Taca Taca Project

Salta Province, Argentina

NI 43-101 Technical Report

Effective Date: 31st of December 2025

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ITEM 1 SUMMARY

This Technical Report on the Taca Taca Project (the property or Project) has been prepared by Qualified Persons David Gray, Michael Lawlor and Andrew Briggs of First Quantum Minerals Pty Ltd (FQM, the issuer or the Company).

A first Technical Report prepared by FQM as an issuer in relation to the subject property was filed in November 2020 (the Initial Technical Report). The purpose of the Initial Technical Report was to document updated Mineral Resource and (maiden) Mineral Reserve estimates for the property, after its acquisition from previous owners, and to provide a commentary on the project development status for the property. The Initial Technical Report was amended and restated in March 2021 (the Amended and Restated Technical Report) to include additional clarifications and confirmatory information relating specifically to:

- pre-tax and post-tax cashflows, and also respective sensitivity analyses
- inclusion of the Mineral Reserve inventory within the Mineral Resource inventory
- an indicative order of accuracy for the updated Project capital cost estimates

Both aforementioned Technical Reports are now superseded by this update. In addition to Mineral Resource and Mineral Reserve updates, this Technical Report describes a Stage 1 40 Mtpa project extending for fifty years following a three-and-a-half-year pre-strip and project development phase. Throughout the document, information is also presented on an expansion of the production rate to a 60 Mtpa Stage 2, commencing five years after the pre-strip phase and shortening the production timeframe to thirty five years. The commissioning of a Stage 2 project would coincide with a decrease in the head grade of available plant feed at that time.

Water investigation field campaigns are planned to continue to define aquifer characterisation and validation of fresh water extraction rates to those assumed in desktop studies and to address the water supply for a 60 Mtpa operation

The effective date for the Mineral Resource and Mineral Reserve estimates is the 31st of December 2025.

1.1 Project overview, location and ownership

Taca Taca is a porphyry copper-gold-molybdenum deposit located in the arid Puna (Altiplano) region of Salta Province, in northwest Argentina. The proposed Project involves the open pit mining and flotation processing of cupriferous ore from this deposit for a period in excess of thirty years.

The Project is located approximately 230 km west of the city of Salta and 55 km east of the Chilean border. The nearest population centre is the village of Tolar Grande (population of approximately 150), which is 35 km east of the Project site.

Figure 1-1 shows the location of the Project relative to major roads, railway lines and other existing infrastructure.

The Project site is situated at a median elevation of 3,625 mRL, in an environment with sparse flora and fauna, and on the edge of an expansive salar. The climate at Taca Taca is arid, with an annual precipitation of approximately 40 mm/year and an evaporation rate of 2,500 mm per year. Temperatures range from minus 11°C to plus 20°C, with January being the warmest month and July being the coldest month. Wind speeds typically range from 3.8 m/s to 23.2 m/s, blowing predominantly from the northwest. Although winds are generally strong, particularly during the winter months, development and operational activities could be carried out year round. The Project is located in a seismically active region.

Taca Taca is 100% owned by the Company through its Argentinian subsidiary *Corriente Argentina SA* (CASA). The Project and associated areas of interest are held in a composite package of mining rights consisting of 83 concessions. Two of the mining concessions have a 50% ownership with third party groups, though these are not over commercially material portions of the known deposit. The property is subject to a 3.0% provincial government royalty and a 1.5% third-party net smelter return royalty.

Figure 1-1 Taca Taca Project location

1.2 Project background and description

The Taca Taca deposit was discovered in the late 1960s. Lumina Copper Corporation (Lumina) acquired an interest in the property when shareholders of Global Copper Corporation (Global Copper) approved a corporate reorganisation in August 2008. This ultimately resulted in the acquisition by Lumina of 100% of the shares of CASA and a 100% interest in the property.

In August 2014, the Company acquired Lumina and its primary asset, Taca Taca. Since that time, the Company has completed detailed reviews of the deposit geology, mineralogy and processing amenability, in addition to assessing development options for the Project. From 2015, the Company has conducted water exploration drilling and aquifer pump tests to confirm sustainable groundwater supply sources for the Project and has been progressing with environmental and engineering phase studies. The Project engineering phase remains in progress.

The proposed Project has the following material components, aspects of which are described in detail in this Technical Report update:

- an open pit mine
- a mining waste dump
- a surface ore stockpile
- a copper concentrator for the processing of copper mineralisation by flotation methods, also recovering gold into the concentrate

- an ability to separate a molybdenum concentrate from the final bulk copper concentrate in a dedicated flotation plant
- concentrate filtration and load out facilities
- a tailings storage facility for the storage of the processing residues
- process water storage tanks
- internal access roads, surface haulage roads and mine haulage roads some of which are to be trolley-assisted routes
- borefields for the supply of fresh water
- borefields for the supply of brine from salars
- overland pipelines between the concentrator and the tailings storage facility, and between water supply borefields and the plant site
- mine services workshops and equipment wash-down facilities
- construction offices, mine administration and camp/village accommodation facilities
- storage space and a rail loadout facility for concentrate product
- storage facilities for parts and consumables, reagents and explosives
- as auxiliary infrastructure, there is an airstrip upgrade, roads for transporting supplies into the Project site, a railway for transporting concentrates and supplies, and an in-coming high voltage electric transmission line

1.3 Project approvals

The primary permit required for the development of the Taca Taca Project is the Environmental and Social Impact Assessment (*Informe de Impacto Ambiental y Social,* ESIA) to be approved by the Secretariat of Mining of the Salta Province. This ESIA must cover the main Project sites including mine, process plant, tailings storage facility, water and brine uses, and associated facilities.

A formal ESIA for the Project was submitted to the government authorities in February 2019 (Ausenco, February 2019). The reference technical framework for the ESIA was taken from the Company's Project Description document (November 2018).

A chronology of related events since that time is as follows:

- *Nov 2019* – first round of observations from the Mining and Energy Secretariat (SME)
- *Feb 2020* – responses to first round of observations submitted
- *Aug 2022* – complementary information incorporated into the ESIA including a study of alternatives for the TSF and WRD, plus a hydrogeological study report
- *June 2023* – second round of observations
- *October 2023* - responses to second round of observations submitted
- *October 2024* – SEGEMAR[1] ESIA review, workshop and site visit
- *December 2025* – Appointment of new Mining Secretary of Salta Province

During the government review process, an outline of the proposed Project water supply was submitted as an additional annexure to the ESIA (FQM, October 2023). There were two separate ESIAs also submitted in respect of the Project power supply and road access (FQM, February 2021 and April 2021, respectively).

1 *Servicio Geológico Minero Argentino (SEGEMAR)* was appointed by the provincial Mining and Energy Secretariat (SME) to act as an independent reviewer of the ESIA. SEGEMAR is a national Argentine government institution aimed at producing geological, mining and environmental information to achieve sustained development, and to mitigate geological hazards.

In October 2024, under the facilitation of the SME, a collaborative workshop was held with SEGEMAR and the provincial authorities. This workshop included a site visit, technical briefings and presentations from Company staff, followed by interactive questioning on numerous aspects of the Project.

The ESIA process requires a final report from the SME on their observations and requests for further information. Once the observations process is satisfactorily concluded, there needs to be a public hearing (*audiencia publica*) prior to the ESIA approval. This process is expected to conclude in Q2 2026.

There were two separate ESIAs also submitted in respect of the Project power supply and by pass road access (FQM, February 2021 and April 2021, respectively)

1.4 Project development status

The proposed Taca Taca Project involves the open pit mining and flotation processing of copper and molybdenum bearing ore, with gold recovered into the copper concentrate. A Stage 1 project at a scale of 40 Mtpa is described in this Technical Report, covering detailed planning, engineering, design and costing commentary.

A Stage 2 expansion to 60 Mtpa throughput is also presented throughout this report, commencing five years after the pre-strip phase and shortening the LOM to 35 years. The level of planning, engineering and cost estimation for Stage 2 has not been completed to the same level of detail as for the Stage 1 40 Mtpa project.

The porphyry copper-molybdenum-gold orebody is hosted by a batholith and is overlain by sediments and volcanoclastics. The optimal pit depth is in excess of 700 m and it is situated immediately adjacent to a brine saturated salar. The mineralisation lies beneath a leached cap and is typically hypogene and supergene, but with distinct transitional (or mixed) mineralisation styles also present. The proposed processing flotation processing method involves a conventional concentrator producing separate copper and molybdenum concentrates, and with gold recovered into the+- copper concentrate.

The material technical changes reflected in this Technical Report update can be summarised as:

- Mineral Resources have been reported within a life of mine pit optimisation shell
- a revised water balance, involving the use of fresh water for both rougher and cleaner flotation, and the use of brine for tailings dilution and discharge only
- a move away from an ultra-class truck fleet, and the adoption of a suitably scaled starter fleet for the waste pre-strip period
- a revised design of mining stages to better suit spatial constraints during the substantial pre-stripping period
- a revised estimate of the non-acid forming (NAF) and potentially acid forming (PAF) waste volumes and hence the sequence of waste dumping onto the Salar de Arizaro
- an updated layout for the Project facilities, including the Stage 1 40 Mtpa process plant, and associated non-process infrastructure (NPI) such as the mine services area and rail load-out (concentrate) facility
- a detailed update of the capital and operating cost estimates for a Stage 1 40 Mtpa project

Throughout this Study, reference to the Taca Taca deposit or Project, relates to the Taca Taca Bajo deposit, unless otherwise noted.

As the Project continues through the engineering phase and into the development and operations phases, production aspects may change and components reviewed and varied to suit then current circumstances and new information.

1.5 Geology and mineralisation

Taca Taca is a porphyry copper-molybdenum-gold deposit hosted by granitic rocks together with dacite, dolerite, and rhyolite intrusions. The porphyry is characterised by hydrothermal alteration that grades from a central potassic core to an outer phyllic and argillic zone. Phyllic alteration is most common and is closely associated with mineralisation.

The style of mineralisation is mainly supergene (chalcocite) and hypogene (chalcopyrite), overlain by a zone of variable thickness of leached copper. Mineralisation is disseminated and in fractures, veinlets, and quartz vein stockworks. Copper sulphides are mostly chalcopyrite and chalcocite with lesser bornite, covellite, and digenite and is broadly zoned with a chalcopyrite-bornite-molybdenite core yielding to a more pyritic halo around the edges.

The leached horizon is depleted of copper mineralisation with a zone of gold mineralisation located within the thicker portion of leached material. Supergene zones are enriched with secondary sulphides and form a discontinuous blanket underneath the leached cap. Supergene mineralisation is variably mixed with hypogene mineralisation according to structure, varying lithology and alteration.

Mineralisation remains open at depth and to the south and east of the deposit.

1.6 Metallurgy

Metallurgical testwork by Lumina was completed over a period of three years from April 2010. Up until 2019 there was no additional laboratory work undertaken, although technical reviews done by the Company in 2017 included an assessment of the potential for gold recovery during the Project pre-strip phase.

The PEA report (Ausenco, 2013) summarises the original testwork findings as follows:

- the ore is of moderate competency and hardness, and is amenable to grinding in a conventional semi-autogenous grind (SAG) – ball milling circuit, with pebble crushing and regrinding
- average recoveries would be approximately 90% for copper, 57% for molybdenum and 64% for gold

The Company's 2017 review of the available testwork data highlighted several deficiencies and some uncertainty around metallurgical performance due to the variability of copper mineralisation styles, especially in relation to the extent of mixed mineralisation (i.e., oxidised and tarnished ores). In addition, the majority of the testwork had been conducted using tap water; limited testwork using brine solutions indicated reduced recoveries and lower concentrate grades.

During the course of reviewing the testwork data variability, and as part of the Mineral Resource modelling by the Company, distinct data groupings (clusters) were identified for recovery and copper concentrate grade related to mineralogy, Cu and Fe assay grades.

During 2019, four metallurgical holes were drilled from which ten samples were selected to represent the first five years of operations. These samples along with brine solutions from the Salar de Arizaro, and brackish water from Valle de Arizaro and Valle de las Burras, were sent to the ALS laboratories in Kamloops, Canada.

The testwork programme included comminution work for mill sizing, flotation work in brine and brackish water to define recoveries and concentrate grades in locked cycle testwork, sedimentation and filtration testwork for thickener and concentrate filter sizing, and environmental testwork to determine the potential for acid generation from tailings. This testwork programme was completed in mid-2020.

The comminution testwork highlighted the toughness of the rock types at Taca Taca and indicated the need for secondary crushing to achieve a Stage 2 design throughput of 60 Mtpa in two milling trains or alternatively with the construction of a third milling train. Secondary crushing would not be required to achieve the Stage 1 throughput of 40 Mtpa.

Flotation testwork indicated high mass pulls to rougher concentrates using brine solutions in rougher flotation. Brackish or fresh water would be required in the cleaner flotation circuit to enable high pH values to be achieved for pyrite depression; otherwise low concentrate grades and low recoveries would occur in this circuit. Nonetheless, copper recoveries were generally lower than obtained in the previous testwork campaigns using tap water. More recent work has indicated that sufficient fresh/brackish water is available to support a 40 Mtpa operation.

The data generated from the recent locked cycle testwork was combined with the variability testwork results obtained in the previous testwork campaigns and compared with the results on similar samples tested in tap water to estimate recoveries and concentrate grades for the distinct ore types and the different ranges of copper and pyrite present. These estimates were coded into the Mineral Resource model for adoption in future mine production scheduling and cashflow modelling.

From the testwork results and mine production schedules the following average life of mine recoveries for processing in fresh water are anticipated:

- copper recovery of 87.1% to a concentrate grade of 25.7% Cu
- molybdenum recovery of 44.3% to a concentrate grade of 47% Mo
- gold recoveries to the copper concentrate of 61.4%, with an average grade of approximately 4.5 g/t

These average recovery values are marginally higher than those reported in the 2021 Technical Report.

1.7 Mining

The Taca Taca deposit grades, geometry, and depth make it suitable for conventional, large-scale, bulk open pit mining methods involving blasthole drills, diesel hydraulic excavators, electric shovels and off-highway haul trucks.

Rather than ultra-class equipment as previously considered, a fleet comprising rope shovels of 5,000 t/hour capacity matched to 290 t capacity trucks is now proposed. This fleet would be assisted with 700 t class face shovels and front-end loaders. A “pioneering” fleet is now considered to be better suited to the initial pre-strip and the potentially confined operating space for effective mining during this period.

Open pit mining would proceed in stages from an initial starter pit, supplying pre-strip development waste for site infrastructure and construction, and ore onto stockpile for process plant commissioning. The annual average and maximum material movements over this four year timeframe are 42.1 Mbcm and 66.1 Mbcm, respectively. There is a pronounced peak in material movements over the next ten years as the first three pit stages are completed and mining proceeds into the fourth stage. The annual average and maximum material movements over this period are 75.7 Mbcm and 77.3 Mbcm, respectively. Thereafter, the average annual material movements reduce to 35.3 Mbcm.

Figure 1-2 shows the Project site layout and specifically the location of the open pit and associated waste dump and ore stockpile area.

1.8 Processing

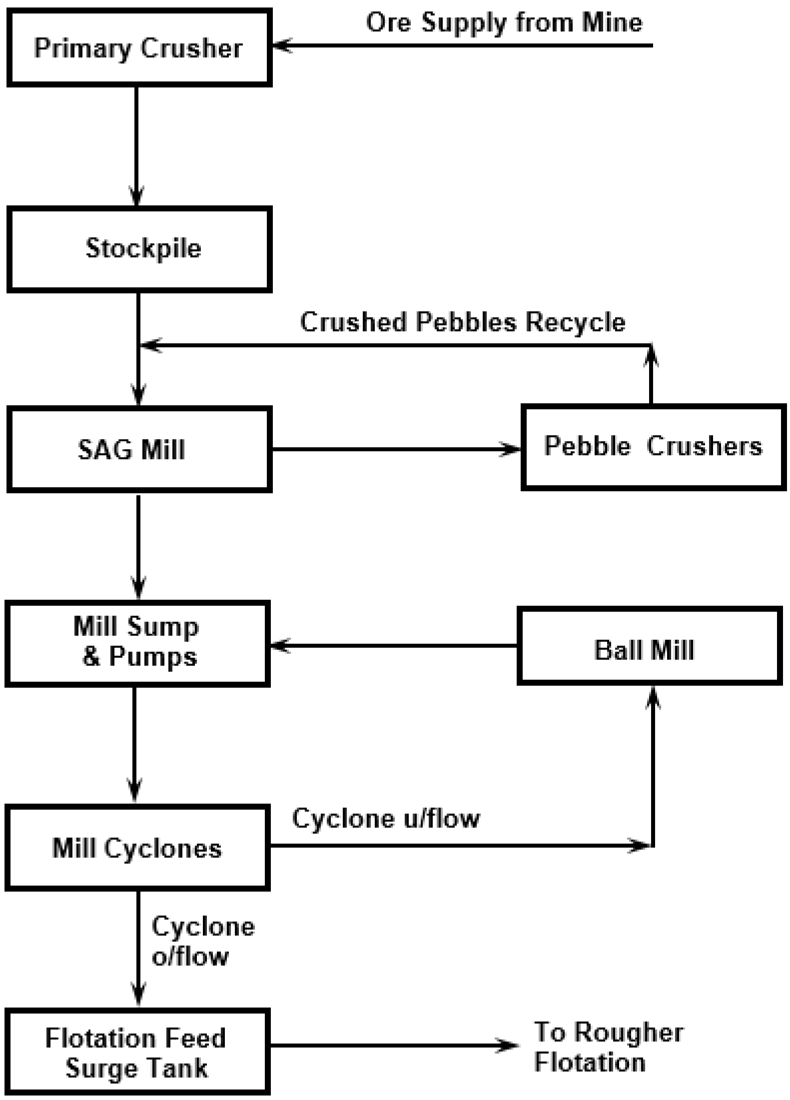
The Taca Taca processing feed would comprise a mix of supergene and hypogene ores with initial feed sourced mainly from supergene zones. Supergene ore is mostly secondary copper sulphide mineralisation (chalcocite) with some primary copper sulphides (chalcopyrite), and minor oxide copper minerals. Hypogene ore is comprised of more than 50% primary copper sulphides. Consequently, the plant feed will always contain significant amounts of secondary sulphides and some tarnished primary sulphides.

