$ 000's	81,766	(16,724)	19,396	246,678	364,175	453,392	378,535	424,798	330,273	218,216	101,662	2,602,168
Tax Payable		US$ 000's	33,962	20,198	29,656	58,642	95,896	127,866	108,112	121,521	99,082	65,465	30,499	790,899
Net Profit After Tax		US$ 000's	47,804	(36,922)	(10,260)	188,037	268,278	325,526	270,423	303,278	231,191	152,751	71,163	1,811,269
After Tax Operating Cash Flow		US$ 000's	47,804	(36,922)	(10,260)	188,037	268,278	325,526	270,423	303,278	231,191	152,751	71,163	1,811,269
Cash Flow														
Cash Flow - After Tax		000's US$	47,804	(36,922)	(10,260)	188,037	268,278	325,526	270,423	303,278	231,191	152,751	71,163	1,811,269
Accumulated Cash Flow		000's US$	47,804	10,882	622	188,658	456,937	782,462	1,052,885	1,356,163	1,587,354	1,740,105	1,811,269	
Payback Period - Simple		years			-	-	-	-	-	-	-	-	-	0.00
Discounted Cash Flow - Mid Year Discounting														
Discounted Cash Flow @ 5%	K92 Calc	US$ 000's	46,652	(34,316)	(9,082)	158,518	215,394	248,911	196,930	210,339	152,708	96,092	42,635	1,324,782
Discounted Cash Flow @ 8%	K92 Calc	US$ 000's	45,999	(32,897)	(8,464)	143,635	189,749	213,184	163,979	170,279	120,190	73,529	31,718	1,110,902
Discounted Cash Flow @ 10%	K92 Calc	US$ 000's	45,579	(32,003)	(8,085)	134,700	174,710	192,719	145,543	148,387	102,833	61,767	26,160	992,310

Discount Rate		%	**5.0%**	**8.0%**	**10.0%**
Net Present Value (NPV) - After Tax		000's US$	**1,324,782**	**1,110,902**	**992,310**

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Mining and Mineral Reserve Estimate

The FS 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.

25.2 New 1.2 Mtpa Treatment Plant

Design criteria, process flow sheets and a mass balance for a 1.2 Mtpa Process Plant treating material from a copper-gold sulphide deposit have been developed. Capital Cost Estimates (Capex) and Operating Cost Estimates (Opex) were prepared for the Processing Plant; Secondary Back-up Power Station and power reticulation.

The total installed capital estimate for the 1.2 Mtpa Processing Plant is estimated to be US$80.2M including a contingency allowance. A new standalone power station for the 1.2 Mtpa Processing Plant is estimated to cost US$9.6M inclusive of contingency and a further US$13.8M is included for power supply inclusive of contingency.

Conventional single stage crushing followed by a traditional SAB milling circuit was chosen in place of the current multistage crushing and ball mill circuit based on the test work and comminution modelling conducted during the study. The milling circuit includes flash flotation and a gravity circuit to capture free gold for smelting on site to produce gold doré.

Conventional sulphide flotation, thickening and filtering is employed to produce a high-grade concentrate which is loaded into shipping containers for transport to smelters.

The overall project schedule from project go-ahead until the first gold pour and project handover is scheduled for 23 months. An implementation schedule was prepared to support the duration of 23 months and is based on a critical path through the supply and installation of the mills.

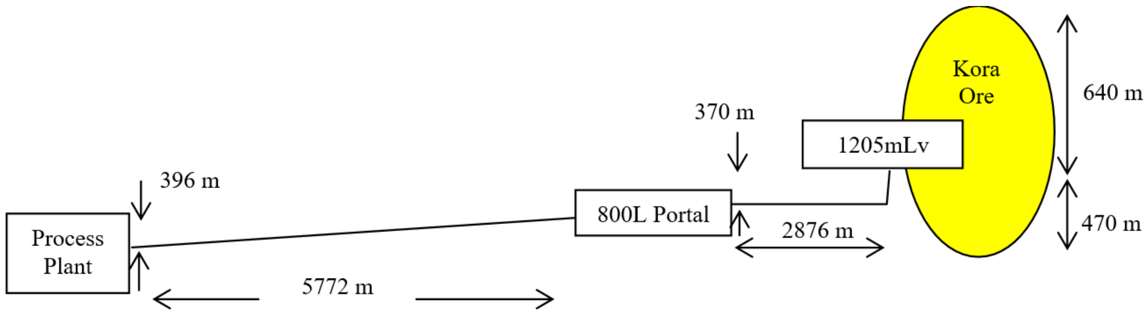
25.3 Paste Fill Plant

This document sets out the findings of a mine backfill Detailed Feasibility Study (DFS) for K92's Kainantu mine.

Within the Kainantu mine complex, K92 are investigating options for mining the 8.2 Mt, high-grade narrow vein Kora orebody. Kora is to be mined using a combination of bottom–up and top-down longitudinal retreat stoping. Kora ore is to be processed at the Kainantu processing plant, which is located approximately 6 km from the portal into the Kora mine. To improve geotechnical stability, facilitate complete ore extraction and realise the environmental and economic benefits of reduced surface tailings disposal, it is proposed to adopt cemented paste backfill as part of the mining process. Cemented paste backfill is to be manufactured using fresh tailings from the Kainantu process plant.

To illustrate the overall site geometry Figure 25.3.1 presents a schematic long section showing the location of the Kainantu process plant, the mine portal (termed 800 Portal) and the Kora orebody.

Figure 25.3.1 Schematic Long Section of the Kainantu Mining Complex and Kore Orebody



A mine production rate of 1.2 Mtpa is to be targeted for the Kora mine. At this time only Kora ore is to be fed into the Kainantu process plant and with this feed rate a tailings production rate of 141 tph is expected. The Kora mining schedule shows a maximum monthly backfill demand of 45,000 m^3 with an in situ dry density of 1.25 t/m^3 if the full available tailings stream was to be utilised. A peak monthly backfill system utilisation of 53% would be required to satisfy this demand.

Kora paste is expected to be transported at an equivalent dry density of 1.15 t/m^3 before shrinking to an in situ dry density of 1.23 t/m^3, meaning that every cubic meter filled underground would require 1.07 m^3 of paste. On this basis, the 141 t/h of available tailings would generate 122 m^3/h of wet paste that must be accounted for in the process design.

A range of different backfill systems were considered and baselined with the following outcome:

- Thickener underflow tailings are to be drawn from the tailings thickener boot, where one stream is directed to a filter and a second is directed into a 200 m^3 mixing tank.

- In the mixing tank the filter cake is combined with thickener underflow to create an uncemented paste with yield stress of approximately 50 Pa. This yield stress was selected because this allows the paste to be hydraulically transported to the 800 Portal (at relatively low friction loss levels), while ensuring that the paste is non-segregating.

- This material is transported hydraulically, using a piston diaphragm pump, some 6 km laterally, and 400 m vertically, to the 800 Portal.

- At the 800 Portal site, paste is placed into a 200 m^3 paste storage tank.

- A 300 m^3 binder silo is located at the 800 Portal, where ground granulated blast furnace slag (Slag) stored. This slag is delivered in 20 ft shipping containers, fitted with consumable bladders, and a sucker / blower is used to empty the containers.

- Uncemented paste, drawn from the paste storage tank, is combined with slag in a small mixing tank before being fed into a second piston diaphragm pump for transport to the 1205 underground mixing station. For the following reasons the optimal process requires slag to be added at the 800 Portal:

  - Binder optimisation studies (Metso Outotec (2021) showed with binder blends with high portion of slag to be optimal. Therefore, if slag is added at the 800 Portal a significant portion of the binder is not required to be hauled underground. This presents the benefit of reducing transport costs and less underground mine traffic congestion. Furthermore, unloading of shipping containers (used to deliver slag to site) requires the container to be tilted. Tilting of containers for unloading requires significant height, which is not considered practical underground.

  - Slag addition only (without GP) does not change the rheological behaviour of uncemented paste, meaning that this material can be added to the paste without making hydraulic transport more difficult.

  - At a pH less than 10 (which is expected to be typical for slag-paste) slag forms a 'protective coating', preventing hydration and any cementation of the slag-paste in the reticulation lines. This reduces the risk of blockages, relative to cemented paste.

- From the 800 Portal, Slag-Paste is hydraulically transported 2,876 m laterally along the Kora 'twin declines' and then vertically 331 m to the 1,205 m Level, where the underground mixing and pumping system are located.

- At the 1,205 Level Slag-Paste is fed directly into the paste mixer, where it's combined with GP cement. Addition of nominally 1% GP cement increases the paste pH, which increases the yield stress to 200-300 Pa (i.e., that of a typical paste fill) and liberates hydration of the slag product.

- GP binder is transported directly from Lae to site and then onto underground in 20 t ISO containers. At the 1,205 Level these ISO containers are pneumatically emptied into a 50 t underground horizontal silo, which meters binder (via a metering system) into the paste mixer.

- From the mixer, cemented paste backfill feeds under gravity to the 1,170 Level and directly into a hydraulic piston pump.

- The hydraulic piston pump is used to distribute paste to stopes throughout the Kora orebody.

Given the relatively low density of Kora paste and the short (20 m) sub-level intervals it is proposed to construct bulkheads of adequate capacity to withstand the full geostatic pressure from a full stope of liquefied paste. Provided an exclusion zone of approximately 180 m is allowed for in mine scheduling, will permit continuous filling.

Even with continuous filling, due to the relatively small volume of typical Kora stopes (3,000 m^3) filling is expected to be complete in approximately one day and flushing of the entire paste transport network (from Process Area to Stope) is not considered practical. However, given the fine-grained nature of the Kainantu tailings and the low paste yield stress, uncemented paste is to be maintained static in the pipework (between the Process area at the 1,205 level) for a limited time period (of approximately four hours) when changing between stopes.

A detailed process description can be found in Section 13.14. Isometric drawings of the paste plant arrangements at the process area, 800 Portal Area and 1,170/1,205 Level Area are presented in Figure 25.3.2, Figure 25.3.3 and Figure 25.3.4, respectively.