The proposed processing method follows the porphyry copper-molybdenum (Cu-Mo) concentrator flowsheets typical in South America. Milling and rougher flotation would be performed in fresh (or brackish) water sourced from off-site borefields.

Figure 1-2 Updated layout of the proposed Project site

Ore will be subject to primary crushing followed by SAG and ball milling to produce a milled product size of 80% passing 180 µm. Two milling trains will be installed, each comprising a 28 MW SAG mill and a 22 MW ball mill (for Stage 1 40 Mtpa processing). An additional milling train would be required to expand the throughput to 60 Mtpa. A simplified flowsheet is shown in Figure 1-3.

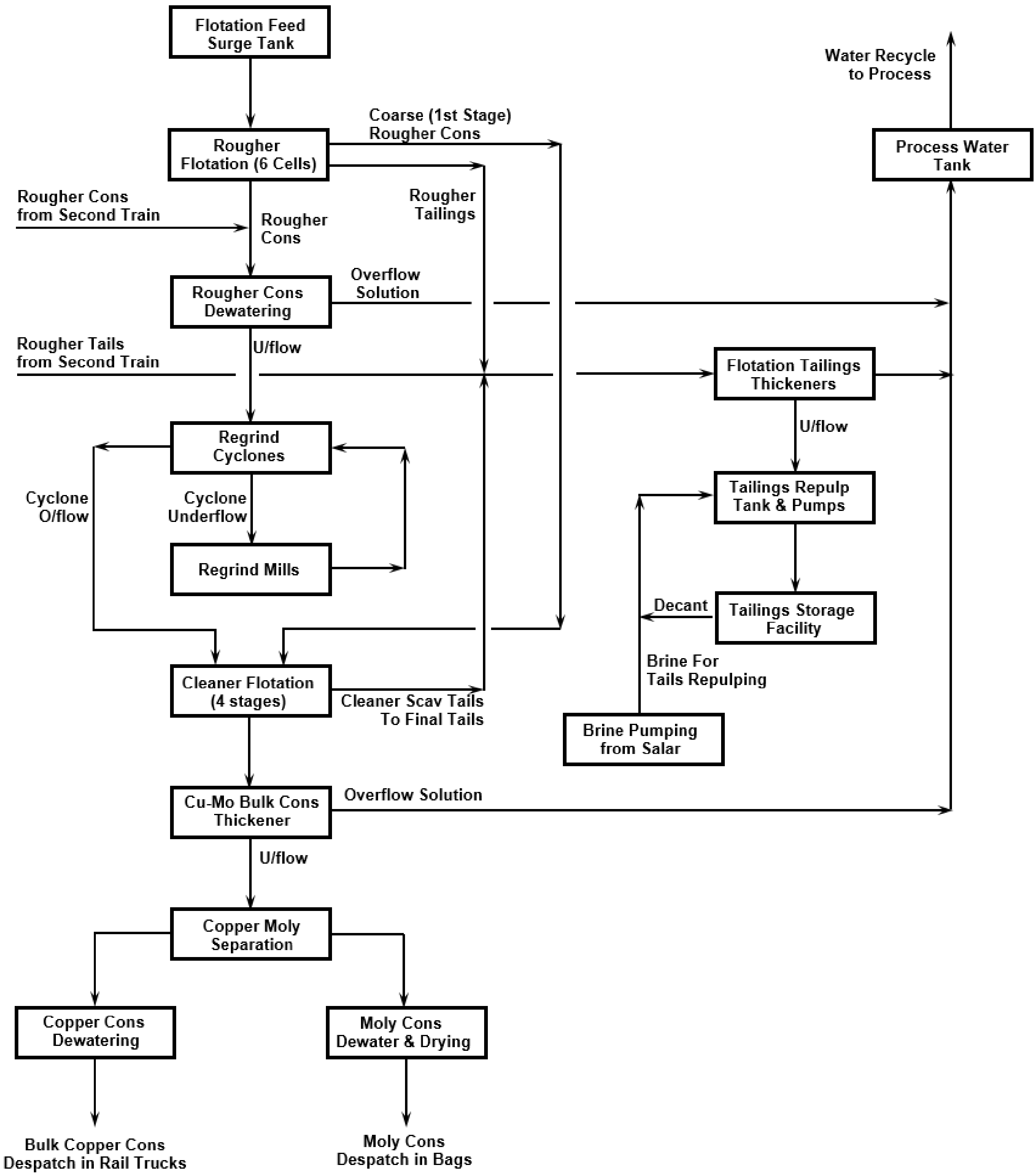
Figure 1-3 Block diagram for comminution circuit



A rougher flotation circuit will produce a rougher flotation concentrate which will be dewatered by thickening, reground to 80% passing 30 µm and re-diluted with good quality water prior to cleaner flotation.

Copper and molybdenum concentrates would be separated from the bulk cleaner concentrate, filtered and dispatched to off-site smelters. Figure 1-4 shows the proposed block flowsheet for the flotation and tailings circuit.

Figure 1-4 Block flowsheet for flotation and tailings



In Stage 1 at 40 Mtpa, ore would be delivered from the mine by haul trucks and crushed in two primary crushers located in proximity to the open pit crest. In Stage 2 a third primary crusher would be added. Following primary crushing, the proposed processing plant would comprise:

- conveying to a coarse ore stockpile with a live capacity of 12 hours
- SAG and ball milling of crushed ore, with size classification by means of hydrocyclones
- pebble crushing on scats generated from the SAG mills
- rougher and scavenger flotation of cyclone overflow slurry
- thickening of flotation tails in high compression thickeners for water recovery
- redilution of thickened tailings with brine for pumping to the tailings storage facility (TSF)

- potential reclaim of decant water from the TSF for usage within the process[2]
- dewatering of rougher concentrates, followed by regrind to 30 µm prior to cleaner flotation
- cleaner flotation of the rougher concentrates to improve the copper grade, with cleaner tails being recycled to the rougher flotation circuit or to final tails
- Cu – Mo separation of the bulk cleaner concentrates in a molybdenum flotation circuit
- dewatering of copper concentrates by thickening and filtration, followed by bulk transportation to off-site smelters
- dewatering of molybdenum concentrates by thickening, filtration and drying, followed by bagging and transportation to off-site smelters
- reagent make-up and dosage systems to support the milling and flotation operations
- water reticulation systems
- compressed air systems to support instrumentation and for automatic valve activation

At the Stage 1 40 Mtpa throughput, an average of 620,000 wet tonnes of copper concentrate (to a maximum of 1,200 ktonnes) is expected to be generated annually at an average grade of 25.7% Cu, along with an average 5,100 tonnes of molybdenum concentrate (to a maximum of 7,200 tonnes) at a grade of 47% Mo. In the first ten years of operations at this scale of processing, the average and peak copper concentrate production figures are 965,000 wet tonnes and approximately 1.2 million wet tonnes, respectively.

Assuming that Stage 2 commences five years after initial commissioning, with an expansion to 60 Mtpa processing, the life of mine copper annual concentrate production would rise to an average of 903,000 wet tonnes, with a peak of approximately 1.4 million tonnes.

Flotation tailings would be dewatered in high pressure thickeners for maximum recovery of good quality water. This would be followed by redilution with brine, and then pumping of the tailings at 65% solids to the TSF located on the Salar de Taca Taca.

Gold would be recovered to the copper concentrate through flotation.

A gold recovery circuit for the treatment of the auriferous leach cap is not proposed as part of the Stage 1 or Stage 2 production plan. Subject to further testwork during the early phases of operations, this material would be stockpiled separately and could be reclaimed and treated later if deemed economic to do so.

1.9 Tailings storage and water reclamation

A downstream raised TSF is planned to be located on the Salar de Taca Taca, a natural embayment of the Salar de Arizaro, located 6 to 12 km north of the processing plant site. The ultimate capacity is approximately 1.24 Bm^3 and could be expanded through further lifts. The site is almost entirely enclosed by the natural land mass and would only require a relatively low final height of 22 m plus an additional 3 m of freeboard, and a 1,500 m length embankment at the entrance to the salar.

The starter embankment would be constructed as an initial waste rock bund, and then downstream raised in continuous stages using non-acid forming waste rock from the mine.

Two tails lines, each with two stages of centrifugal pumps would be installed to deliver tailings to the TSF, with spigots arranged around the facility's periphery. Water run-off from the site and from sediment collection ponds would be pumped to the tailings thickener and, subject to further engineering analysis, excess water then recycled back to the plant.

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[2] The current process plant water balance assumes that there will be no reclaim from the TSF.

1.10 Infrastructure

FQM commissioned several consultants to supplement FQM's internal engineering team and expand upon the preferred Project infrastructure components identified and reported on in the 2021 Technical Report. Suitably comprehensive reports, drawings and spreadsheet information packages were produced by the following engineering firms, for a 40 Mtpa Stage 1 project:

- Lycopodium Ltd (Lycopodium, April 2025) were engaged to undertake the design work for the processing plant (copper and molybdenum processing), primary crushing to copper concentrate filtration and tails pumping.
- Fluor Australia Pty Ltd (Fluor, October 2024 and February 2025) were engaged to design the mining and non-process infrastructure including a Salta Operations Centre, the site village, remote water and brine borefields, construction and off-site roads, airstrip, mining service area, copper concentrate conveyor and storage area, plus rail loading facilities.
- Process E&I were engaged to provide the electrical and instrument design drawings across the site, including HV switchyard, power distribution to mine, plant and infrastructure, and support for the HV powerline from La Puna to site.

The immediate Project site layout and the broader overall layout are shown in Figure 1-5. This updated Project layout covers numerous detailed site infrastructure components, specifically the Stage 1 40 Mtpa processing plant, the non-processing infrastructure (NPI), mine services area (MSA), administration facilities and camp. Engineering of off-site infrastructure in Salta and in relation to the site road access and the power supply connections into the site have also been addressed.

Some aspects of this layout were updated after it was produced, namely the layout of the primary crusher bench within the open pit, the alignment of the crushed ore conveyor route, and the MSA and NPI detailed layouts.

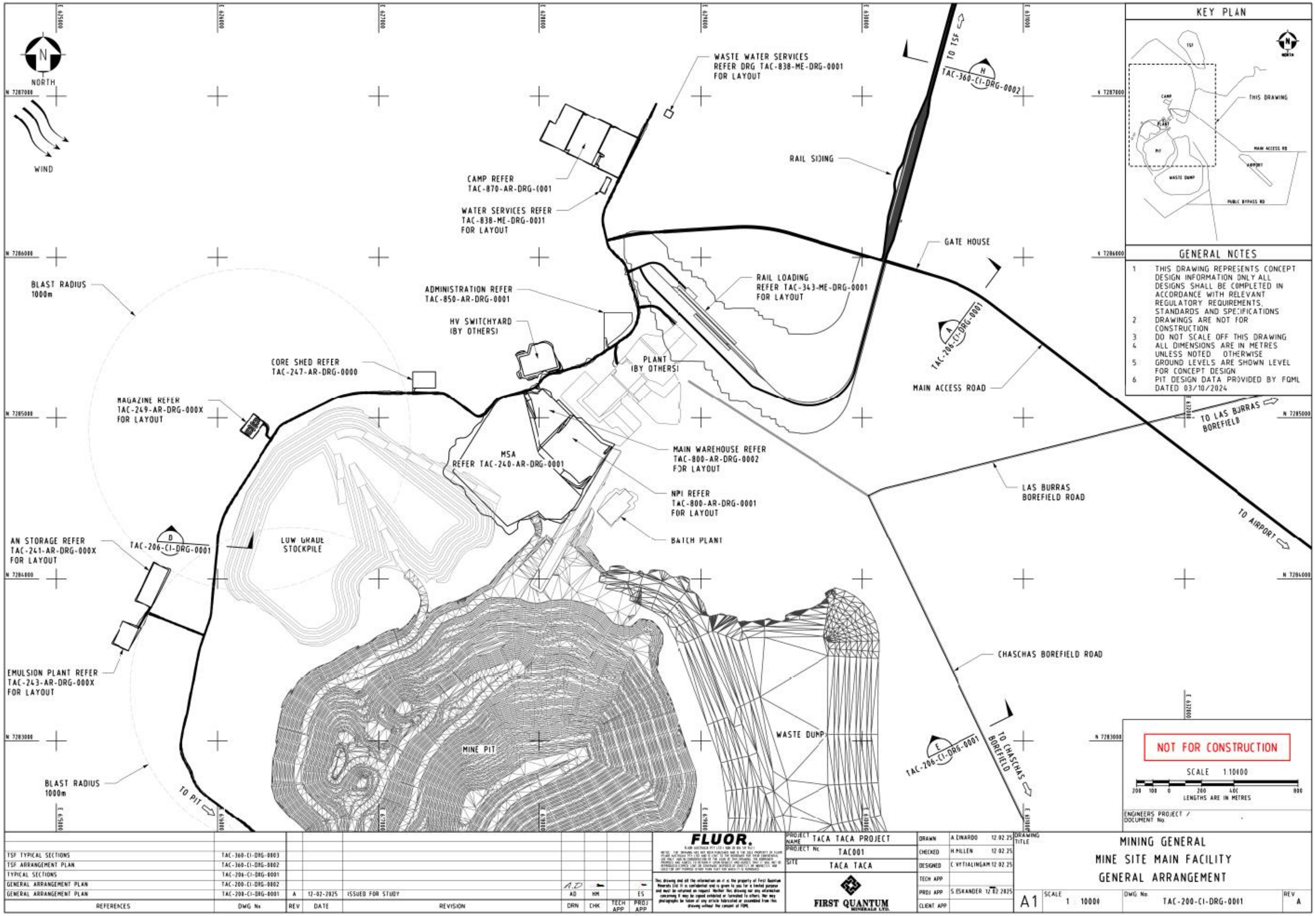
1.11 Power and water supply

The total power demand for the Project is expected to be in the range of 137 MW to 205 MW at a processing rate of 40 Mtpa up to 60 Mtpa. A preferred power supply and transmission route has been identified involving 122.5 km of new transmission line and a new switching station to connect to an existing 345 kV line that extends through northern Argentina and into Chile. A preliminary design and estimates have been produced by an Argentine specialist consultant to support the development of the ESIA required for the power supply infrastructure.

The proposed new transmission line will connect the site to the national grid and enable the Project to source its entire electricity supply requirements through a long-term power supply agreement with an electricity supplier, to be determined through a competitive tender process. The Company has identified options to source 100% of its electrical energy requirements from renewable sources. Further alternatives exist, if required, to source a portion of the energy requirements from natural gas power plants in Salta and regionally.

Local and regional borefields will be developed to supply a combination of fresh water and brine for the Project. Most of the processing water supply is intended to be fresh water abstracted from regional borefields. Tailings thickening to 70% solids would result in the return of fresh water to the process plant. Brine from the adjacent salar is intended for use in the re-pulping of thickened tailings prior to pumping to the TSF. The required fresh water make-up for Stage 1 40 Mtpa processing is 1,965 m^3/hr, and for brine is 471 m^3/hr. For Project wide consumption (i.e., including the mining operations and the camp requirements), these figures increase to 2,130 m^3/hr and 600 m^3/hr, respectively.

Figure 1-5 Site infrastructure layout plan (source: Fluor, February 2025)



Fresh water supply investigations to date have focussed the search and drilling investigations to four regional basins located at 30 km to 50 km distance from the Project site. The water supply status of the Project as at Q4 2025 is:

- Major water resources have been identified at Valle de Arizaro, Socompa, Valle de Chaschas and Valle de las Burras, with thick zones of permeable, water saturated sands and gravels intersected in several drill holes and backed up by geophysical prospecting data.
- Historic and more recent FQM pump testing to date has shown good transmissivity results in all four basins suggesting pumping at rates of 40 to 50 L/s per bore will be possible in each basin.
- The four identified fresh water supply basins have a combined estimated yield in excess of that required for process water make-up for the Stage 1 40 Mtpa project.
- Additional and potentially more distant, fresh water supply sources will be required to sustain processing for the Stage 2 60 Mtpa project if the assumed mid-point of water extraction is confirmed during future field campaigns
- Remote sensing weather data, monitoring, and environmental baselining works are continuing, thereby allowing for increased confidence in water balance predictions for the Project.
- Field investigations in the second half of 2025 have indicated a potential for up to 1,000 m^3/hr of borefield supply of brine from the Salar de Arizaro and Salar de Taca Taca.
- Water sources identified for the Project are characterised by elevated dissolved mineral content and are not suitable for human consumption.
- The Project water sources are hydraulically separated from groundwater and surface water resources currently used by local communities.

An allowance has been included in the demand calculations to enable irrigation of the local catchments and to minimise impacts to sensitive ecosystems.

A specialist consultant has assisted with the specification of bore design and estimated numbers, bore spacing at each borefield source, and the nomination of pumping rates. A capital cost estimate for the bore pumps and pipelines has been completed, considering the number of bores required, the drilling depth, bore pumps, the pipeline distances and the pumping head.

1.12 Road and rail access

Existing public roads provide access to the Project site. The 2021 Technical Report described a site access route involving a deviation from RN N°27 at a point south of Cauchari (at km 28), passing over the Cerro Maćon in the north to later re-join RN N°27. Access is now designed to continue along the original RN N°27 route through Pocitos and onwards to Tolar Grande thereby reducing the length of new road to be constructed. Certain sections of the original route have been upgraded by the Government to suit heavy road haulage.

The Project is located approximately 5 km from an existing railway line that connects Salta with Mejillones, Chile. This railway will be used for copper concentrate transport to a port at Mejillones Bay, for subsequent shipment to smelters globally. Construction of a new rail spur, a new maintenance and repair facility for locomotives and railcars, adjacent to the concentrate load-out facility, and rehabilitation across sections of the railway line will be required. Engineering of the railway line is now being addressed in some detail and high-level discussions are underway with the owners of respective sections of the rail corridor.

1.13 Mineral Resource estimate

The December 2025 updated Mineral Resource estimate has considered geological domains for weathering, rock type, alteration and styles of mineralisation.

Data from a total of 399 diamond (DD) and reverse circulation (RC) drill holes, comprising 164,822 m of analysed samples, were included in this update. The most recent drilling campaign, completed in 2022, has no impact on this update as most of the added holes were located external to the pit shell.

Grade estimation was completed following the same methodology as described in the 2021 Technical Report. Apart from modelling improvements to the overburden and base of leached material, estimation domains and parameters were not modified due to limited data additions. The base of the leach cap was updated following a review of drilling data and geochemical characteristics.