Figure 25.3.2 Process Layout at the Process Area

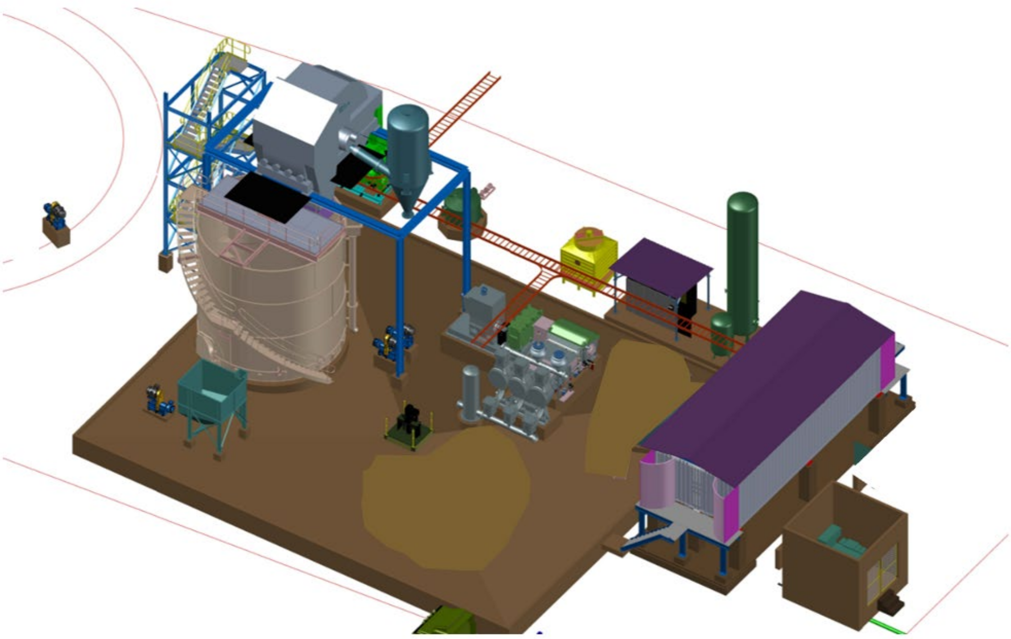


Figure 25.3.3 Process Layout at the 800 Portal Area

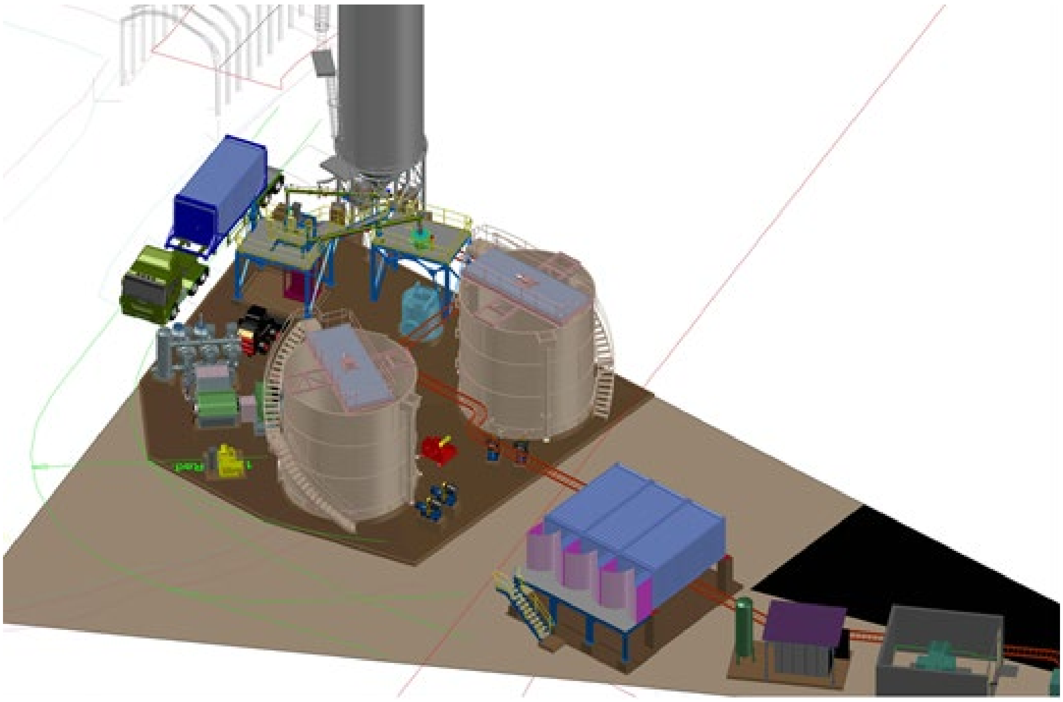
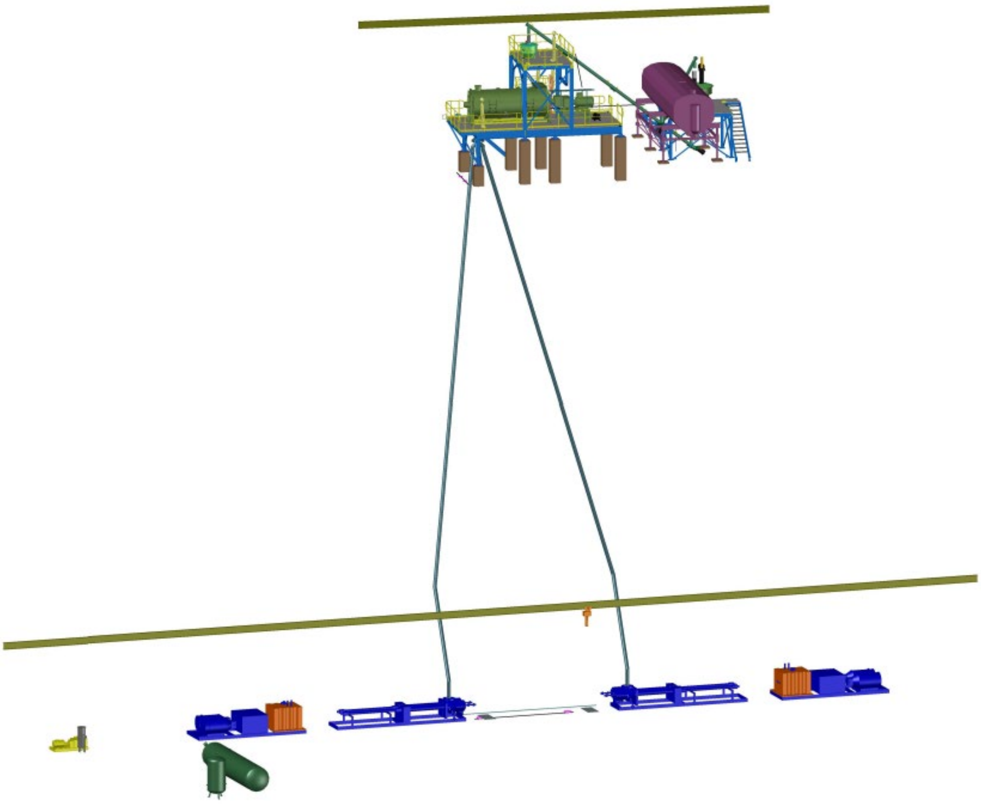


Figure 25.3.4 Process Layout at the 1170/1205 Area



Overland piping from the process plant to the 800 Portal is approximately 6,000 m and this is to be 200NB Schedule 120 steel pipe. To provide security in publicly accessible areas, connections are to be welded. However, to accommodate thermal expansion and contractions provision is made for expansion loops and a limited number of flexible couplings (in secure compounds). Provision has also been made for two dump sumps (within these compounds) to allow 'last resort' cleaning of the pipeline if necessary.

Piping between 800 Portal and the 1,205 Level is to run a distance of 2,876 m through the twin portals before ascending approximately 330 m through a borehole to the 1,205 Level. Pipework in the twin portals is to be 200NB schedule 120 pipe with 809N weld ring couplings, while the internal borehole is to be lined with threaded 200NB schedule 120 steel pipe. Provision has been made for 1.5 m length of casing to permit drilling and installation using a raise drill rig.

Downstream of the 1,170 Level pump station (i.e., to stopes), piping is to be 200NB Schedule 80 with 809N weld ring couplings. The DFS capital budget makes provision for the supply and installation of 350 m of horizontal piping and 60 m of vertical drilling and casing. Pipework and installation costs beyond this are included in operating costs.

For all paste placed underground a GP cement binder addition of 1.0% is required, with the slag content varying between 2 to 9%, with 98% of fill placed having a slag content less than 3% (i.e., total binder content less than 4%).

For operation of the system, it is proposed to have a total of 12 personnel onsite at any given time. These personnel are to be distributed according to the following:

- 4 operators, consisting of a single operator at each of the three site locations as well as an additional operator (each on 3 swing rosters, i.e., 12 total).
- 2 operators dedicated to haulage and unloading of slag and GP cement (each on three swing rosters, i.e., 6 total).
- 5 underground operators (each on 3 swing rosters, i.e., 15 total).
- 1.5 fill coordinators.

Based on a detailed analysis of the required equipment and plant arrangements the Capital and Operating cost for the system was determined to AACE Class 3 accuracy. The total Capital and Operating cost for the proposed system is summarised in Table 25.3.1.

Table 25.3.1 Kora Paste Cost Estimate (AACE Class 3)

Item		USD
Capital Cost*		\$39.9M
Operating Cost	Fixed Opex	\$1.0M/y
	Variable Opex	\$21.8/m^3
Combined Opex (at 428 Km^3/y)		\$25.4/m^3

*** The capital cost associated with the required underground development is included in the mining costs component of the study.**

A preliminary project implementation schedule shows a total project implementation period of approximately 21 months from placement of order and a peak of 69 people onsite during construction.

Of the operating costs, binder is expected to constitute approximately 42%; labour, 22%; power, 13%; and underground consumables (pipe and bulkheads) contributing 14%.

At the time of compiling this report K92 were investigating options to streamline delivery of slag to site. With a more efficient slag delivery strategy it is expected that the unit price of slag could reduce to a value similar to the current GP cement cost (\$170 USD/m^3). If this was to be the case the paste operating cost would reduce to \$21.0USD/m^3.

Given that binder, labour, and bulkhead costs represent a significant portion of the operating costs, should K92 utilise the fill system to dispose of tailings into (large) historic mined voids the unit operating costs could be reduced significantly. Assuming fixed costs are absorbed by fill placed into Kora, any additional fill placed into historic voids could be deposited at an operating cost of \$16.0 USD/m^3. Following on from the previous paragraph, should slag be supplied at a unit price equal to that of GP cement, this cost would reduce to \$14.3 USD/m^3.

To satisfy the system utilisation requirements it is considered necessary to fill Kora stopes in a single pass. Due to the significant consequences of a bulkhead failure resulting in a large paste inrush, for a typical 3,000 m^3, its necessary to implement a bunded exclusion zone that is nominally 170 m beyond the paste bulkhead. Consideration should be given to this exclusion requirement when scheduling Kora mining activities.

The most significant risk to the feasibility of this project is associated with the bearing capacity for the paste plant infrastructure at the 800 Portal site. It is highly recommended that K92 conduct a thorough geotechnical investigation and design of this area to ensure that the required bearing capacity can be achieved.