Geometallurgical domains were refined according to iron and sulphur concentrations, in addition to copper grade, to more effectively differentiate material types for the assignment of variable metallurgical recoveries. This approach improves the spatial representation of metallurgical responses while remaining consistent with the existing geological interpretation.

Estimated block grades were demonstrated to reflect sample grades and the prevailing in-situ mineralisation. Validation results are in support and include use of summary statistics, visual validations, swath plots and comparison with previous estimates.

The block model estimate was classified according to confidence in the estimates, drill hole sample spacing as well as the degree of geological and grade continuity. The classified Mineral Resources was constrained to a life-of-mine pit shell, as per the Mineral Reserves, and therefore supports reasonable prospects for economic extraction.

Mineral Resource estimates were reported at a 0.11 % copper equivalent (CuEq) cut-off grade to be consistent with this report's updated Mineral Reserves.

The December 2025 Mineral Resource statement is listed in Table 1-1. The Mineral Resource inventory is inclusive of the Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1 Mineral Resource statement at 31st December 2025 within the life of mine pit shell and using a 0.11% CuEq cut-off

Classification	Tonnes (mt)	Density (t/m^3)	Cu (%)	Mo (%)	Au (g/t)	CuEq* (%)	Cu Metal (kt)	Mo Metal (kt)	Au metal (koz)
Measured	441	2.67	0.58	0.015	0.13	0.67	2,557	68	1,868
Indicated	1,637	2.65	0.38	0.011	0.07	0.43	6,159	185	3,847
Measured plus Indicated	**2,078**	**2.66**	**0.42**	**0.012**	**0.09**	**0.48**	**8,716**	**253**	**5,715**
Inferred	145	2.66	0.27	0.007	0.06	0.31	389	10	263

* CuEq = Cu% + (Au g/t*recovery*metal price)/((Cu%* recovery*metal price)/Cu%) + (Mo % *recovery* metal price)/((Cu%* recovery*metal price)/Cu%)

1.13.1 Model data for mine planning – metallurgical recovery

Due to the use of fresh water during rougher and cleaner processing, the metal recovery projections have changed since the 2021 Technical Report. The updated and summarised process recovery projections are as listed in Table 1-2.

Table 1-2 Average process recovery projections

	Cu rec (%)	Mo rec (%)	Au rec (%)
Secondary	83.2%	40.0%	41.6%
Mixed	88.3%	40.0%	55.0%
Primary	89.5%	60.0%	66.6%

1.13.2 Model data for mine planning – acid rock drainage

In the 2021 Technical Report it was stated that the volume of non-acid forming (NAF) mined waste would likely be insufficient to provide a reasonable base layer under the waste dump on the adjacent salar. This conclusion was drawn from preliminary acid base accounting (ABA) work, a preliminary interpretation of the NAF/PAF (potentially acid forming) threshold based on %S grades in the Mineral Resource model, and the then prevailing life of mine production schedule.

As part of the Mineral Resource update for this Technical Report, the %S discriminator was re-evaluated, resulting in updated NAF/PAF definition criteria to better reflect the deposit-specific weathering profile and sulphide mineralisation characteristics. The updated framework recognises that sulphur oxidation behaviour and acid generation potential vary with both weathering state and sulphide mineralogy (with chalcocite expected to oxidise more rapidly than chalcopyrite and pyrite).

The update also incorporated an immediate sub-surface zone of scree, gravels and sands classified as NAF and classified all sedimentary rocks and evaporite units as such. Statistical analysis of %S distributions within these material domains, supported by expert guidance and analogue porphyry copper experience, was used to select appropriate %S cut-offs for ARD classification. Based on this re-evaluation, the ARD threshold applied to the surface leached zone was increased from 0.1%S to 0.3%S. The impact of this change was a significant increase in the volume of NAF available to be used for waste rock dump basal layer construction, TSF embankment construction, and infrastructure earthworks.

1.14 Mineral Reserve estimate

An updated Mineral Reserve estimate has been produced for Taca Taca (Table 1-3). The mine plan was developed using the Measured and Indicated Mineral Resource, whilst Inferred Mineral Resource was allocated to waste. Mining assumes conventional open pit operations using truck-and-shovel technology. The estimated Mineral Reserve was determined using metal prices of $3.50/lb for copper, $12.00/lb for molybdenum, and $1,800/oz for gold, with a supporting production schedule derived from the ore and waste mining inventory within a practical pit design produced from a selected pit optimisation shell.

The actual marginal cut-off grade for the Mineral Reserve varies according to the copper recovery assigned to the various mineralogical groupings. However, the overall average marginal copper cut-off grade is approximately 0.11% Cu_{eq}.

Table 1-3 Taca Taca Mineral Reserve statement at 31st December 2025

Classification	Tonnes (Mt)	Cu (%)	Mo (%)	Au (g/t)	Cu metal (kt)	Mo metal (kt)	Au metal (koz)
Proven	432.1	0.58	0.015	0.13	2,509.5	66.8	1,835.6
Probable	1,558.0	0.38	0.011	0.07	5,919.0	177.6	3,696.6
Prov. + Prob.	**1,990.1**	**0.42**	**0.012**	**0.09**	**8,428.5**	**244.4**	**5,532.2**

A retrospective pit optimisation was completed to test the impact on the Table 1-3 Mineral Reserve inventory due to the adoption in the Project economic analysis of higher metal prices and higher operating costs. A summary analysis on this topic is provided in Item 1.17 and Item 1.19.2.

1.15 Production schedule

The Stage 1 40 Mtpa production schedule for this Technical Report update is listed in Table 1-4, whilst Figure 1-6 shows the corresponding annual mining tonnage (ore, waste and stockpile rehandle) for the Project life of mine. Figure 1-6 and Figure 1-7 show the corresponding plant feed and recovered metal profiles, respectively.

Features of the Stage 1 40 Mtpa mining and production schedule are as follows:

- Pre-strip mining commences in mid-Year 1 whilst processing commences in Year 5.
- The Project life is 50 years (processing years) at 40 Mtpa.
- 422.1 Mt of waste is mined in the pre-strip period, during which time 10.5 Mt of ore is mined onto a stockpile for subsequent active and longer-term reclaim.
- The total material mined over the life of operations amounts to 5,615.5 Mt (1,737.0 Mbcm), of which 1,990.1 Mt is ore with average grades of 0.42% Cu, 0.012% Mo and 0.09 g/t Au, and 2,903.4 Mt is waste.
- The overall life of mine strip ratio (waste tonnes: ore tonnes) is 1.46 : 1.
- The direct feed ore to the plant is 1,268.1 Mt at an average grade of 0.56% Cu, whilst 348.6 Mt at an average grade of 0.26% Cu is ore reclaimed from active stockpiles, and 373.3 Mt at an average grade of 0.13% Cu is ore (marginal ore) reclaimed from longer term stockpiles, mostly after mine depletion.
- The Inferred Mineral Resource that is mined and stockpiled as waste amounts to 124.5 Mt, or 4.3% of the total waste mined. This material is encountered in the mining schedule after Year 9 and following completion of mining the stage 1 pit.
- The Stage 1 crusher feed ramps up from Year 5 at 24.9 Mt, and thereafter to 40 Mtpa until Year 50.
- In terms of total plant feed at 40 Mtpa (after mining dilution/recovery):
  - the weighted average copper grade is 0.73% Cu for the first eight years from initial commissioning,
  - then 0.53% Cu to Year 17,
  - then 0.44% Cu to Year 37,
  - then 0.21% Cu to Year 41,
  - and finally 0.13% Cu for the remaining nine years of Project life when reclaiming from longer term stockpiles
- Before the final thirteen years of long term stockpile reclaim, the total plant feed is 1,465.8 Mt at an average grade of 0.52% Cu.
- The annual average copper metal production to Year 12 (after eight years of processing) is 239.5 kt, and ranging between 112.4 kt and 283.1 kt. Thereafter, the annual average is 162.1 k5, ranging between 121.20 kt and 207.4 kt (ignoring the final years of stockpile reclaim). In terms of life of Project totals:
  - 1,915.9 kt of copper is recovered in the first eight years,
  - then 4,712.2 kt of copper to Year 41,
  - and finally 683.6 kt of copper for the remaining thirteen years of Project life when reclaiming from longer term stockpiles
- In terms of total metal produced during the Project life, the production plan generates 7,312 kt of copper, 108 kt of molybdenum, and 3,367 k(t)oz of gold.

A preliminary 60 Mtpa Stage 2 production schedule was produced, featuring an expansion from Stage 1 at Year 5. Summary features of this preliminary 60 Mtpa schedule are:

- The Project life reduces from 50 years to 35 years
- In terms of total plant feed at 60 Mtpa (after mining dilution/recovery):
  - the weighted average copper grade reduces to 0.66% Cu for the first eight years from initial commissioning,
  - then 0.51% Cu to Year 17,
  - then 0.46% Cu to Year 22,

- then 0.44% Cu to Year 30,
- and finally 0.16% Cu for the remaining years of Project life when reclaiming from longer term stockpiles

Table 1-4 Life of mine production schedule; Stage 1 40 Mtpa

Year	Stage	Mining						Processing				Metal Recovered		
		Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)	Waste (Mt)	Total Mined (Mt)	Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)	Cu (kt)	Mo (kt)	Au (k(t)oz)
1	Pre-strip					19.3	19.3							
2						94.3	94.3							
3		0.5	0.18	20.34	0.06	146.7	147.3							
4		9.9	0.28	34.41	0.06	161.7	171.6							
5	Production	52.3	0.37	52.72	0.07	143.0	195.2	24.9	0.61	71.60	0.09	112.4	0.0	38.5
6		51.5	0.59	80.74	0.10	143.8	195.2	40.0	0.73	92.31	0.12	240.6	1.4	84.1
7		45.5	0.77	110.81	0.14	150.3	195.8	40.1	0.78	110.56	0.14	271.5	1.8	109.6
8		42.4	0.69	136.20	0.12	152.8	195.2	40.0	0.78	119.50	0.13	271.9	2.0	97.9
9		52.7	0.67	182.71	0.14	142.5	195.2	40.0	0.80	178.74	0.15	283.1	3.0	113.8
10		46.7	0.67	196.76	0.17	148.5	195.2	40.0	0.80	206.76	0.18	282.8	3.7	137.9
11		56.2	0.56	118.68	0.11	139.5	195.8	40.1	0.71	147.23	0.13	246.8	2.7	102.1
12		69.9	0.42	87.35	0.09	125.4	195.2	40.0	0.60	127.53	0.12	206.9	2.3	96.8
13		57.8	0.40	79.91	0.08	137.4	195.2	40.0	0.50	102.33	0.10	170.7	2.0	79.3
14		55.0	0.37	84.69	0.08	140.2	195.2	40.0	0.46	103.02	0.10	160.0	2.0	81.4
15		57.8	0.39	114.85	0.10	137.9	195.8	40.1	0.49	132.91	0.11	172.3	2.3	88.0
16		63.4	0.40	108.27	0.09	116.8	180.2	40.0	0.52	128.16	0.12	181.7	2.4	92.2
17		74.0	0.43	118.59	0.11	82.0	156.0	40.0	0.59	136.44	0.13	207.1	2.4	103.2
18		73.8	0.44	111.62	0.10	58.0	131.8	40.0	0.59	129.65	0.12	207.4	2.4	97.7
19		57.3	0.42	110.48	0.11	59.8	117.1	40.1	0.59	132.26	0.14	207.4	2.5	112.4
20		51.9	0.42	96.65	0.09	64.9	116.8	40.0	0.50	110.42	0.11	173.6	2.1	87.3
21		48.5	0.42	98.30	0.09	68.3	116.8	40.0	0.50	109.82	0.10	174.7	2.1	79.9
22		43.6	0.37	149.35	0.10	73.0	116.6	40.0	0.41	132.48	0.10	142.7	2.5	78.8
23		55.5	0.40	130.07	0.09	43.3	98.8	40.1	0.47	141.05	0.11	163.4	2.6	84.7
24		56.6	0.35	139.17	0.08	41.9	98.5	40.0	0.46	133.11	0.09	157.9	2.3	67.2
25		60.9	0.34	106.43	0.07	37.6	98.5	40.0	0.45	110.43	0.08	154.1	1.8	62.2
26		60.3	0.38	112.94	0.07	38.2	98.5	40.0	0.47	126.75	0.09	162.2	2.2	67.9
27		62.6	0.31	135.46	0.07	36.2	98.8	40.1	0.42	148.16	0.08	148.2	2.6	66.2
28		59.2	0.27	107.75	0.06	39.3	98.5	40.0	0.35	118.42	0.08	120.2	2.1	60.9
29		68.4	0.31	110.86	0.06	30.1	98.5	40.0	0.43	106.09	0.07	149.0	2.1	55.8
30		67.0	0.33	115.16	0.05	31.5	98.4	40.0	0.45	108.85	0.06	158.6	2.2	49.6
31		56.0	0.32	129.17	0.07	19.4	75.4	40.1	0.39	134.14	0.08	138.6	2.6	65.3
32		49.5	0.33	145.94	0.07	10.6	60.2	40.0	0.38	146.26	0.08	134.8	2.8	64.6
33		47.5	0.36	132.79	0.07	12.7	60.2	40.0	0.40	128.82	0.08	141.7	2.4	63.3
34		45.8	0.37	134.65	0.06	14.4	60.2	40.0	0.41	134.36	0.07	146.0	2.4	59.0
35		47.0	0.37	146.59	0.07	13.3	60.3	40.1	0.42	146.18	0.08	148.9	2.7	61.2
36		50.9	0.43	166.00	0.07	9.2	60.2	40.0	0.51	172.74	0.08	181.5	3.0	61.1
37		53.1	0.42	170.87	0.07	7.1	60.2	40.0	0.50	175.16	0.08	176.9	2.9	65.6
38		52.7	0.41	155.48	0.08	5.5	58.2	40.0	0.47	161.40	0.09	167.2	2.7	68.5
39		27.8	0.44	166.93	0.07	2.8	30.6	40.1	0.40	156.62	0.08	141.8	2.7	59.2
40		27.7	0.54	163.31	0.08	2.5	30.2	40.0	0.44	151.99	0.08	155.1	2.6	62.7
41		30.9	0.56	162.92	0.07	1.5	32.4	40.0	0.48	154.74	0.08	168.5	2.7	59.1
42								40.0	0.21	130.66	0.08	71.6	2.3	62.3
43								40.1	0.21	130.41	0.08	71.5	2.3	61.3
44								40.0	0.21	130.11	0.08	71.0	2.3	59.7
45								40.0	0.21	130.11	0.08	71.0	2.3	59.7
46								40.0	0.13	77.70	0.03	45.5	1.4	28.0
47								40.1	0.13	74.91	0.03	44.2	1.3	26.4
48								40.0	0.13	74.91	0.03	44.1	1.3	26.3
49								40.0	0.13	74.91	0.03	44.1	1.3	26.3
50								40.0	0.13	74.91	0.03	44.1	1.3	26.3
51								40.1	0.13	74.91	0.03	44.2	1.3	26.4
52								40.0	0.13	74.91	0.03	44.1	1.3	26.3
53								40.0	0.13	74.91	0.03	44.1	1.3	26.3
54								43.9	0.13	74.91	0.03	44.1	1.3	26.3
Total		**1,990.1**	**0.42**	**122.8**	**0.09**	**2,903.4**	**4,893.5**	**1,990.1**	**0.42**	**122.8**	**0.09**	**7,311.7**	**108.2**	**3,367.0**

Figure 1-9 and Figure 1-10, show comparative charts for Stage 1 40 Mtpa and Stage 2 60 Mtpa processing, depicting the plant feed and recovered metal profiles, respectively. From these charts, it can be appreciated that the drop in the 60 Mtpa feed grade from Year 9 is compensated for by the increase in processing rate, resulting in the peak of recovered metal over the ensuing five years.