At the time of completing this study, K92 were also considering the possibility of adopting a mine production rate of 1.7 Mtpa for their Kainantu operation (compared with the 1.2 Mtpa production rate considered in the DFS). To service this increased production rate, it is expected that the paste production rate must be increased to 174 m^3/h. A paste system with this capacity can be implemented at Kainantu for an estimated cost of \$45.0M and Operating Cost of \$24.1/m^3 (AACE Class 5 accuracy).

Considering the urgency to implement a paste system at Kainantu, and long lead time for major equipment, it is understood to be important to commence paste system implementation as quickly as possible. To assist with defining a way forward, scalability of the 122 m^3/hr system, to a 174 m^3/h system was considered. This assessment showed that:

1. If it is possible that the 122 m^3/h paste system (for 1.2 Mtpa mine production) may be scaled to 174 m^3/h (for 1.7 Mtpa mine production) all 4 positive displacement paste pumps should be upgraded initially at an upgrade cost of approximately $0.8M.

2. If the 122 m^3/h system is likely to be upgraded to 174 m^3/h then the centrifugal pumps and paste mixer should be upgraded, in addition to the paste pumps, at a combined cost of $1.6M.

The second upgrade option is expected to put K92 in a position that the system production capacity could be upgraded through the addition of only equipment that must be duplicated (i.e., no equipment replacement upgrades).

25.4 Tailings Storage Facility

Tailings generated from the production expansion will be stored in the existing tailings storage facility (TSF) and 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.

25.5 Risk Assessment

25.5.1 Risk Assessment Objectives

A risk assessment was undertaken to investigate potential risks associated with the Kora Expansion Project, as well as risks associated with permitting and approvals. The risk assessment included the identification and assessment of the key risks according to their likelihood of occurrence and consequence. Based on this assessment, the risks were categorised into one of four risk classes (low, moderate, high, and extreme). In principle management and mitigation measures were developed to limit and manage the risks.

This report includes only the significant risks identified as part of the assessment.

25.5.2 Risk Rating Matrix

Table 25.5.1 presents the significant risks identified in the assessment.

Table 25.5.1 Risk Rating Matrix

Risk Title	Cause	Likelihood	Consequence	Overall Class Risk	Controls
Supply Chain holdups	Multiple 3PL (3rd party logistics) Triage (incorrect priority)	Rare	Moderate	High	Engage 3rd party larger logistics company with placement of K92 on priority communication and expediting. Onsite experience Lae ports & expediting officer
Domestic Port Constraints	Inability to handle increased container traffic Delays handling oversized Inability to handle bulk binder	Moderate	Possible	High	Initial packaging and constraining requirement issued to suppliers Port assessment and infrastructure improvement design
Ground not suitable for Mill foundation	Incorrect or lack of Geotech information	Moderate	Possible	High	Test pits done and comparison to existing plant drawn Ground investigation for all major infrastructure construction - Geotech consultant to be engaged by K92. Interpretative Geotech report to be requested as a deliverable
Paste Plant foundation not suitable	Bearing capacity issues (cost increase due to need for piling foundation)	Moderate	Possible	High	800 Portal designed to minimise high bearing capacity equipment. High bearing capacity equipment located near natural surface (on the highwall) Ground investigation for all major infrastructure construction - Geotech consultant to be engaged by K92. Interpretative Geotech report to be requested as a deliverable
TSF Construction	Permits not received or incorrect Incorrect management / handling	Catastrophic	Unlikely	High	Early planning application

Risk Title	Cause	Likelihood	Consequence	Overall Class Risk	Controls
Excessive Stope overbreak resulting in excessive dilution	FGZ proximity, inappropriate stope size	Major	Possible	Extreme	Stope design based on geotechnical data using industry accepted design methods. Effective filling of stope voids with Paste fill. Ensure adequate stand-off to FGZ Improve FGZ management and geotechnical controls
Mine Production Schedule Slippage	Long-lead mobile fleet delivery	Moderate	Possible	High	LOM plan & long lead item procurement schedule
Variability in mineral resource estimated grade	Change in orebody properties/characteristics (particularly at the orebody periphery). Under preforming diamond drilling (Both quality & quantity)/No access to drilling site. Core logging errors & poor productivity PEA has inferred material which has lower uncertainty	Major	Unlikely	High	Diamond drilling at the required grid spacing for geological grade continuity Geology procedures and process established
Variability in mineral resource estimated tonnes	Under-drilled areas have a high chance of tonnes variability PEA has inferred material with has lower uncertainty	Major	Unlikely	High	Diamond drilling at the required grid spacing for geological grade continuity Geology procedures and process established
Landownership disputes	Existing land court matter impact on expansion project construction.	Expected	Significant	High	Input from local K92community relations team regarding sensitivity of land court matter. Implement Stakeholder Engagement Process for the expansion project.

25.5.3 Risk Assessment Findings

A total of 142 risks have been identified during the studies formal risk assessment with 48 being High Risk and 1 being Extreme Risk. There have been 10 risks that have been identified as requiring a higher level of management/control to ensure the risk ranking is maintained (see Table 25.5.1):

- Supply chain holdups.
- Domestic Port constraints.
- Mill foundations.
- Paste Plant 800 foundations.
- TSF permitting delays.
- Excessive stope overbreak.
- Mobile fleet lead-time.
- Variability in mineral resource grade.
- Variability in mineral resource tonnes.
- Landownership disputes.