Figure 1-6 Chart of annual mining volumes; Stage 1 40 Mtpa

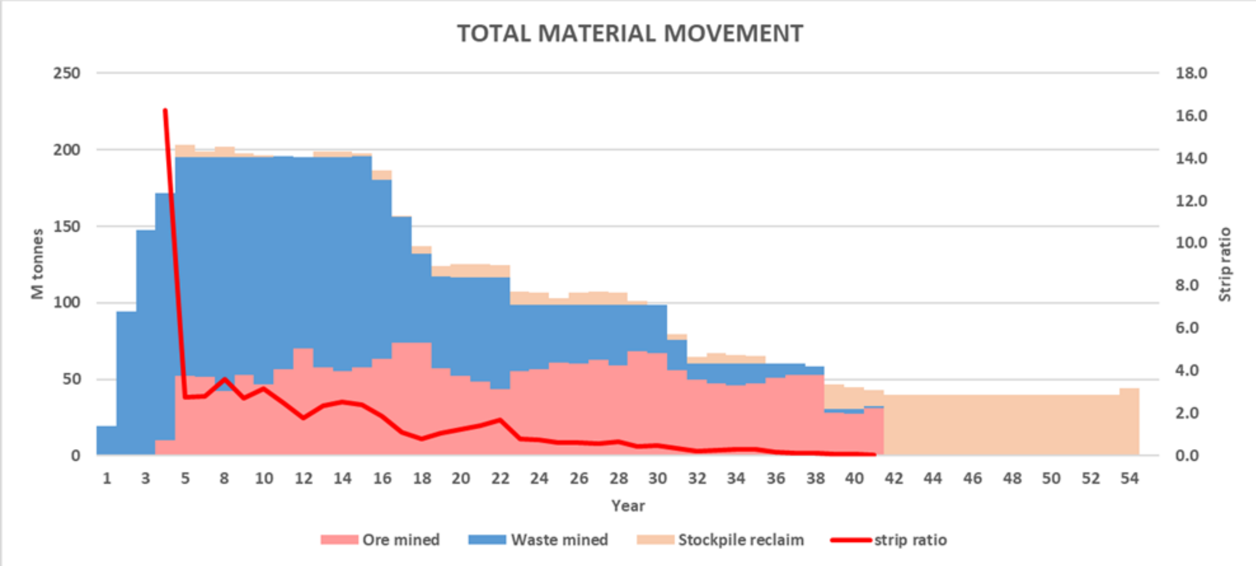


Figure 1-7 Chart of annual plant feed tonnage and feed grade profile; Stage 1 40 Mtpa

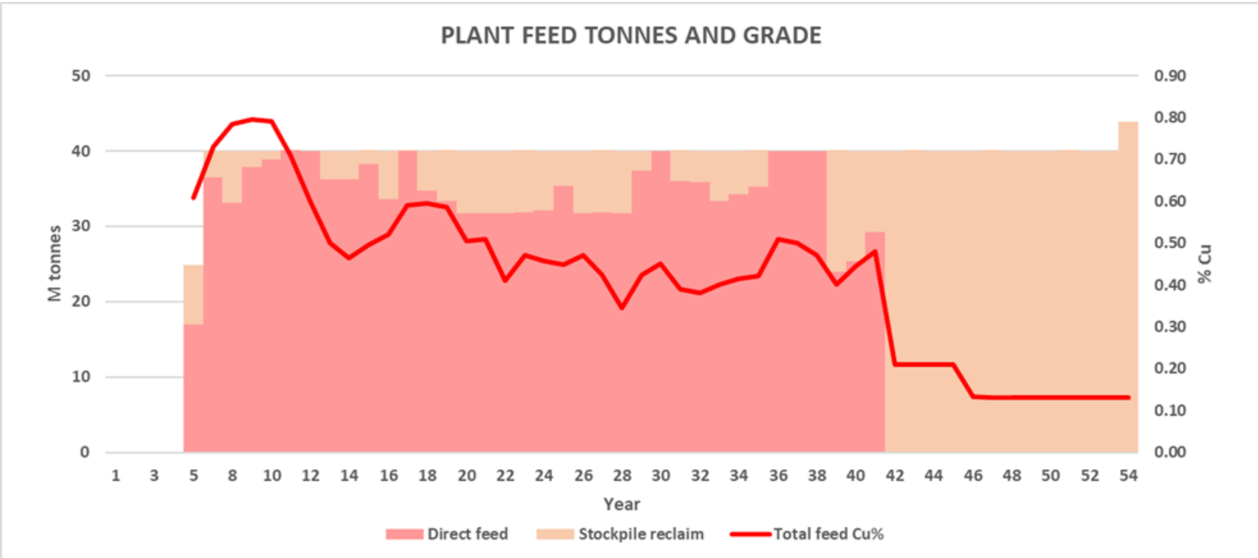


Figure 1-8 Chart of annual recovered copper metal and plant recovery profile; Stage 1 40 Mtpa

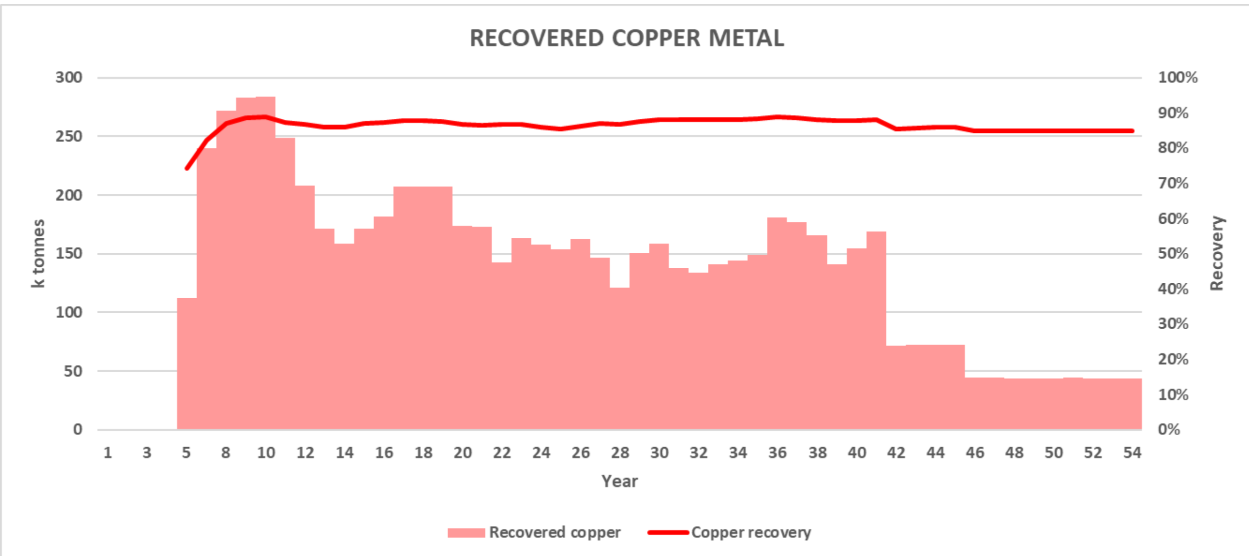


Figure 1-9 Chart of annual plant feed tonnage and feed grade profile; Stage 1 40 Mtpa vs Stage 2 60 Mtpa schedules

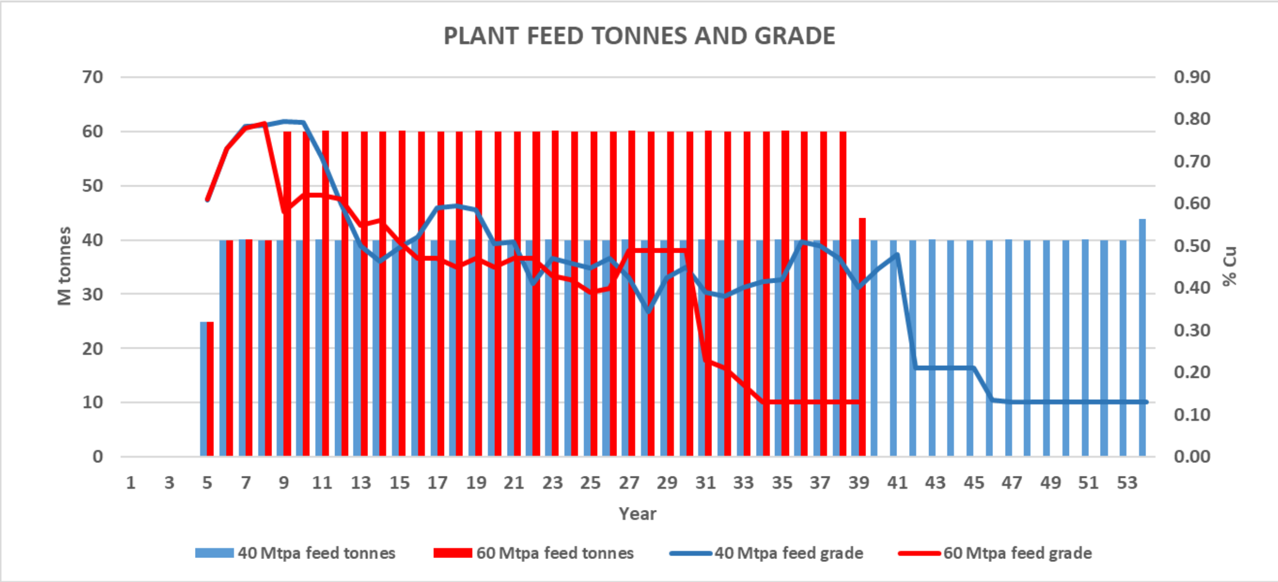
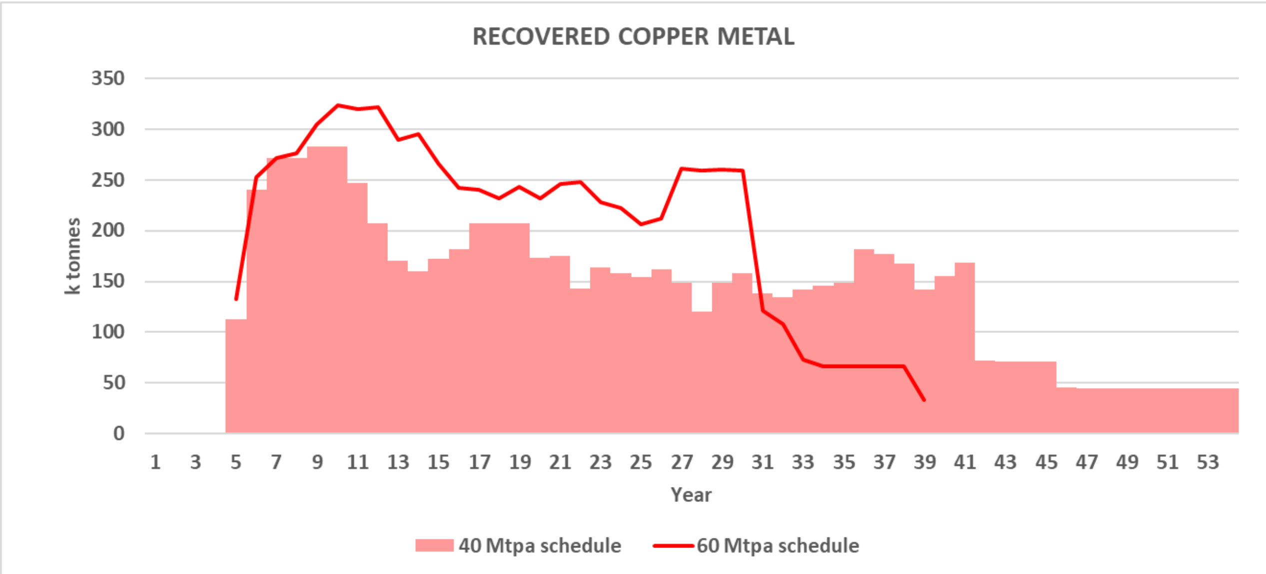


Figure 1-10 Chart of annual recovered copper metal profile; Stage 1 40 Mtpa vs Stage 2 60 Mtpa schedules



1.16 Capital, sustaining and operating cost estimates

The Company commissioned an Engineering Cost Study between 2024 and 2025 for the purposes of developing an updated and comprehensive infrastructure design and associated capital cost estimate, activity schedule and execution plan for the Stage 1 40 Mtpa Project.

In conjunction with this study, capitalised mining pre-strip (ore and waste) costs were developed by FQM mining engineers to reflect the unit mining costs and the production schedule for the mine pre-strip period. The initial mining infrastructure costs were also developed by FQM, referencing the production schedule and associated mining fleet complement. Otherwise, the processing, rail, infrastructure and indirect costs as determined by Lycopodium for the Project development period (Lycopodium, June 2025), are as listed in Table 1-5.

Table 1-5 also lists the operational capital estimate inclusive of deferred stripping charges and ultimate mine closure costs. Sustaining costs are also listed and these account for explicit cost estimates associated with mining equipment replacements over the life of the operations, in addition to a nominal allowance for processing plant and infrastructure replacements.

Table 1-5 Summary of Stage 1 Project capital and sustaining cost estimates

	UNITS	Lycopodium Total	FQM Adjustments	FQM Total
Project development capital				
Mining				
Mining pre-strip ore and waste	$'000	$674,078	$80,591	$754,669
Mining infrastructure	$'000	$802,709	$198,572	$1,001,281
Subtotal mining	**$'000**	**$1,476,786**	**$279,164**	**$1,755,950**
Processing	$'000	$1,077,482	$0	$1,077,482
Rail	$'000	$12,776	$0	$12,776
Infrastructure	$'000	$727,043	$0	$727,043
Indirects	$'000	$658,469	$0	$658,469
Total development capital	**$'000**	**$3,952,556**	**$279,164**	**$4,231,720**
Operational capital				
Deferred stripping	$'000	n/a	n/a	$1,333,076
Closure costs	$'000	n/a	n/a	$71,696
Total operational capital	**$'000**	**n/a**	**n/a**	**$1,404,772**
Sustaining costs				
Mining	$'000	n/a	n/a	$2,404,574
Processing and infrastructure	$'000	n/a	n/a	$649,002
Total sustaining costs	**$'000**	**n/a**	**n/a**	$3,053,576
Total capital and sustaining costs	**$'000**	**n/a**	**n/a**	**$8,690,067**

The capital cost estimates developed by Lycopodium were stated to be accurate to +20% / -10%. Escalation was excluded from the Lycopodium estimate, as were all duties and taxes. All costs were presented in constant Q1 2025 US dollar terms.

Updated mine operating costs comprising drill, blast, load and haul costs were derived in Q3 2025 by FQM mining engineers. These derivations were estimated from first principles using productivity parameters for the proposed equipment fleet, simulated haul profiles related to the staged pit designs and production schedule, and from corresponding ore/waste haulage destinations.

Updated process operating costs for the Project were developed from first principles by FQM metallurgists and process engineers. Operating costs were determined for a process plant with an annual throughput of 40 Mt of ore at a P_{80} grind size of 180 µm, and assuming a 24 hour per day operation, for 365 days per year. In addition, G&A (general and administration) costs were estimated for administration and non-process infrastructure, including labour, energy and maintenance costs.

The estimated overall average unit operating costs (Stage 1 40 Mtpa) are:

- pre-strip mining ore and waste:
  - $1.84/t mined, average (range: $1.75/t to $3.01/t)
- operations mining of ore and waste (including stockpile reclaim):
  - $1.93/t mined, average (range: $0.97/t to $2.80/t)
- Processing:
  - $7.25/t processed, average
- G&A:
  - $ 1.52/t processed, processed
- water supply tariff and sundry taxes:
  - $0.05/t processed, average

For the 40 Mtpa Stage 1 cashflow model, several adjustments and indirect charges were subsequently included with the above mining, processing and G&A operating costs. The impact of these inclusions was assessed in a retrospective pit optimisation sensitivity analysis.

A preliminary 60 Mtpa cashflow model was produced for the Project economic analysis. This model accounted for $1,018.7 M of expansion expenditure primarily between Years 6 and 8. Preliminary 60 Mtpa operating cost estimates were also included for modelling purposes, along with scenario specific annual sustaining cost estimates.

The estimated overall average unit operating costs (Stage 2 60 Mtpa) are:

- operations mining of ore and waste (excluding stockpile reclaim):
  - $1.85/t mined, average
- processing:
  - $6.76/t processed, average
- G&A:
  - $1.40/t processed, average

1.17 Economic analysis

An economic analysis in the form of a basic cashflow model was produced to support the Mineral Reserve estimate, and to demonstrate an overall positive cashflow for mining and 40 Mtpa processing over the life of the Project. The initial Stage 1 development capital and sustaining costs are included in the analysis for completeness, and the model is summarised in Table 1-6.

The revenues in this cashflow model are calculated from the following metal prices based on consensus forecasts at Q3 2025:

- copper = $4.50/lb ($9,920/t)
- molybdenum = $18.00/lb ($39,680/t)
- gold = $3,000/oz

On a pre-tax basis over a fifty year project life, the Stage 1 Project is cashflow positive from Year 6 and payback on the initial development capital is in Year 8. (i.e., four years after Project commissioning). The undiscounted cashflow for the Mineral Reserve production schedule is $38,248.6 M, with an NPV equal to $7,362.4 M at an 8% discount rate and $5,138.4 M at a 10% discount rate. The internal rate of return is 23.3%.

After adopting notional depreciation schedules and tax assumptions, the estimated tax payable for the Project is $10,739.6 M. Under these circumstances, the NPV is equal to $4,691.4 M at an 8% discount rate and $3,065.5 M at a 10% discount rate. The internal rate of return is 18.4%. The Project remains cashflow positive from Year 6 and payback on the initial development capital is in Year 9.

The Stage 1 40 Mtpa C1 costs and AISC are $1.39/lb Cu and $1.74/lb Cu for the life of mine, respectively.