26.0 RECOMMENDATIONS

26.1 Mine

Items for consideration are detailed in the following list:

- Optimization studies to increase NPV through scheduling and design improvements.
- Optimization study to consider ventilation design alternatives for surface raisebore versus underground adits to surface.
- Optimization of ore pass locations, and potential use of truck chutes at their termination points as opposed to rehandle with a loader.
- Optimization study to consider viability of underground and/or land conveyor suitability.
- 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 mineralized zones Judd, K1 and K2 and the FGZ.
- Additional geotechnical testwork is required for all lithologies to confirm FS 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.
- Mapping of occurrences of any water ingress and testing of water for corrosion potential.
- 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.

26.2 1.2 Mtpa Treatment Plant

Following approval to proceed to the next phase of the project, it is recommended Geotech drilling and investigations are completed on the process plant location to confirm and optimize the foundation design parameters for the detail design.

26.3 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 Stage 3 (RL 530 m) 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.

26.4 Mineral Resource Estimates

26.4.1 QAQC

K92ML needs to review its procedure for deciding on the failure of a standard assay. Consideration needs to be given to consecutive standard assay failures within single batches and what threshold is used to trigger an enquiry for clarification with the laboratory.

Documentation of the QAQC data needs to be improved so that it is much more easily assessed when receiving new assay results and for any subsequent resource estimation. Preferably a report is completed immediately at the end of any drilling campaign so that it is already available for review prior to any resource estimation work. This acknowledges that the outcomes of the QAQC programme are significant factors in the classification of the resource estimates. It also allows for any issues to be addressed prior to starting resource estimation work.

A continuation of the laboratory duplicate sampling has been recommended, particularly focussing on increasing the number of higher grade samples that are tested. This may require employing a more manual role in sample selection for higher grades.

26.4.2 Mineral Resource Estimates

Continue infill drilling to upgrade both the Kora Consolidated and Judd Mineral Resources.

Drilling to the south of both Judd and Kora, following up significant isolated drill intercepts, with a view to expanding the current Mineral Resources.

Continue reconnoitre drilling of other potential mineral lodes/systems identified from surface mapping and sampling within the general Kora Consolidated and Judd lode areas.

New drilling should assess areas of geological complexity; e.g. the possible ‘doubled up oblique fault section’ in the middle of the deposit.

26.4.3 Exploration

The entire mineralised district held under tenement should be assessed but with priority given to certain areas.

The highest priority is to test (and extend) the Kora trend towards A1. This will facilitate the expansion of the current inferred Kora and Judd resource. Much of the vein system towards the south is under shallow cover (colluvium) and will require drill testing owing to the paucity of outcrop.

The Kora Consolidated lode(s) is demonstrably rich in copper towards the south, thus potentially being closer to the original fluid source. This corresponds with an area of extensive dilation, where high grade Au/Cu veins are haloed by long intervals of moderate grade. This corridor, linking Kora Consolidated and Judd with the A1 porphyry target, should be a priority in the ongoing exploration program.

Maniape and Arakompa both have historic resources which makes them 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 that 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.

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. Fortunately, much of the K92ML tenement package is accessible by road, at least to within walking distance of the prospects. This will enable teams to assess lithocaps and vein expressions, as well as geophysical anomalies, by systematic exploration, while advanced drill programs are carried out concurrently.

26.5 Environmental Studies, Permitting and Social or Community Impact

26.5.1 Forward Works Plan

This section outlines the recommended forward works plan for environment and community studies to support the permitting of the Kora Expansion Project, i.e. the preparation, submission and assessment of the EIS. This plan will also contribute to the management of the risks identified in Section 25.4.3 This is intended to provide K92 with a guide to the scope of investigations and activities required, the schedule for this, and an indicative cost estimate for doing so.

Scope of Activities to be Completed

A desktop review and gap analysis of existing environment and social data was conducted to inform the scope for the EIS for the Kora Expansion Project. Additional environment and social data collection were identified as part of this review. The studies will build on the information already available from previous environmental and social baseline studies and monitoring campaigns and previous approval documentation, plus other similar projects in PNG.

Table 26.5.1 and Table 26.5.2 list the objectives for the proposed long-term and short-term biophysical and socio-economic studies, respectively, for input to the EIS for the expansion project.

In addition to the technical studies outlined in Table 26.5.1 and Table 26.5.2, the following key tasks are required during preparation of the EIS:

- Client and study management and reporting.
- Interaction with the Definitive Feasibility Study engineering team to source the engineering information for the EIS.
- Preparation of the EIS and Environmental Management and Monitoring Program documents.
- Local stakeholder engagement.
- Regulator engagement.

Table 26.5.1 Proposed Biophysical Studies for Input to the EIS

Main Technical Study	Timeframe	Objective
Surface water (hydrology)	Long-term	Characterize stream flow in the existing mine area and downstream. Estimate the concentrations of contaminants in downstream waterbodies due to the expansion project. Assess cumulative water quality impacts (along with existing impacts)
Geochemical and physical characterization of mine waste materials	Long-term	Characterize chemical and toxicological properties of tailing and waste rock to be produced from the Kora deposit. Determine any changes to existing selective handling requirements and management measures for waste rock. Inform modelling for waste rock storage and paste backfill of tailings.
Groundwater	Short-term	Characterize the groundwater regime (i.e. groundwater level and quality data) to inform hydrogeological model for mine expansion. Assess impacts on groundwater beneficial uses and values due to construction, operation, and closure of the expansion project.
Aquatic ecology	Short-term	Characterize aquatic fauna (including selected invertebrates), flora and habitats, particularly in areas likely to be affected by the expansion Determine the expansion project impacts on aquatic habitats and biota Predict the likely impacts to beneficial values associated with the freshwater environment.
Terrestrial ecology	Short-term	Characterize areas of native vegetation or habitat (if any) potentially impacted by the project expansion