Table 1-6 Stage 1 40 Mtpa cashflow model summary

PHYSICALS	UNITS	TOTAL	Y1 - Y4	Y5 - Y9	Y10 -Y14	Y15 - Y19	Y20 - Y24	Y25 - Y29	Y30 - Y34	Y35 - Y39	Y40 - Y44	Y45 - Y49	Y50 - Y54	Y55
MINING (AFTER MINING DILUTION & RECOVERY)														
Ore mined direct to plant	Mt	**1,268.1**	0.0	156.3	191.5	179.9	159.1	168.0	179.5	179.3	54.6	0.0	0.0	0.0
Ore mined to stockpile	Mt	**722.0**	10.5	88.1	94.2	146.4	97.0	143.4	86.3	52.2	4.0	0.0	0.0	0.0
Ore reclaimed from stockpile	Mt	**722.0**	0.0	28.7	8.6	20.3	41.0	32.1	20.6	20.9	145.5	200.1	200.1	3.9
Waste mined to dump	Mt	**2,903.4**	422.1	732.3	691.0	454.7	291.5	181.4	88.6	37.9	4.0	0.0	0.0	0.0
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)														
Total direct feed	Mt	**1,268.1**	0.0	156.3	191.5	179.9	159.1	168.0	179.5	179.3	54.6	0.0	0.0	0.0
	% Cu	**0.56**	0.00	0.82	0.62	0.57	0.52	0.46	0.43	0.48	0.58	0.00	0.00	0.00
	ppm Mo	**136.89**	0.00	131.75	137.74	129.95	127.14	122.79	131.87	166.09	163.90	0.00	0.00	0.00
	g/t Au	**0.10**	0.00	0.14	0.13	0.12	0.10	0.08	0.07	0.08	0.08	0.00	0.00	0.00
Total reclaim feed	Mt	**722.0**	0.0	28.7	8.6	20.3	41.0	32.1	20.6	20.9	145.5	200.1	200.1	3.9
	% Cu	**0.19**	0.00	0.36	0.38	0.47	0.28	0.23	0.21	0.26	0.21	0.15	0.13	0.13
	ppm Mo	**98.05**	0.00	43.47	129.24	148.99	118.59	117.74	118.44	130.83	130.44	86.50	74.91	74.91
	g/t Au	**0.06**	0.00	0.07	0.12	0.14	0.09	0.09	0.09	0.08	0.08	0.04	0.03	0.03
Total plant feed	**Mt**	**1,990.1**	0.0	185.0	200.1	200.2	200.1	200.1	200.1	200.2	200.1	200.1	200.1	3.9
	% Cu	**0.42**	0.00	0.75	0.61	0.56	0.47	0.42	0.41	0.46	0.31	0.15	0.13	0.13
	ppm Mo	**122.80**	0.00	118.05	137.38	131.88	125.39	121.98	130.49	162.41	139.57	86.50	74.91	74.91
	g/t Au	**0.09**	0.00	0.13	0.13	0.13	0.10	0.08	0.08	0.08	0.08	0.04	0.03	0.03
Cu insitu	**kt**	**8,426.4**	0.0	1,388.5	1,228.8	1,113.2	936.5	848.5	812.4	920.9	620.2	292.1	260.1	5.1
Mo insitu	**kt**	**244.4**	0.0	21.8	27.5	26.4	25.1	24.4	26.1	32.5	27.9	17.3	15.0	0.3
Au insitu	**k(t)oz**	**5,480.2**	0.0	756.0	810.7	804.7	646.0	510.8	482.5	517.5	500.5	254.7	193.0	3.8
AVERAGE RECOVERIES														
Copper recovery	%	**87.1%**	0.0%	87.2%	87.1%	87.5%	86.5%	86.6%	88.1%	88.3%	87.0%	85.1%	84.8%	84.8%
Molybdenum recovery	%	**44.3%**	0.0%	40.8%	46.0%	45.6%	46.7%	44.5%	47.9%	43.2%	43.5%	43.9%	43.5%	43.5%
Gold recovery	%	**61.5%**	0.0%	60.2%	61.4%	61.3%	61.6%	61.3%	62.6%	61.0%	61.0%	66.1%	68.2%	68.2%
Ramp-up factor	*%*		*0.0%*	*97.3%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*
Adjusted copper recovery	%	**86.8%**	0.0%	84.9%	87.1%	87.5%	86.5%	86.6%	88.1%	88.3%	87.0%	85.1%	84.8%	84.8%
Adjusted Molybdenum recovery	%	**44.3%**	0.0%	36.1%	46.0%	45.6%	46.7%	44.5%	47.9%	43.2%	43.5%	43.9%	43.5%	43.5%
Adjusted gold recovery	%	**61.5%**	0.0%	58.6%	61.4%	61.3%	61.6%	61.3%	62.6%	61.0%	61.0%	66.1%	68.2%	68.2%
METAL RECOVERED														
Unadjusted Cu recovered	**kt**	**7,342.8**	**0.0**	**1,211.3**	**1,070.2**	**974.5**	**810.4**	**734.5**	**715.6**	**813.1**	**539.5**	**248.7**	**220.7**	**4.3**
Unadjusted Mo recovered	**kt**	**109.1**	**0.0**	**9.0**	**12.6**	**12.0**	**11.7**	**10.9**	**12.5**	**14.1**	**12.2**	**7.6**	**6.5**	**0.1**
Unadjusted Au recovered	**k(t)oz**	**3,380.8**	**0.0**	**455.3**	**497.5**	**493.6**	**397.9**	**313.1**	**301.8**	**315.7**	**305.1**	**166.7**	**131.7**	**2.6**
Adjusted Cu recovered	**kt**	**7,310.3**	**0.0**	**1,178.8**	**1,070.2**	**974.5**	**810.4**	**734.5**	**715.6**	**813.1**	**539.5**	**248.7**	**220.7**	**4.3**
Adjusted Mo recovered	**kt**	**108.3**	**0.0**	**8.2**	**12.6**	**12.0**	**11.7**	**10.9**	**12.5**	**14.1**	**12.2**	**7.6**	**6.5**	**0.1**
Adjusted Au recovered	**k(t)oz**	**3,369.6**	**0.0**	**444.0**	**497.5**	**493.6**	**397.9**	**313.1**	**301.8**	**315.7**	**305.1**	**166.7**	**131.7**	**2.6**
CONCENTRATE PRODUCED														
Cu concentrate	kt(wet)	**31,605.6**	0.0	5,094.3	4,554.4	4,126.2	3,568.0	3,200.2	3,119.5	3,357.4	2,358.4	1,167.5	1,039.4	20.4
Cu concentrate grade	%	**25.7%**	0.0%	25.7%	26.1%	26.2%	25.2%	25.5%	25.5%	26.9%	25.4%	23.7%	23.6%	23.6%
Mo concentrate	kt(wet)	**256.1**	0.0	19.4	29.9	28.4	27.6	25.7	29.5	33.2	28.8	18.0	15.4	0.3
Mo concentrate grade	%	**47.0%**	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%

CASHFLOWS	UNITS	TOTAL	Y1 - Y4	Y5 - Y9	Y10 -Y14	Y15 - Y19	Y20 - Y24	Y25 - Y29	Y30 - Y34	Y35 - Y39	Y40 - Y44	Y45 - Y49	Y50 - Y54	Y55
PAYABILITY														
Cu	%	**96.1%**	0.0%	96.1%	96.1%	96.2%	96.0%	96.1%	96.1%	96.3%	96.0%	95.8%	95.8%	95.8%
Mo	%	**86.0%**	0.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%
Au	%	**90.0%**	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
Payable metal recovered														
Cu	**kt**	**7,025.9**	0.0	1,133.0	1,029.2	937.4	778.3	705.7	687.5	782.8	518.3	238.2	211.3	4.1
Mo	**kt**	**93.2**	0.0	7.0	10.9	10.3	10.0	9.3	10.7	12.1	10.5	6.5	5.6	0.1
Au	**koz**	**3,032.6**	0.0	399.6	447.8	444.2	358.1	281.8	271.6	284.1	274.6	150.0	118.5	2.3
GROSS REVENUE														
Metal prices														
Cu	$/lb	**$4.50**	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50
Mo	$/lb	**$18.00**	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00
Au	$/oz	**$3,000**	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000
Revenue after payability														
copper revenue	**$M**	**$69,702.2**	$0.0	$11,239.8	$10,210.9	$9,299.8	$7,721.2	$7,000.8	$6,821.0	$7,766.4	$5,141.6	$2,363.1	$2,096.5	$41.1
molybdenum revenue	**$M**	**$3,697.5**	$0.0	$279.4	$431.2	$410.2	$398.5	$370.4	$425.6	$479.9	$416.0	$259.4	$222.5	$4.4
gold revenue	**$M**	**$9,097.8**	$0.0	$1,198.9	$1,343.3	$1,332.7	$1,074.2	$845.3	$814.8	$852.3	$823.8	$450.1	$355.6	$7.0
Total revenue	**$M**	**$82,497.6**	**$0.0**	**$12,718.1**	**$11,985.4**	**$11,042.6**	**$9,194.0**	**$8,216.5**	**$8,061.4**	**$9,098.6**	**$6,381.4**	**$3,072.6**	**$2,674.5**	**$52.4**
METAL COSTS														
TCRCs	$M	**$8,020.5**	$0.0	$1,206.2	$1,097.8	$1,007.9	$883.2	$806.7	$790.3	$849.2	$629.9	$373.4	$346.1	$29.7
Royalties	$M	**$1,117.2**	$0.0	$172.7	$163.3	$150.5	$124.7	$111.1	$109.1	$123.7	$86.3	$40.5	$34.9	$0.3
Total metal costs	**$M**	**$9,137.6**	**$0.0**	**$1,378.9**	**$1,261.1**	**$1,158.5**	**$1,007.9**	**$917.8**	**$899.4**	**$972.9**	**$716.2**	**$413.9**	**$381.0**	**$30.1**
CAPITAL COSTS														
Mining	$M	**$1,756.0**	$1,658.3	$97.6										
Processing	$M	**$1,077.5**	$960.2	$117.3										
Rail	$M	**$12.8**	$11.4	$1.4										
Infrastructure	$M	**$727.0**	$647.9	$79.1										
Indirects	$M	**$658.5**	$586.8	$71.7										
Subtotal development capital	$M	**$4,231.7**	**$3,864.6**	**$367.1**										
Deferred stripping	$M	**$1,333.1**		$667.9	$506.3	$140.6	$18.3							
Closure costs	$M	**$71.7**												$71.7
Total capital costs	**$M**	**$5,636.5**	**$3,864.6**	**$1,035.0**	**$506.3**	**$140.6**	**$18.3**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$71.7**
SUSTAINING COSTS														
Mining	$M	**$2,404.6**	$0.0	$424.7	$466.3	$358.2	$351.6	$298.6	$145.4	$111.0	$64.0	$123.2	$61.8	$0.0
Processing and infrastructure	$M	**$649.1**	$0.0	$62.6	$65.0	$65.0	$65.0	$65.0	$65.0	$65.0	$65.0	$65.0	$65.0	$1.6
Total sustaining costs	**$M**	**$3,053.7**	**$0.0**	**$487.2**	**$531.3**	**$423.2**	**$416.6**	**$363.5**	**$210.4**	**$176.0**	**$129.0**	**$188.2**	**$126.8**	**$1.6**
OPERATING COSTS														
Mining	$M	**$8,716.7**	$44.2	$1,111.3	$1,318.3	$1,367.7	$1,173.2	$1,178.6	$933.5	$744.8	$345.0	$256.9	$243.3	$0.0
Processing	$M	**$14,425.2**	$0.0	$1,390.5	$1,444.4	$1,444.7	$1,444.2	$1,444.2	$1,444.4	$1,444.7	$1,444.2	$1,444.2	$1,444.4	$35.0
General and administration	$M	**$3,015.6**	$0.0	$321.7	$331.7	$327.6	$318.8	$314.3	$313.6	$318.6	$277.6	$243.5	$241.6	$6.4
Other	$M	**$101.3**	$7.4	$9.2	$9.2	$9.2	$9.2	$9.2	$9.2	$9.2	$9.2	$9.2	$9.2	$1.8
Total operating costs	**$M**	**$26,258.7**	**$51.5**	**$2,832.8**	**$3,103.6**	**$3,149.2**	**$2,945.4**	**$2,946.4**	**$2,700.8**	**$2,517.4**	**$2,076.1**	**$1,953.8**	**$1,938.6**	**$43.3**
OTHER COSTS														
Taxes on bank transactions	$M	**$149.3**	**$23.6**	**$17.2**	**$16.7**	**$15.0**	**$13.7**	**$13.4**	**$11.9**	**$11.1**	**$9.1**	**$8.7**	**$8.5**	**$0.4**
MINERAL RESERVE CASHFLOW (PRE-TAX)														
Undiscounted cashflow	**$M**	**$38,248.6**	-$3,951.5	$6,595.4	$6,666.0	$6,108.8	$4,842.6	$3,981.8	$4,258.2	$5,441.8	$3,516.3	$537.8	$223.3	$27.9
NPV_8 (indicative)	**$M**	**$7,362.4**												
NPV_{10} (indicative)	**$M**	**$5,138.4**												
IRR	**%**	**23.3%**												
MINERAL RESERVE CASHFLOW (POST-TAX)														
Taxable income	**$M**	**$35,982.2**	$0.0	$4,993.7	$5,962.2	$5,277.6	$4,074.0	$3,356.5	$3,677.4	$4,890.1	$3,172.8	$413.4	$164.5	$0.0
Tax paid	**$M**	**-$10,739.6**	$0.0	-$1,546.7	-$1,768.6	-$1,572.0	-$1,219.2	-$1,012.9	-$1,088.4	-$1,420.3	-$915.7	-$134.8	-$61.5	$0.6
Undiscounted cashflow	**$M**	**$27,509.0**	-$3,951.5	$5,048.8	$4,897.4	$4,536.8	$3,623.4	$2,968.9	$3,169.8	$4,021.5	$2,600.6	$403.0	$161.9	$28.5
NPV_8 (indicative)	**$M**	**$4,691.4**												
NPV_{10} (indicative)	**$M**	**$3,065.5**												
IRR	**%**	**18.4%**												

A basic sensitivity analysis on the Stage 1 40 Mtpa cashflow model, post-tax, indicates:

- copper metal price and copper recovery are confirmed to be the most sensitive variables
- a 10% increase in copper prices would increase the NPV by 26% to $5,926 M and the IRR to 20.7% on a post tax basis
- conversely, a 10% decrease would reduce the NPV by 26% to $3,453 M and the IRR to 16% on a post tax basis
- the next most sensitive cashflow model item is the magnitude of the development capital costs; a 10% increase in which would reduce the NPV by 8% and the IRR by 8%
- processing and mine operating costs follow next and then gold metal price and gold recovery
- the least sensitive variables are the molybdenum price and recovery, TCRCs and G&A operating costs

The analysis was repeated for the Stage 2 60 Mtpa cashflow model, post-tax, and showed similar trends.

Relative to the inputs for the original pit optimisation, the Stage 1 40 Mtpa cashflow model (pre-tax) inclusive of higher metal prices and operating costs, yields positive annual cashflows throughout the life of the operation.

An additional cashflow model was produced for the Stage 2 60 Mtpa scenario. This model adopted the preliminary Stage 2 60 Mtpa production schedule, in addition to preliminary operating cost estimates, an estimated capital cost for expansion from the Stage 1 40 Mtpa capacity, and the estimated corresponding sustaining costs. The revenues were calculated from the same above listed metal prices. This model is summarised in Table 1-7.

On a pre-tax basis over a thirty five year life, the Stage 2 60 Mtpa Project remains cashflow positive throughput the expansion capital spend. The payback year on the incremental positive cashflows is from the fourth year (Year 12) following the expansion. The undiscounted cashflow for the shortened Mineral Reserve production timeframe is $39,479.7 M, with an NPV of $9,087.3 M at an 8% discount rate and an NPV of $6,433.2 M at a 10% discount rate. The internal rate of return is 23.3%.

After adopting notional depreciation schedules and tax assumptions, the estimated tax payable for the Project is $11,072.2 M. Under these circumstances, the NPV is $5,917.1 M at an 8% discount rate or $3,973.3 M at a 10% discount rate. The internal rate of return is 19.3%.

The Stage 2 60 Mtpa C1 costs and AISC are $1.26/lb Cu and $1.60/lb Cu for the life of mine, respectively.

1.18 Environmental and social summary

Detailed environmental baseline data collection began in 2016. A Project alternatives analysis and a separate Project description document were completed in 2018 to complement the Project ESIA and the ESIA was submitted to the authorities in February 2019.

The main permit required for the development of the Project is the ESIA, which must be approved by the Mining Secretariat of the Province of Salta. This report covers the main sites of the Project, including the mine, the processing plant, tailings dam, waste rock dump, and associated facilities. After the submission of this ESIA, the Secretariat made observations, which were addressed in 2020. Subsequently, additional studies on tailings and waste rock management were required, which were completed and submitted in August 2022.

In June 2023, a second round of observations was received, which were addressed in October 2023, including updates regarding water use during the construction and operation phases and updates to the social baseline. During 2024, progress was made on the evaluation process, which involved the intervention of SEGEMAR, who conducted a review of the ESIA. Additionally, together with the Secretariats of Mining and Energy (SME), Water Resources (SRH), and Environment of Salta (SA), a workshop and a visit to the Project were held.

Table 1-7 Stage 2 60 Mtpa cashflow model summary

PHYSICALS	UNITS	TOTAL	Y1 - Y4	Y5 - Y9	Y10 - Y14	Y15 - Y19	Y20 - Y24	Y25 - Y29	Y30 - Y34	Y35 - Y39	Y40 - Y44
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)											
	Mt	1,990.1	0.0	205.0	300.2	300.3	300.2	300.2	300.2	284.1	0.0
	% Cu	0.42	0.00	0.69	0.59	0.47	0.45	0.45	0.25	0.14	0.00
	ppm Mo	123.41	0.00	120.86	144.34	101.79	136.03	158.78	119.64	79.24	0.00
	g/t Au	0.09	0.00	0.13	0.13	0.10	0.08	0.08	0.06	0.04	0.00
Cu insitu	kt	8,450.0	0.0	1,420.5	1,777.0	1,423.6	1,344.7	1,356.8	738.9	388.4	0.0
Mo insitu	kt	245.6	0.0	24.8	43.3	30.6	40.8	47.7	35.9	22.5	0.0
Au insitu	k(t)oz	5,521.8	0.0	835.7	1,235.2	950.1	812.5	745.0	598.2	345.0	0.0
AVERAGE RECOVERIES											
Copper recovery	%	87.1%	0.0%	87.1%	87.2%	85.9%	87.5%	88.4%	85.0%	84.8%	0.0%
Molybdenum recovery	%	44.3%	0.0%	41.3%	45.5%	45.5%	46.4%	44.1%	44.9%	40.3%	0.0%
Gold recovery	%	61.5%	0.0%	60.4%	61.3%	61.1%	61.7%	61.3%	63.4%	62.4%	0.0%
Ramp-up factor	*%*		*0.0%*	*97.4%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*0.0%*
Adjusted copper recovery	%	86.6%	0.0%	84.8%	87.2%	85.9%	87.5%	88.4%	85.0%	84.8%	0.0%
Adjusted Molybdenum recovery	%	44.3%	0.0%	36.6%	45.5%	45.5%	46.4%	44.1%	44.9%	40.3%	0.0%
Adjusted gold recovery	%	61.4%	0.0%	58.8%	61.3%	61.1%	61.7%	61.3%	63.4%	62.4%	0.0%
METAL RECOVERED											
Unadjusted Cu recovered	kt	7,358.6	0.0	1,237.5	1,549.2	1,223.5	1,176.1	1,199.4	627.8	345.1	0.0
Unadjusted Mo recovered	kt	109.2	0.0	10.3	19.7	13.9	19.0	21.0	16.2	9.0	0.0
Unadjusted Au recovered	k(t)oz	3,386.7	0.0	504.8	757.8	580.3	501.5	456.6	379.0	206.6	0.0
Adjusted Cu recovered	kt	7,310.3	0.0	1,205.0	1,549.2	1,223.5	1,176.1	1,199.4	627.8	329.4	0.0
Adjusted Mo recovered	kt	108.3	0.0	9.6	19.7	13.9	19.0	21.0	16.2	9.0	0.0
Adjusted Au recovered	k(t)oz	3,369.6	0.0	493.6	757.8	580.3	501.5	456.6	379.0	200.7	0.0
CONCENTRATE PRODUCED											
Cu concentrate	kt(wet)	31,605.6	0.0	5,259.8	6,557.0	5,417.9	5,087.8	4,990.3	2,777.2	1,515.6	0.0
Cu concentrate grade	%	25.7%	0.0%	25.7%	26.1%	26.2%	25.2%	25.5%	25.5%	26.9%	0.0%
Mo concentrate	kt(wet)	256.1	0.0	22.6	46.7	32.8	44.8	49.7	38.2	21.4	0.0
Mo concentrate grade	%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%

CASHFLOWS	UNITS	TOTAL	Y1 - Y4	Y5 - Y9	Y10 - Y14	Y15 - Y19	Y20 - Y24	Y25 - Y29	Y30 - Y34	Y35 - Y39	Y40 - Y44
PAYABILITY											
Cu	%	96.1%	0.0%	96.1%	96.2%	96.0%	96.1%	96.2%	95.9%	95.8%	0.0%
Mo	%	86.0%	0.0%	68.8%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	0.0%
Au	%	90.0%	0.0%	72.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	0.0%
Payable metal recovered											
Cu	kt	7,009.9	0.0	1,157.7	1,490.2	1,174.7	1,130.3	1,154.5	602.8	299.9	0.0
Mo	kt	92.8	0.0	8.2	17.0	11.9	16.3	18.1	13.9	7.4	0.0
Au	koz	2,975.0	0.0	409.6	682.1	522.3	451.3	411.0	341.1	157.7	0.0
GROSS REVENUE											
Metal prices											
Cu	$/lb	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50
Mo	$/lb	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00
Au	$/oz	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000
Revenue after payability											
copper revenue	$M	$69,697.3	$0.0	$11,484.9	$14,783.9	$11,653.6	$11,213.0	$11,453.1	$5,980.1	$3,128.6	$0.0
molybdenum revenue	$M	$3,697.5	$0.0	$326.4	$673.9	$473.8	$647.8	$717.4	$551.8	$306.4	$0.0
gold revenue	$M	$9,097.8	$0.0	$1,332.7	$2,046.2	$1,566.9	$1,354.0	$1,232.9	$1,023.4	$541.9	$0.0
Total revenue	$M	$82,492.6	$0.0	$13,143.9	$17,504.0	$13,694.3	$13,214.7	$13,403.4	$7,555.3	$3,976.9	$0.0
METAL COSTS											
TCRCs	$M	$7,622.6	$0.0	$1,239.7	$1,527.7	$1,273.0	$1,210.2	$1,199.1	$718.9	$454.1	$0.0
Royalties	$M	$1,123.1	$0.0	$178.6	$239.6	$186.3	$180.1	$183.1	$102.5	$52.8	$0.0
Total metal costs	$M	$8,745.6	$0.0	$1,418.2	$1,767.3	$1,459.3	$1,390.3	$1,382.1	$821.4	$506.9	$0.0
CAPITAL COSTS											
Mining	$M	$1,861.2	$1,658.3	$202.8							
Processing	$M	$1,606.2	$960.2	$646.0							
Rail	$M	$12.8	$11.4	$1.4							
Infrastructure	$M	$945.8	$647.9	$297.9							
Indirects	$M	$824.4	$586.8	$237.6							
Subtotal development capital	$M	$5,250.4	$3,864.6	$1,385.8							
Deferred stripping	$M	$1,243.9		$825.3	$193.6	$225.0					
Closure costs	$M	$71.7								$21.2	$50.5
Total capital costs	$M	$6,566.0	$3,864.6	$2,211.1	$193.6	$225.0	$0.0	$0.0	$0.0	$21.2	$50.5
SUSTAINING COSTS											
Mining	$M	$2,267.3	$0.0	$450.3	$554.1	$473.8	$436.5	$209.1	$87.8	$55.6	$0.0
Processing and infrastructure	$M	$605.6	$0.0	$69.5	$90.0	$90.0	$89.9	$89.9	$90.0	$86.3	$0.0
Total sustaining costs	$M	$2,872.9	$0.0	$519.8	$644.1	$563.8	$526.4	$299.1	$177.8	$141.9	$0.0
OPERATING COSTS											
Mining	$M	$8,356.1	$44.2	$1,102.0	$1,951.4	$1,811.0	$1,564.4	$1,183.9	$401.0	$298.2	$0.0
Processing	$M	$13,457.5	$0.0	$1,544.5	$1,999.2	$1,999.6	$1,998.8	$1,998.8	$1,999.2	$1,917.4	$0.0
General and administration	$M	$2,777.8	$0.0	$349.9	$452.7	$435.0	$432.8	$433.8	$357.2	$316.4	$0.0
Other	$M	$92.4	$7.4	$10.5	$12.4	$12.4	$12.4	$12.4	$12.4	$12.4	$0.0
Total operating costs	$M	$24,683.7	$51.5	$3,006.9	$4,415.6	$4,258.0	$4,008.5	$3,629.0	$2,769.9	$2,544.4	$0.0
OTHER COSTS											
Taxes on bank transactions	$M	$144.7	$23.6	$21.8	$21.2	$20.5	$18.4	$16.1	$12.1	$11.1	$0.0
MINERAL RESERVE CASHFLOW (PRE-TAX)											
Undiscounted cashflow	$M	$39,479.7	-$3,951.5	$5,532.2	$10,419.8	$7,234.1	$7,309.9	$8,056.7	$3,980.4	$787.3	$110.8
NPV$_{10}$ (indicative)	$M	$6,433.2									
NPV$_{8}$ (indicative)	$M	$9,087.3									
IRR	%	23.3%									
MINERAL RESERVE CASHFLOW (POST-TAX)											
Taxable income	$M	$37,129.3	$0.0	$4,518.5	$8,946.2	$6,108.7	$6,266.7	$7,188.8	$3,452.7	$647.8	$0.0
Tax paid	$M	-$11,072.2	$0.0	-$1,434.3	-$2,649.0	-$1,833.0	-$1,859.5	-$2,095.2	-$999.4	-$201.9	$0.0
Undiscounted cashflow	$M	$28,407.5	-$3,951.5	$4,097.9	$7,770.8	$5,401.1	$5,450.4	$5,961.5	$2,981.0	$585.5	$110.8
NPV$_{10}$ (indicative)	$M	$3,973.3									
NPV$_{8}$ (indicative)	$M	$5,917.1									
IRR	%	19.3%									

Additional and separate ESIAs were submitted in September 2020 and November 2022 for the proposed access road and powerline, respectively.

These submissions remain under review by the relevant authorities.

1.19 Conclusions and recommendations

1.19.1 Mineral Resource modelling and estimation

The Mineral Resource estimate was completed using validated data and appropriate, industry-standard estimation methods. Estimation domains were based on the existing three-dimensional geological model, with a refined interpretation of the base of the leach cap incorporated for this update. Lithology, alteration, and mineralisation style interpretations remain unchanged.

Geometallurgical domains were refined using iron and sulphur content, in addition to copper grade, to support the application of variable metallurgical recovery assumptions within the constraining life of mine pit shell.

While the estimates demonstrate good validation at the deposit scale, drill hole spacing in some areas does not support locally reliable estimates at a selective mining unit scale. The current Mineral Resource estimate has negligible change when compared to the previous estimate and benefits from improved definition of the leached horizon and geometallurgical domains relevant to metallurgical processing.

Estimate confidences support Measured and Indicated Mineral Resources for conversion into Proven and Probable Mineral Reserves with 80% of Mineral Resources classified as Indicated.

1.19.2 Mine planning and Mineral Reserve estimation

A conventional approach has been adopted in the process of optimising a mine planning model derived from the Mineral Resource model, to be followed by detailed stage and ultimate open pit designs, production scheduling and Mineral Reserve estimation. The Mineral Reserve has considered appropriate modifying factors and reflects an achievable mining plan and production schedule for the Project, at this stage of evaluation.

The pit optimisation operating cost inputs were based on the Stage 2 60 Mtpa production scenario with subsequent mine planning, production scheduling and economic analysis adopting the Stage 1 40 Mtpa scenario for project development considerations.

An optimisation sensitivity analysis was completed to test the compounded impact of increased metal prices, together with the increased operating costs associated with a 40 Mtpa operation rather than 60 Mtpa. The overall optimisation impact, with higher metal prices somewhat compensating for the higher operating costs, was a realisation of no significant impact to the ultimate pit design. The 40 Mtpa cashflow model, which includes the higher prices and operating costs, yields a positive annual cashflow throughout the life of the operation.

Further mine geotechnical drilling and analysis is required to support the design of a deep pit (+700 m) with emphasis on the eastern wall adjoining the Salar de Arizaro. A recommended programme of further drilling and the specific location of drillholes has been devised by the Company’s mining geotechnical specialist. It is recommended that the geotechnical investigations be integrated with ongoing hydrogeological modelling, for the sake of analysing drained pit slope conditions. This programme of work is planned to commence in Q2 2026.

1.19.3 Metallurgy

Metallurgical testwork undertaken by Lumina indicated that a plant designed on the basis of a conventional porphyry copper flowsheet, as used throughout South America, would give good recoveries of copper and molybdenum to commercial grade flotation concentrates.

The majority of the early testwork was performed in tap water, but some work indicated that the use of the brine from the salar adjacent to the site would be possible for the milling and rougher flotation circuits, but not for cleaner flotation.

A drilling and metallurgical testwork programme was completed in 2019 and 2020 to define recoveries and concentrate grades for material to be mined in the early years of the operation and using water available on site.

Testwork samples were obtained from four drill holes, with core from each hole composited by depth to provide ten metallurgical samples. Water for the testwork was brine from the Salar de Arizaro, and brackish water from bores at Valle de las Burras (TW-10) and Valle de Arizaro (T-22), typical of the quality that would be used for water supply for the Project.

This work enabled the Company to better define several areas of the process design, as follows:

- defined the primary grind size of 80% passing 180 µm and confirmed mill sizing
- confirmed flotation residence times, and reagent requirements
- confirmed recoveries and concentrate grades in locked cycle flotation tests
- confirmed concentrate regrind size requirements of 80% passing 20 to 30 µm
- defined dewatering requirements for tailings and concentrates

Average recoveries over the mine life are expected to be:

- 87.1% Cu recovery to a concentrate grade of 25.7% Cu
- 44.3% Mo recovery to a concentrate of 47% Mo
- 61.4% Au recovery at a grade of 4.5 g/t in the copper concentrate

These recovery projections are marginally higher than those mentioned in the 2021 Technical Report.

1.19.4 Processing

The processing flowsheet follows that of typical porphyry copper concentrators common in Chile and Peru.

After the commissioning year, the processing plant will have a Stage 1 capacity of 40 Mtpa. The Stage 1 40 Mtpa plant will comprise two milling trains, two rougher flotation trains each containing one row of seven cells, and two cleaner flotation circuits.

Process design and equipment sizing has followed standard practice, enhanced through experience gained by the Company in the design, construction, and operation of other similar sized projects.

The metallurgical testwork programme confirmed the design criteria for the milling and dewatering equipment required for the treatment of ores generated from the starter pit, and in the presence of brine solutions.

There is fresh water available from four water basins and proposed borefields, sufficient for the Stage 1 40 Mtpa processing. For the Stage 2 expansion to 60 Mtpa throughput, additional fresh water supply of an estimated 415 m^3/ hr is required to be sourced. It is assumed that additional water can be sourced from either efficiencies in Stage 1, from the four borefields once actual operational experience is obtained or from the further afield regional water supply basins.

A direct benefit of the change to milling and flotation in fresh water is the higher recoveries and concentrate grades.

As the engineering phase proceeds, refinements to the processing flowsheet and design will be considered. Further work will be carried out in conjunction with mine planning, to review the plant ramp-up profile and duration, and to further assess the potential recovery of gold sourced from the leach cap.

1.19.5 Water supply

Regional borefields will be developed to supply fresh water for the Project. Field investigations concluded in 2025 from which analyses are in progress to assess the total brine extraction potential from the Salar de Arizaro and Salar de Taca Taca. There are four identified fresh water supply basins, the combined abstraction from which is currently determined to be adequate for a 40 Mtpa operation. Additionally, there are several additional fresh water supply sources being evaluated to support a 60 Mtpa operation.

The Stage 1 project at 40 Mtpa, including processing, mining and the camp, is expected to consume:

- an average of 2,130 m^3/hr or 51,120 m^3/day of raw water, and
- an average of 600 m^3/hr or 14,400 m^3/day of brine

For a production rate expanded to 60 Mtpa, the respective average consumption rates would be 3,190 m^3/hr or 76,560 m^3/day of fresh water and 846 m^3/hr or 20,400 m^3/day of brine.

Current fresh water sources covered by the ESIA application and pit dewatering should provide a net average supply of 2,856 m^3/hr. Additional regional sources would increase average supply to a projected 3,792 m^3/hr.

The currently projected rate of brine supply from various sources is an average of over 1,000 m^3/hr. Investigations in late 2025 indicated that this rate of supply may be obtainable solely from borefields in the adjacent salars. Hydrogeological investigations and modelling are continuing in this regard.

1.19.6 Cost estimation and economic outcomes

The Stage 1 40 Mtpa process plant and related infrastructure capital costs have been determined by means of detailed bottom-up estimates, supplemented with consultant/vendor estimates. This is a culmination of collaborative engineering design effort between FQM and consultants Lycopodium, Fluor and Process E&I.

The capital cost estimates compiled by Lycopodium are stated to be accurate to +20% / -10%. Escalation was excluded from the estimate, as were all duties and taxes. All costs were presented in constant Q1 2025 dollars.

Mine and process operating costs, plus general and administration costs (G&A) have been estimated from first principles by FQM engineers and metallurgists. Metal costs, including treatment and refining charges (TCRCs), have been advised by the Company's metals marketing specialists.

The Mineral Reserve estimate is supported by a pre-tax and post-tax cashflow model, the basis of which is the Stage 1 40 Mtpa life of mine production schedule and inclusive of the estimated development/sustaining capital and operating costs, metal costs, recovered metal payability, royalties, depreciation and corporate tax. Several sensitivity analyses have been undertaken on the 40 Mtpa scenario; modelling of a Stage 2 60 Mtpa cashflow scenario has also been undertaken.

1.19.7 Further work recommendations

Water supply investigations

At this stage of engineering, there is estimated to be sufficient fresh water supply for processing at Stage 1 40 Mtpa throughput. This supply relates to four regional borefields surrounding the Project site. Field investigations and further hydrogeological modelling and analysis is continuing, in order to evaluate the sustainability and viability of the supplemental supply required to support a Stage 2 60 Mtpa throughput.

Brine abstraction investigations were concluded in 2025, during which significant transmissive layers were identified in both the Salar de Arizaro and the Salar de Taca Taca, with the potential for meeting the required pumping rates under operational conditions. Two brine pumping wells of nearly 300 m in depth were constructed in the Salar de Taca Taca. A full brine water hydrogeological model is currently being developed.

Geology and Mineral Resource

80% of Taca Taca Mineral Resource is classed as Indicated due to a relatively wide grid spacing. A staggered 75 m by 150 m drill grid spacing would support upgrading Indicated to Measured categories from improved definition of geology and mineralisation for increased estimate accuracy. Prioritised drilling is recommended as follows:

1. Starter Pit:

   A 75 m by 150 m drill programme across the first two years of inpit ore mining is recommended to support detailed mine planning. Mineralisation in this area is high grade within a supergene chalcocite-rich horizon directly below the leached cap. Ensuring confidence in the relative positions, volumes and grades of this mineralisation will improve the ability to deliver feed according to plan. Results would improve estimate confidence of early ore feed for mine planning and would support future drilling requirements, such as grade control drill grid spacings.

2. TK2 Fault:

   Areas proximal and west of the deposit-scale TK2 (West) fault are considered a high priority for further definition. Drilling would investigate the risk of disruptions (losses) to mineralisation continuity across the fault zone, whilst also defining the extents of shallower ore typical to this area. Geotechnical data would also be collected to determine the impact of the fault zone on pit slope stability and hydrogeology.

Extensional drilling is recommended for mineralised areas external to the current life of mine pit shell. Mineralisation is open to the south and east of the deposit, as well as, at depth.

In addition, the following geological work is recommended:

1. Only ~9% of samples currently include sequential copper analysis, limiting copper species domaining within the Mineral Resource estimate and increasing uncertainty in domain boundaries and local estimation selectivity. Future infill drilling should systematically include sequential copper (with pyrite and metallurgical testwork integration) to improve domaining, reduce internal mixing, and strengthen recovery and production forecasting confidence.
2. 30% of samples were drilled using reverse circulation methods with the remainder been diamond core samples. The potential for bias between these two methods sample assays needs to be assessed and managed.
3. A 3D structural model is required to be compiled from the integration of multiple data sets. A recent high-resolution ground-magnetic geophysical survey should be interpreted by a qualified geophysicist and structural geologist. The results can be compared with topographical surveys, surface mapping, drill core logging, and geo-chemical modelling of multi-element assay data. Abundant evidence for

faulting in drill-core suggests a 3D model will be important for predicting local changes in weathering profiles and potential disruptions to mineralisation continuity at a mining scale. It will also contribute to geotechnical and hydrogeological modelling.

4. A 3D alteration model should be created, focussing on relative pyrite abundance and vein type/intensity alongside the delineation of broad gangue mineralogy. Geochemical interpretation of existing multi-element assay and SWIR (short-wave infrared) data combined with visual drill core logging and validation would provide primary inputs. Preliminary groundwork was completed by Scott Halley in 2019 but incorporation into the block model would benefit planning, operations, and processing.
5. Predicted work index values should be compared to the alteration model and historic point load data with a view to delineating zones of variable comminution properties.
6. Geometallurgical recoveries were developed and updated according to a historical and recent test work. Results were linked to rock type and key multi-element and sequential copper data. Additional test work and improved geological definition will increase the accuracy of these geometallurgical variables.
7. Improved resolution on rock strength and rock quality domains focussed on weathered and supergene zones will benefit pit design. Variations in rock strengths and quality will have implications for slope stability, blasting, mining methods, and comminution.

Geotechnical engineering

Further mine geotechnical drilling and analysis is required to support the design of a deep pit (+700 m) with emphasis on the eastern wall adjoining the Salar de Arizaro. A recommended programme of further drilling and the specific location of drillholes has been provided by the Company's mine geotechnical specialist. The geotechnical investigations are to be integrated with ongoing hydrogeological modelling, with the objective of assessing the requirements for preferred drained pit slope conditions. The planned month for commencing this work is April 2026.

A limited civil geotechnical programme of shallow drill holes and penetrometer testing has been carried out across the Project area, but only on the salar surface, and in consideration of bearing pressures for the TSF embankment, the waste dump, rail load-out and the airstrip. Civil geotechnical work will be required across the wider Project site, particularly at the plant location.

The Project site is located within a seismically active region and as such further work is recommended on the specification of design criteria for infrastructure elements.