Table 26.5.2 Proposed Socio-Economic Studies for Input to the EIS

Main Technical Study	Timeframe	Objective
Socio-economic baseline characterization	Short-term	Determine current social and economic characteristics of key local communities, with a focus on demographics, local economies, local governance structures and physical (e.g., water, power, and transport) and social (e.g., schools, hospitals) infrastructure. Define the social values of the affected communities and seek the community's views of the potential impacts of the proposed mine expansion.
Socio-economic impact assessment	Short-term	Identify lessons learnt from the existing operation to inform the assessment of issues that may arise from the proposed expansion. Predict socio-economic impacts of the expansion project on local communities. Identify potential issues for the expansion project, drawing on lessons learnt from existing operations, to maximize positive impacts and minimize negative impacts associated with expansion project development. Review existing operations issues and impacts to identify appropriate management and mitigation measures to address them.

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28.0 CERTIFICATE OF QUALIFIED PERSONS

I Sandra (Sandy) Hunter hereby state:

1. I am employed as a Principal Process Engineer, with the firm Lycopodium Minerals Pty Ltd, Level 2, 60 Leichhardt Street, Spring Hill, Queensland 4000 Australia.
2. This certificate applies to the technical report with an effective date of 1 January 2022, titled 'Independent Technical Report, Kainantu Gold Mine Integrated Development Plan, Kainantu Project, Papua New Guinea'.
3. I am a practising Process Engineer and registered Chartered Professional (Metallurgy) and Fellow of the Australasian Institute of Mining and Metallurgy.
4. I am a graduate of Murdoch University with a Bachelor of Science with Honours in Mineral Science (Extractive Metallurgy) 2001. I have worked as a metallurgist, metallurgy manager and process engineer continuously since 1996. My relevant experience includes process engineer for studies and projects in numerous commodities around the world, with particular focus on gold projects for detailed design and commissioning in Africa and Asia Pacific. Additional relevant experience includes my operational roles to the level of metallurgy manager for gold processing plants in Australia.
5. 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 am a 'Qualified Person' for purposes of NI 43-101.
6. I have visited the project site on one occasion dated 28 to 30 April 2022.
7. I am responsible, either wholly or partly, for sections 1, 13, 17, 18, 21, 22, 24, 25 and 26.
8. I am independent of the Issuer pursuant to Section 1.5 of NI 43-101.
9. I do not beneficially own, directly or indirectly, any securities of K92 Mining Ltd or any associate or affiliate of such company.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report I am responsible for have been prepared in compliance with NI 43-101.
12. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 26th day of October 2022.

Sandra Hunter

AusIMM
THE MINERALS INSTITUTE
CHARTERED PROFESSIONAL
METALLURGY
Sandy Hunter

Sandra (Sandy) Hunter, FAusIMM(CP)

I Andrew Guy Kohler hereby state:

1. I am a full-time employee of K92 Mining Limited employed as Mine Geology and Exploration Manager and reside at 259 Canning Highway Perth, Western Australia 6152, Australia. (Telephone +61-415 842510).
2. In 1986 I graduated from Curtin University of Technology Western Australia with a bachelor's degree in Applied Science (Geology) and in 2004 from Edith Cowan University with a Postgraduate Certificate in Geostatistics.
3. I have over 30 years' 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 and mining projects in Australia, Papua New Guinea, India, Malaysia, Laos and Tanzania. I am currently working as Mine Geology and Mine Exploration Manager at Kainantu 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, as Chief Geologist 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 2005 to 2008, at Phu Kham Copper- Gold operation in Laos. In these roles I have been responsible for mine and exploration geology, drilling, surveying, mine planning, environment (AMD), and assay laboratory.
4. Applicable to the Kainantu Project is my extensive experience in mineral deposits in volcanic terrains, specifically the Penjom gold mine in Malaysia and the Panaustralian copper gold mine in Laos. I have also worked on epithermal/hydrothermal and porphyry-style mineralization in similar environments in Papua New Guinea, Laos, Malaysia as well as Australia.
5. I am a Member of the Australian Institute of Geoscientists (Member No. 6795).
6. For the purposes of the Independent Technical Report titled 'Kainantu Gold Mine Integrated Development Plan, Kainantu Project, Papua New Guinea', effective date 1 January 2022, 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 Sections 1 to 11 of the technical report.
8. I have visited the Kainantu Project on 12 August to 8 September 2021. Prior to that, I was employed at the Kainantu operation on a roster basis, on a 19 on and 9 days off roster from 16 August 2016 to 26 February 2021, before working full time in K92ML's Perth office.
9. I have read the Rule and this technical report is prepared in compliance with its 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 requirement 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. 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. I do hold options directly in K92 Mining Inc, as part of K92 Mining Limited's employee share plan. I have no direct or indirect interest in K92PNG, 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 this 26th day of October 2022.

Andrew Guy Kohler, BAppSc (Geol)

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 46 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 resulted in improved processing operations. I have also taken leading roles in the design and commissioning and optimization of 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, Stillwater and East Boulder concentrators (USA), Millennium Minerals Limited, 1.5 Mtpa Wedgetail Gold Project, Platinum Australia Limited Smokey Hills, Kalplats and Rooderand (RSA), Sylvania Resources Limited PGM recovery from Chrome Mine Tailings), European Metals Holdings Cinovec Lithium Project (Czech Republic), Walkabout Resources Lindi Jumbo Graphite Project (Tanzania).
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 two occasions dated 31 March to 5 April 2017 for 6 days and 9 June to 12 June 2022.
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.
9. I am responsible, either wholly or partly, for Sections 1, and 13 of the Technical Report.
10. I do not hold, directly or indirectly, any shares in K92ML, K92PNG, K92 Holdings, K92 or other companies with interests in the exploration assets thereof. I am independent of K92ML, K92PNG, 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 26th day of October 2022.

Evan Kirby

Evan Kirby, FSAIMM

I Tara Halliday hereby state:

1. My name is Tara Jane Halliday, I work with Tetra Tech Coffey based at Level 11, 2 Riverside Quay, Southbank, Victoria, Australia as a Senior Principal, Environmental and Social Consultant.
2. This certificate applies to the environment and community section of the technical report titled 'Independent Technical Report, Kainantu Gold Mine Integrated Development Plan, Kainantu Project, Papua New Guinea' and dated 1 January 2022.
3. I hold a B.Eng (Environmental) from RMIT University, Australia. I have 22 years' experience in environmental and social management and advisory, much of this has been in the resource sector. I have worked on more than 50 mining projects across a range of geographies. I am a Fellow 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 not visited the K92 Mining Inc project area.
5. I am a responsible person for the Environment and Community section of the technical report titled 'Independent Technical Report, Kainantu Gold Mine Integrated Development Plan, Kainantu Project, Papua New Guinea' and dated 1 January 2022 (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, 21, 25 and 26, of the Technical Report.
10. To the best of my knowledge, information and belief as at 24 February 2022, 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 26th day of October 2022.

Tara Halliday

I Dr Matthew Helinski hereby state:

1. I am employed as a Principal Backfill Engineer, with the firm MineFill Services Pty Ltd, Unit 3B/ 88 Munibung Rd, Cardiff NSW 2285, Australia.
2. This certificate applies to the technical report with an effective date of 1 January 2022, titled 'Independent Technical Report, Kainantu Gold Mine Integrated Development Plan, Kainantu Project, Papua New Guinea'.
3. I am a practising Backfill Engineer, with an undergraduate degree in Civil Engineering, from the University of Newcastle and a PhD in Geotechnical Engineering from the University of Western Australia.
4. I have worked as a Backfill Engineer for the past 20 years on over 100 projects globally. My relevant experience includes many backfill system design studies, technical designs, and system implementation as well as many audits of backfill systems by others. In many cases my role in a specific project has spanned that of Lead Engineer during the scoping and concept development stage, though to the Project Director role during EPC implementation and Commissioning Engineer. This has allowed me to develop an extensive knowledge of both technical and commercial aspects associated with backfill system design and implementation.
5. 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 and past relevant work experience, I am a 'Qualified Person' for purposes of NI 43-101.
6. I am responsible, either wholly or partly, for sections 1, 13, 18, 21, 24 and 25.
7. I do not beneficially own, directly or indirectly, any securities of K92 Mining Ltd or any associate or affiliate of such company.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 26th day of October 2022.

Dr Matthew Helinski

I Shane McLeay herby state:

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 (mining). Hons. I have practised my profession continuously since 1995. My relevant experience for the purpose of the Technical Report is: Over 20 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 have had no involvement with the property prior to my engagement on the Feasibility Study.
6. I am responsible, either wholly or partly, for sections 1, 15, 16, 21, 24, 25 and 26.
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 "Independent Technical Report, Kainantu Gold Mine Integrated Development Plan, Kainantu Project, Papua New Guinea" dated 1 January 2022 for K92 Mining Inc, in compliance with NI 43-101 and Form 43-101F1.
9. That, at the effective date of this technical report 1 January 2022, 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.

Dated this 26th day of October 2022.

Shane McLeay, FAusIMM

I Patrick McCann hereby state:

1. I am a Principal Consultant for Entech Mining Ltd, an independent mining consultant, with an office at 100 King St West, Toronto, Ontario, Canada;
2. I am a graduate from the University of British Columbia, with a BASc (mining). I have practised my profession continuously since 2002. My relevant experience for the purpose of the Technical Report is: Over 20 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 licenced member of the Professional Engineers Ontario.
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 from 14/10/2021 – 21/10/2021. I had no involvement with the property prior to my engagement on the Feasibility Study;
6. I am responsible, either wholly or partly, for sections 1, 15, 16, 21, 24, 25 and 26.
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 'Independent Technical Report, Kainantu Gold Mine Integrated Development Plan, Kainantu Project, Papua New Guinea' dated 1 January 2022 for K92 Mining Inc, in compliance with NI 43-101 and Form 43-101F1.
9. That, at the effective date of this technical report 1 January 2022, 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.

Dated this 26th day of October 2022.

Patrick McCann

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 some 29 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.
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, K92PNG, K92 Holdings, K92 or other companies with interests in the exploration assets thereof. I am independent of K92ML, K92PNG, 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 NI43-101 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 26th day of October 2022.

Ralph Holding, CPEng, FIEAust

27th October 2022

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 35 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 23 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 125 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 30 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 and 2020.
5. I have visited the project's mining lease and operations on one occasion dated 21 to 23 October 2018 for 3 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.
7. I am responsible, either wholly or partly, for Sections 1, 14 and 26 of the Technical Report.
8. I do not hold, directly or indirectly, any shares in K92ML, K92PNG, K92 Holdings, K92 or other companies with interests in the exploration assets thereof. I am independent of K92ML, K92PNG, 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 __27th__ day of October, 2022.

____"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