Metallurgy and process engineering

Further testwork requirements can be summarised as follows:

- Cu-Mo separation testwork to define equipment sizing and reagent requirements
- optimisation of flotation reagent requirements – particularly frother (and the type of frother)
- some additional work on the dewatering of rougher concentrates, and optimum flocculant addition (type and quantity)
- confirmation of recoveries in fresh/brackish water for as many of the metallurgical domains and ore types as possible, particularly those representing the first five years of process plant feed.
- gold recovery testwork from the leach cap (longer term testwork)

None of this work is critical for the current plant design.

Trade-off studies are required to define the process designs that will be undertaken as the Project engineering phase proceeds and will include:

- an assessment of the regrind circuit power requirements and mill sizing
- evaluation of the economics of producing a molybdenum concentrate
- a re-look at the design of the molybdenum flotation circuit in light of experience gained from Cobre Panamá (when that circuit commences operation)
- a more detailed investigation of how to discharge slurry to the TSF and how to reclaim decant water, if feasible
- a review of process building requirements and equipment design to cater for the climatic conditions (wind)

ITEM 2 INTRODUCTION

2.1 Purpose of this Technical Report

This Technical Report on the Taca Taca Project (the property) has been prepared by Qualified Persons (QPs) David Gray, Michael Lawlor and Andrew Briggs of First Quantum Minerals Pty Ltd (FQM, the issuer or the Company).

The purpose of this Technical Report is to document updated Mineral Resource and Mineral Reserve estimates for the property, completed since the last filing of a Technical Report in March 2021 (FQM, 2021). Additional information contained herein is a summary of those technical aspects of the 2021 report which have been revised, updated or newly added.

2.2 Terms of reference

This Technical Report has been written to comply with the reporting requirements of the Canadian Securities Administrators' National Instrument 43-101 'Standards of Disclosure for Mineral Projects' and 'Form 43-101F1 Technical Report' (NI 43-101 or the Instrument, 2011).

The effective date for the Mineral Resource and Mineral Reserve estimates is the 31st of December 2025.

2.3 Qualified Persons and authors

The Mineral Resource estimate was prepared under the direction and supervision of David Gray. Mr Gray of FQM meets the requirements of a Qualified Person (QP) according to his Certificate of Qualified Person attached in Item 28.

The Mineral Reserve estimate was prepared under the direction of Michael Lawlor, with the assistance of FQM staff. Mr Lawlor of FQM meets the requirements of a QP according to his Certificate of Qualified Person attached in Item 28. Mr Lawlor takes responsibility for those items not addressed specifically by the other QPs.

Metallurgical testing, mineral processing and process recovery aspects of this Technical Report were addressed by Andrew Briggs. Mr Briggs of FQM meets the requirements of a QP according to his Certificate of Qualified Person attached in Item 28.

The following table identifies which items of the Technical Report have been the responsibility of each QP.

Table 2-1 QP details

Name	Position	NI 43-101 Responsibility
David Gray BSc Hons (Geology), FAIG, MAusIMM	**Group Manager, Mine and Resource Geology,** FQM (Australia) Pty Ltd	**Author and Qualified Person** Items 7 – 12, 14, 25 and 26
Michael Lawlor BEng Hons (Mining), MEngSc, FAusIMM	**Mining Technical Advisor** FQM (Australia) Pty Ltd	**Author and Qualified Person** Items 1 - 6, 15 and 16, 18 to 20, 21 in respect of mine operating and metal costs, and 22 to 26
Andrew Briggs BSc (Eng), ARSM, FSAIMM	**Group Consulting Metallurgist** FQM (Australia) Pty Ltd	**Author and Qualified Person** Items 13, 17, 21 in respect of process operating and G&A costs, 25 and 26

2.4 Principal sources of information

Information used in compiling this Technical Report was derived from previous technical reports on the property, and from the reports and documents listed in the References item (Item 27).

2.5 Site visits

Site visits by each of the QPs were as follows:

- David Gray visited the Project in October 2018 and March 2019. Mr Gray inspected drill core and drilling sites, reviewed geological data collection and sample preparation procedures, and carried out independent data verification. He also visited all accessible areas of the site.
- Michael Lawlor visited the Project in July 2016. Mr Lawlor visited all accessible areas of the site.
- Andrew Briggs first visited the Project in February 2012, when he reviewed all documentation provided by the previous owners. He also visited the Project in September 2018. Mr Briggs inspected drill core and drilling sites, reviewed metallurgical data collection and sample preparation procedures, and carried out independent data verification. He also visited all accessible areas of the site, including the proposed site for the TSF.

Since the 2012 to 2019 site visits, there have been no material changes at the Project site nor developments that would materially impact the information or conclusions presented herein. In the opinion of the QPs therefore, further site visits have been deemed as unnecessary for the purposes of this Technical Report update.

2.6 Conventions and definitions

Reference in this Technical Report to dollars or $, relates to United States dollars. Copper and molybdenum metal production is reported in (metric) tonnes and (imperial) pounds, where the conversion factor is 1 tonne (t) = 2,204.62 pounds (lb). Gold production is reported in (troy) ounces and with an adopted abbreviation of (t)oz.

The conventional chemical abbreviation for copper of Cu is used throughout this report, whilst the abbreviation for molybdenum is Mo and for gold is Au. ASCu is used to denote Acid Soluble Copper and TCu is used to denote Total Copper.

Unless otherwise referenced in the captions, drawings and diagrams in this report were sourced from within the Company.

Where not explained in the text of this report, specific terms and definitions are as listed in Table 2-2.

Table 2-2 Terms and definitions

Term	Definition
µm, mm, cm, m, km	microns, millimetres, centimetres, metres, kilometres
ABA	acid base accounting
ADIFSE	Administración de Infraestructuras Ferroviarias Sociedad del Estado
Ag	silver
AGP	Atacama Gas Pipeline
Am^3/h	actual cubic metres per hour (volumetric flow rate unit for gases)
AN	ammonium nitrate
ANCOLD	Australian National Committee on Large Dams
ANFO	ammonium nitrate fuel oil
AP	acid generating potential
ARD	acid rock drainage
ASCu	acid soluble copper
Au	gold
bcm	bank cubic metres
BCRA	Argentine Central Bank
bn	bornite
BOO	build, own, operate
BWi	bond work index
CAMMESA	Compañía Administradora del Mercado Eléctrico Mayorista
CCTV	closed-circuit television
CO_2	carbon dioxide
cpy	chalcopyrite
csv	comma separated value
Cu	copper
CuCon	copper concentrate
CWi	crusher work index
DEFRA	Department for Environment, Fuel and Rural Affairs
DN	normal diameter
Dwg	drawing
DWT	deadweight tonnage
ECS	expert control system
EGL	effective grinding length
ERT	emergency response team
ESIA	environmental, social impact assessment
EW	electrowinning
FCAB	Ferrocarril de Antofagasta a Bolivia
FEL	front end loader
FID	financial investment decision
FPIC	free, prior and informed consultation
ft	foot
G&A	general and administration
g, kg	grams, kilograms
g/t, kg/t	grams per tonne, kilograms per tonne
GAC	granulated activated carbon
GISTM	Global Industry Standard on Tailings Management
H_2S	Hydrogen sulphide
ha	hectares
HDPE	high-density polyethylene
HPGR	high pressure grinding rolls
HR	human resources
HV	high voltage, heavy vehicle
ICP	inductively coupled plasma
ICP-OES	inductively coupled plasma - optical emission spectroscopy
INPRES	Instituto Nacional de Prevención
IPCC	In pit crushing and conveying
IROC	integrated remote operations centre
IRR	internal rate of return

Term	Definition
IT	information technology
K	hydraulic conductivity
kL, ML	kilo litres, mega litres
KNA	kriging neighbourhood analysis
KNAN	K non-mineral oil
kV	kilovolt
kWh	kilowatt hour
kWh/t	kilowatt hours per tonne
L/s	litres per second
lb	pounds
lcm	loose cubic metres
LHD	load, haul, dump
LOM	life of mine
LUC	localised uniform conditioning
LV	light vehicle
m	metre
M&A	Montgomery and Associates
m/s	metres per second
Ma	mega annum (million years)
MAA	multiple accounts analysis
masl	metres above sea level
MBR	membrane bioreactor
MCC	motor control centre
MDE	maximum design earthquake
mE, mN	coordinates: metres East, metres North
MIBC	methyl isobutyl carbinol
MLA	mineral liberation analysis
Mm^3	million cubic metres
Mo	molybdenum
MO, LG, MG, HG	marginal ore, low grade, medium grade, high grade
MPU	mobile processing unit
mRL	metres reduced level
MSA	mine services area
Mtpa	million tonnes per annum
MVA	mega volt ampere
MW	mega watts
NAF	net acid forming
NaHS	Sodium hydrosulphide
NaOH	sodium hydroxide
NNA	neural network analysis
NNP	net neutralising potential
NOA	Noreste Argentino
NP	neutralising potential
NPI	non-processing infrastructure
NPR	net potential ratio
NPV	net present value
OEM	original equipment manufacturer
OK	ordinary kriging
OMC	Orway Mineral Consultants
ONAF	oil natural air forced
ONAN	oil natural air natural
OPGW	optical ground wire
OSA	on-stream analyser
oz	(troy) ounces
P_{80}	80% passing (sieve size)
PAF	potentially acid forming
PAX	potassium ethyl xanthate
pH	potential of hydrogen

Term	Definition
PN	normal pressure
ppm	parts per million
py	pyrite
Q1, Q2, Q3, Q4	quarter 1 to 4
QAQC	quality assurance, quality control
RAAC	Regulaciones Argentinas de Aviación Civil
RIGI	Incentive Regime for Large Investments
RL	reduced level
ROM	run of mine
RQD	Rock Quality Designation
RTZ	Rio Tinto
RWi	rod work index
SABC	semi autogenous ball mill crusher
SAG	semi autogenous grinding
SCADA	supervisory control and data acquisition
SCH	schedule
SegemAR	Servicio Geológico Minero Argentino
SEX	sodium ethyl xanthate
SIN	Argentine Interconnection System
SiO_2	silica
SME	Mining and Energy Secretariat
SMU	smallest mining unit
SOC	Salta Operations Centre
SPT	standard penetrometer test
SRTM	Shuttle Radar Topography Mission
STD WT	standard weight
SX	solvent extraction
t, kt, Mt	tonnes, thousands of tonnes, millions of tonnes
$t/m^2/h$	tonnes per square metre per hour
TA	trolley assist
TCu	total copper
TDH	total dynamic head
TDS	total dissolved solids
tpa	tonnes per annum
tpd	tonnes per day
tph	tonnes per hour
TSF	tailings storage facility
TUSA	Transportes Universales
USACE	United States Army Corps of Engineers
USGS	United States Geological Survey
UV	ultraviolet
V, kV	volts, kilovolts
W, MW	watts, megawatts
w/w	weight/weight (ratio of the mass of components in a mixture)
WBS	work breakdown structure
WGS	Western Geodetic System
WNW	west north west
WRD	waste rock dump
WWTP	waste water treatment plant

ITEM 3 RELIANCE ON OTHER EXPERTS

The QPs of this Technical Report do not disclaim any responsibility for the content contained herein. However, certain relevant information included in Item 22 Economic Analysis, specifically the information in Items 22.1.2 and 22.2.6, has been provided by the Company’s internal taxation team and relates to:

- the applicable corporate tax rate in Argentina
- the estimated Project taxable income arising from capital depreciation and taxation calculations

The QPs have relied on this information for the purposes of the Project Economic Analysis in Item 22.

ITEM 4 PROPERTY DESCRIPTION, LOCATION AND TENURE

4.1 Proposed Project

The proposed Taca Taca Project involves the open pit mining and flotation processing of copper and molybdenum ore for a period in excess of thirty years.

A proposed Stage 1 40 Mtpa Project follows a three and a half year mining pre-strip and Project development phase, and extends for a duration of fifty years. Project engineering efforts to date have focussed on the design, layouts and costings for this Stage 1 operation. Information is presented in this document on an expansion of the production rate to a 60 Mtpa Stage 2, commencing five years after the pre-strip phase and shortening the production timeframe to thirty five years.

The porphyry copper-gold-molybdenum orebody is hosted by a batholith and is overlain by sediments and volcanoclastics. The optimal pit depth is in excess of 700 m and it is situated immediately adjacent to a brine saturated salar. The mineralisation lies beneath a leached cap and is typically hypogene and supergene, but with distinct transitional (or mixed) mineralisation styles also present. The proposed processing flotation processing method involves a conventional concentrator producing separate gold and molybdenum products, and with much of the gold recovered into the copper concentrate.

Throughout this Study, reference to the Taca Taca deposit or Project, relates to the Taca Taca Bajo deposit, unless otherwise noted.

4.2 Project description

The Project description relates to a two stage development, commencing with a Stage 1 40 Mtpa operation which could be expanded to a Stage 2 60 Mtpa operation. The Project has the following material components, aspects of which are described in detail in this Technical Report update:

- an open pit mine
- a mining waste dump
- a surface ore stockpile
- a copper concentrator for the processing of copper mineralisation by flotation methods, also recovering gold into the concentrate
- an ability to separate a molybdenum concentrate in a separate flotation circuit
- concentrate filtration and load out facilities
- a tailings storage facility for the storage of the processing residues
- process water storage tanks
- internal access roads, surface haulage roads and mine haulage roads (including trolley-assisted routes)
- borefields for the supply of fresh from four catchments
- borefields for the collection of brine from the Salar de Arizaro and Salar de Taca Taca
- overland pipelines between the concentrator and the tailings storage facility, and between water supply borefields and the plant site
- mine services workshops and equipment wash-down facilities
- construction offices, mine administration and camp/village accommodation facilities
- storage space and a rail loadout facility for concentrate product
- storage facilities for parts and consumables, reagents and explosives
- as auxiliary infrastructure, there is an airstrip upgrade, roads for transporting supplies into the Project site, a railway for transporting concentrates and supplies, and an in-coming high voltage electric transmission line

4.2.1 Project updates and developments

The technical changes reflected in this Technical Report update can be summarised as:

- Reporting of the Mineral Resource estimate within the constraints of a life of mine optimisation shell
- The adoption of increased metal prices
- An update of the capital and operating cost estimates
- A revised water balance, involving the use of fresh water for both rougher and cleaner flotation, and the reduced use of brine during tailings discharge
  - there is a change in the water supply projections from fresh water borefields and from brine abstraction, and hence the water balance has been updated accordingly.
- A move away from an ultra-class truck fleet, and the adoption of a suitably scaled starter fleet for the waste pre-strip period.
  - the concept of an ultra-class mining fleet has changed, mostly in respect of haul trucks
  - in place of 365 tonne capacity trucks, 290 tonne capacity trucks are now being considered
  - additionally, a pioneering fleet is being introduced, comprising 92 tonne capacity trucks and compatible hydraulic excavators
- A revised design of mining stages to better suit spatial constraints during the substantial pre-stripping period
  - the open pit stage designs have changed, as has the layout of pit, waste dump and surface haul roads
- A revised estimate of the NAF and PAF waste volumes and hence the sequence of waste dumping onto the Salar de Arizaro
- An updated layout for the Project facilities, including the process plant, and associated non-process infrastructure (NPI) such as the mine services area and rail load-out (concentrate) facility
  - the fundamental infrastructure components of the Project are unchanged, other than modifications to the positioning and layout of the plant site and related facilities, and to the train load-out configuration
  - the infrastructure facilities have been redesigned to a greater degree of detail, sufficient now to improve the comprehensiveness and accuracy of the Project capital cost estimates.
  - the proposed primary ore crushing facility is now located near surface in an excavation adjacent to the initial starting stage of the open pit, thereby replacing the previous concept of crushing adjacent to a surface ROM (run-of-mine) pad
  - whilst this crushing facility could remain in the same location for the life of the operation, there is flexibility in the design to enable it to be repositioned at some suitable future time to a lower elevation on the western side of the pit
- A possible satellite pit at Little Taca is no longer under consideration.
  - following sterilisation drilling, the mineralisation in this area is now considered to be discontinuous and unworthy of Mineral Resource classification
  - as such, the immediate surface area is now the proposed site of the mine services area (MSA)

4.3 Project location

The proposed Taca Taca Project is situated in the Puna (Altiplano) region of Salta Province, in northwest Argentina, approximately 230 km west of the city of Salta and 55 km east of the Chilean border (Figure 4-1). A minor population centre is at Tolar Grande, about 35 km east of the Project site.

The site is located at a median elevation of 3,625 m RL, and at latitude 24.7° S and longitude 68.0° W. The UTM co-ordinates at the centre of the site are 7,283,500 mN and 2,628,000 mE.

4.4 Project ownership

The Taca Taca Project is 100% owned by FQM through its Argentinian subsidiary Corriente Argentina SA (CASA). FQM, which is an international mining company listed on the Toronto stock exchange, acquired the Project from previous owners Lumina Copper Corporation (Lumina), in August 2014.

4.5 Mineral tenure, rights, payment agreements and encumbrances

The Company holds a significant package of mining rights in the region, consisting of 83 mining concessions (*minas*). The main Project area is contained within a composite package of 13 concessions (*minas*) over the deposit and adjacent areas comprising the Taca Taca Mining Group. Two of the mining concessions have a 50% ownership with a third party group, though these are not over commercially material areas of the known deposit. The other concessions are held 100% by the Company.

Figure 4-2 shows the location of the Project relative to Company concessions in the municipality of Tolar Grande, Los Andes Department, Salta Province, Argentina. Table 4-1 and Table 4-2 list details for each of the concessions. The Taca Taca mining group concessions cover an area of 2,560 ha, whilst the additional concessions cover an area of 113,389 ha.

CASA has the right to receive and deal with 100% of the copper and gold or any other mineral production from the mining properties, except for those listed in Figure 4-2 for which CASA's title is restricted to a 50% share. The 50% joint venture concessions are shown with a light purple coloured perimeter.

The mining legal framework in Argentina is set out in the Mining Code of National Law No. 1919 which awards strong rights to owners of mining concessions, including perpetual duration for mining exploitation concessions.

Under the Mining Code, these mining concessions vest the Company with property title over the mine, including the right to explore and exploit within the concession area. The mining concessions are granted in perpetuity and are not subject to a validity term if bi-annual canon payments are made. The National Congress fixes the annual canon per concession, and this is paid in advance in two equal instalments on June 30th and December 31st of each year. The canon payments relating to CASA properties total approximately $50,000 per annum. The Taca Taca concessions have fulfilled all the mandatory requirements and CASA can continue to hold the concessions in good standing.

The Mining Code contains clear provisions which have not been subject to changes under various government administrations to date.

Several of the CASA concessions listed in Table 4-2 are the subject of an option agreement with a third party, Silvia René Rodríguez. These concessions, located near Socompa on the Argentine/Chile border, are within an "Area of Influence" surrounding the *Mina Socompita* concession which is wholly owned by Silvia René Rodríguez . Although of no relevance to the Taca Taca Project development footprint, the concessions which are subject to this agreement present some possible future exploration potential.

Figure 4-1 Taca Taca Project location map

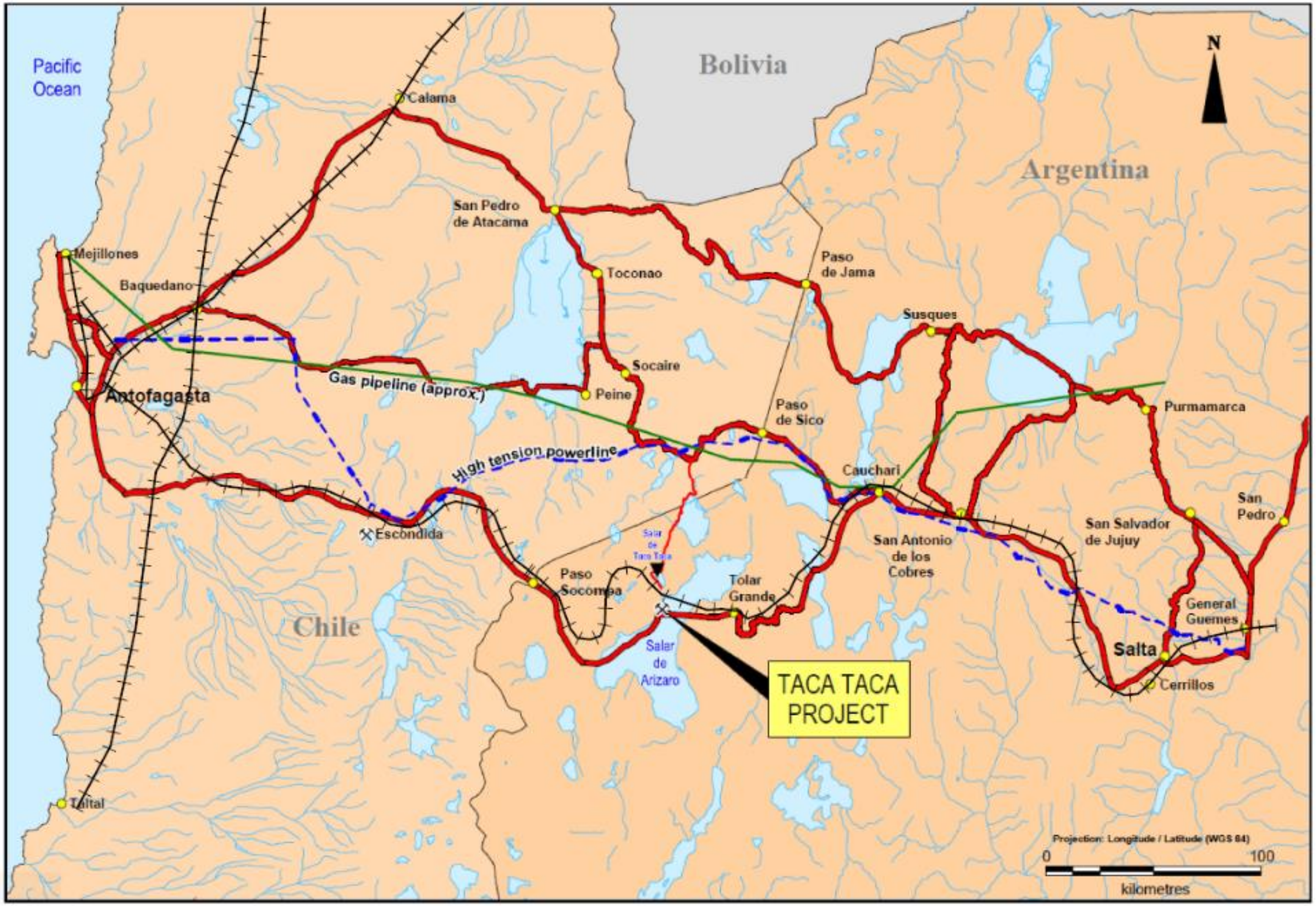


Figure 4-2 Taca Taca Project mineral concessions map

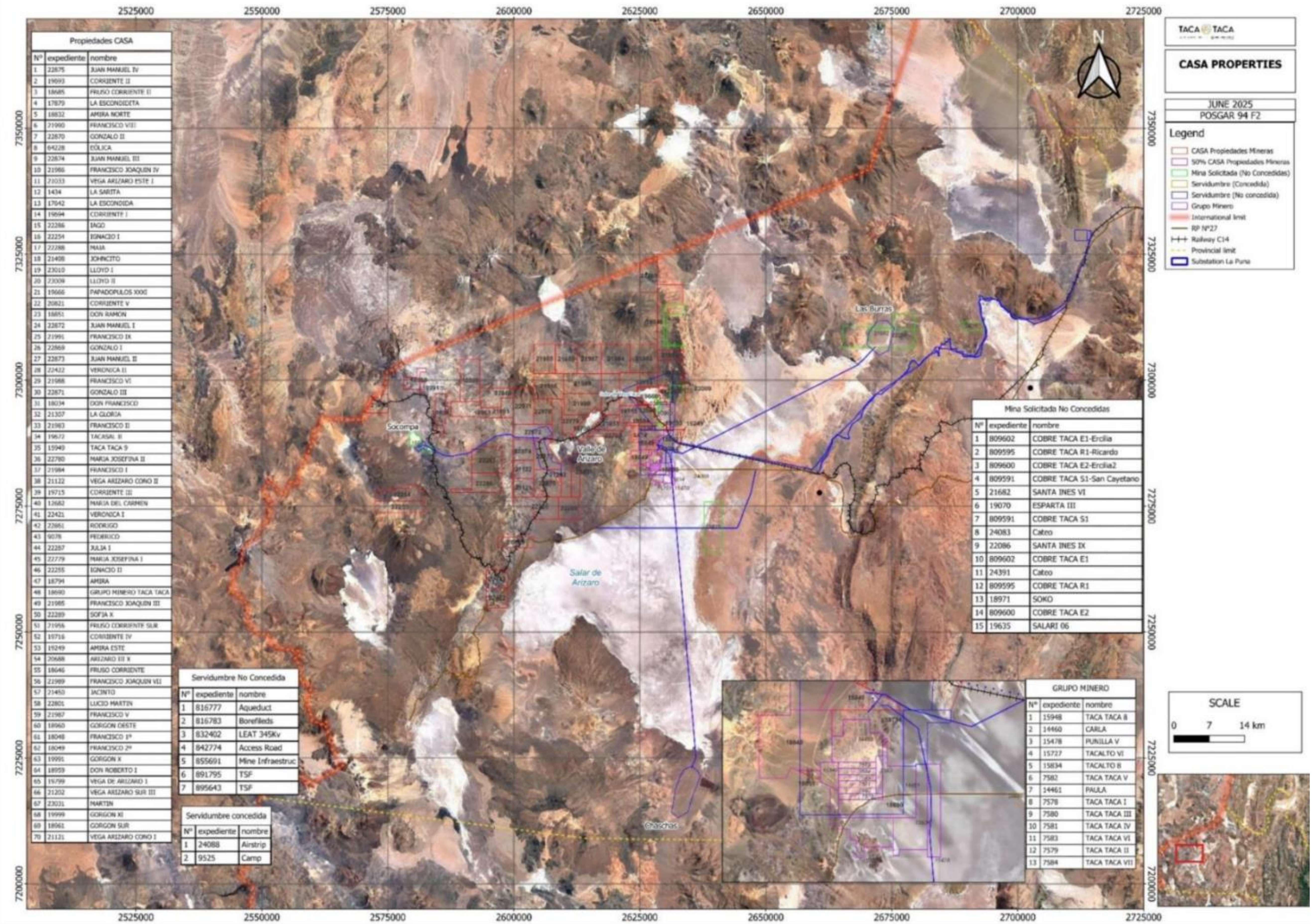


Table 4-1 Details of Project mineral concessions, Taca Taca Mining Group

Number	Concession	Number	Area (ha)	Owner	Royalty
	Taca Taca Mining Group				
1	Mina Carla	14460 - 1992	400.1	CASA 100%	1.50%
2	Mina Paula	14461 - 1992	599.6	CASA 100%	1.50%
3	Mina Punilla V	15478 - 1996	281.2	CASA 100%	
4	Mina Tacalto 6	15727 - 1996	394.2	CASA 100%	
5	Mina Tacalto 8	15834 - 1997	399.8	CASA 100%	
6	Mina Taca Taca 1	7578 - 1970	63.0	CASA 100%	1.50%
7	Mina Taca Taca 2	7579 - 1970	54.2	CASA 100%	1.50%
8	Mina Taca Taca 3	7580 - 1970	54.0	CASA 100%	1.50%
9	Mina Taca Taca 4	7581 - 1970	53.9	CASA 100%	1.50%
10	Mina Taca Taca 5	7582 - 1980	54.1	CASA 100%	1.50%
11	Mina Taca Taca 6	7583 - 1970	54.0	CASA 100%	1.50%
12	Mina Taca Taca 7	7584 - 1970	53.9	CASA 100%	1.50%
13	Mina Taca Taca 8	15948 - 1997	98.4	CASA 100%	1.50%
	Total area		**2,560.3**		

Table 4-2 Details of Project mineral concessions, additional concessions

Number	Concession	File No.	Surface (ha)	Owner	Royalty	Comments
1	Mina Taca Taca 9	15949-1997	376.0	CASA 100%	1.50%	
2	Mina Fruso Corriente	18646-2007	3,500.0	CASA 100%		
3	Mina Fruso Corriente II	18685-2007	2,500.0	CASA 100%		
4	Mina La Sarita	1434-1942	168.0	CASA 100%		
5	Mina Federico	9078-1974	40.0	CASA 100%		
6	Mina Don Ramon	18851-2007	26.0	CASA 100%		
7	Mina Amira Norte	18832-2007	1,500.0	CASA 100%	1.50%	
8	Mina Amira	18794-2007	433.6	CASA 100%	1.50%	
9	Mina Amira Este	19249-2008	81.1	CASA 100%	1.50%	
10	Mina Don Francisco	18034-2004	340.0	CASA 100%		
11	50% Mina Francisco 1	18048-2005	1,300.0	CASA 50%	0.75%	
12	50% Mina Francisco 2	18049-2004	1,000.0	CASA 50%	0.75%	
13	Mina La Gloria	21307-2011	199.4	CASA 100%	1.50%	
14	Mina Corriente I	19694-2009	134.4	CASA 100%		
15	Mina Corriente II	19693-2009	71.9	CASA 100%		
16	Mina Corriente III	19715-2009	2,500.0	CASA 100%		
17	Mina Corriente IV	19716-2009	3,500.0	CASA 100%		
18	Mina Corriente V	20281-2011	523.0	CASA 100%		
19	Mina Francisco Joaquin I	21984-2013	3,262.4	CASA 100%		
20	Mina Francisco Joaquin II	21983-2013	3,000.0	CASA 100%		
21	Mina Francisco Joaquin III	21985-2013	2,500.0	CASA 100%		
22	Mina Francisco Joaquin IV	21986-2013	2,500.0	CASA 100%		
23	Mina Francisco Joaquin V	21987-2013	2,752.0	CASA 100%		
24	Mina Francisco Joaquin VI	21988-2013	3,000.0	CASA 100%		
25	Mina Francisco Joaquin VII	21989-2013	3,000.0	CASA 100%		
26	Mina Francisco Joaquin VIII	21990-2013	2,924.3	CASA 100%		
27	Mina Francisco Joaquin IX	21991-2013	927.7	CASA 100%		
28	Mina Ignacio I	22254-2013	2,299.5	CASA 100%		
29	Mina Ignacio II	22255-2013	2,300.0	CASA 100%		
30	Mina Iago	22286-2014	2,569.7	CASA 100%		
31	Mina Julia I	22287-2014	3,000.0	CASA 100%		
32	Mina Maia	22288-2014	3,000.0	CASA 100%		
33	Mina Sofia X	22289-2014	1,765.7	CASA 100%		
34	Mina Veronica I	22421-2014	1,500.0	CASA 100%		
35	Mina Veronica II	22422-2014	1,654.1	CASA 100%		
36	Mina Johncito	21498-2012	47.4	CASA 100%		
37	Mina La Escondida	17642-2003	37.9	CASA 100%		
38	Mina La Escondidita	17879-2004	6.6	CASA 100%		
39	Mina Fruso Corriente Sur	21956-2013	1,000.0	CASA 100%		
40	Mina Tacasal II	19672-2009	2,422.0	CASA 100%		
41	Mina Papadopulos XXXI	19666-2009	284.1	CASA 100%		
42	Mina Maria Josefina I	22779-2016	1,492.8	CASA 100%		
43	Mina Maria Josefina II	22780-2016	777.7	CASA 100%		

Number	Concession	File No.	Surface (ha)	Owner	Royalty	Comments
44	Mina Lucio Martin	22801-2016	18.3	CASA 100%		
45	Mina Rodrigo	22861-2016	2,219.3	CASA 100%		
46	Mina Gonzalo I	22869-2016	2,095.0	CASA 100%		
47	Mina Gonzalo II	22870-2016	1,899.8	CASA 100%		
48	Mina Gonzalo III	22871-2016	2,907.2	CASA 100%		
49	Mina Juan Manuel I	22872-2016	2,444.4	CASA 100%		
50	Mina Juan Manuel II	22873-2016	2,445.1	CASA 100%		
51	Mina Juan Manuel III	22874-2016	1,738.9	CASA 100%		
52	Mina Juan Manuel IV	22875-2016	2,957.4	CASA 100%		
53	Mina Maria del Carmen	12682-1986	90.5	CASA 100%		
54	Mina Lloyd I	23010-2017	2,971.8	CASA 100%		
55	Mina Lloyd II	23009-2017	148.9	CASA 100%		
56	Mina Eolica	64228-1956	35.3	CASA 100%		
57	Mina Gorgon Oeste	18960-2007	982.4	CASA 100%		Shared with Silvia René Rodriguez in a Framework of an agreement
58	Mina Jacinto	21450-2012	1,195.3	CASA 100%		
59	Mina Arizaro III X	20688-2010	1,499.3	CASA 100%		
60	Mina Vega Arizaro Cono II	21122-2011	1,464.7	CASA 100%		
61	Mina Vega Arizaro Este I	21033-2011	1,407.0	CASA 100%		
62	Mina Gorgon X	19991-2009	2,499.0	CASA 100%		Shared with Silvia René Rodriguez in a Framework of an agreement
63	Mina Gorgon Sur	18961-2007	2,600.0	CASA 100%		
64	Mina Martin	23031-2017	1,479.0	CASA 100%		
65	Mina Vega Arizaro Cono I	21121-2016	1,484.0	CASA 100%		
66	Mina Vega Arizaro Sur III	21202-2011	1,461.0	CASA 100%		
67	Mina Vega de Arizaro I	19799-2000	1,843.0	CASA 100%		
68	Mina Gorgon XI	19999-2009	3,227.0	CASA 100%		Shared with Silvia René Rodriguez in a Framework of an agreement
69	Mina Don Roberto	18959-2007	1,500.0	CASA 100%		Shared with Silvia René Rodriguez in a Framework of an agreement
70	Grupo Minero Taca Taca	18690-2007	2,557.6	CASA 100%		
	Total area		**113,388.6**			

Alongside the mining properties, FQM also holds and has applied for easements for the Project's site and regional infrastructure as listed in Table 4-3.

Table 4-3 Project easements

N°	Easement	File No.	Surface (ha)	Status
1	Camp	9525-B	6	Granted
2	Airstrip	24088-19	236	Granted
4	Aqueduct to four catchments	816777-23	917	Ongoing
5	Borefields of four catchments	816783-23	9,643	Ongoing
6	345kv transmission line	832402	985	Ongoing
7	Access road	842774	601	Ongoing
8	Mine and project facilities	855,691	365	Ongoing
9	Mine and project facilities	891,795	471	Ongoing
10	Mine and project facilities	895,643	6,900	Ongoing

4.6 Royalties

The levying of royalties and administration of the Mining Code are the responsibility of the Salta provincial government, which has a long history of being stable and supportive of the mining Industry.

Ten of the Taca Taca Mining Group concessions are subject to a contractual royalty of 1.5% of net smelter return (the Taca Taca royalty). The three concessions which are not subject to a 1.5% royalty are located on the Salar de Arizaro. In addition, there is 3% royalty payable to the Province of Salta, net of smelting/refining, transport, general and administration costs, and process operating costs[3].

Franco Nevada Corp., through a wholly owned subsidiary, holds the right to receive a 72% interest in the Taca Taca royalty, whilst the remaining 28% interest is held by other parties.

4.7 Incentive Regime for Large Investments (RIGI)

On July 8th 2024, the Government of Argentina's President Javier Milei enacted the "Law of Grounds and Starting Points for the Freedom of Argentines", which includes a new incentive regime for large investments *(Régimen de Incentivo para Grandes Inversiones*, "RIGI") with a two-year window to apply starting on the same date. The current deadline for RIGI applications is July 8th, 2026 with possibility of a one-year extension. The legislation provides unrestricted foreign exchange access and a specific tax and customs regime, focusing on predictability, stability, and legal certainty across various sectors, including mining. On September 19th 2024, Salta province formally adhered to the regime, extending its benefits to include local tax stability.

4.8 Environmental liabilities

There are no known environmental liabilities currently existing on the Taca Taca property.

4.9 Permits that must be acquired

The primary approval required for the development of the Taca Taca Project is the approval of the Project ESIA by the Secretariat of Mining of Salta Province. This ESIA covers the main Project sites including mine, process plant, tailings storage facility, and associated facilities.

The Project ESIA was submitted in February 2019 (Ausenco, February 2019).

Several Project description documents were submitted to support these ESIA submissions, including separate descriptions on aspects of the proposed mining and processing plan (FQM, November 2018), the power supply route alternatives and the proposed site access route alternatives. Engineering details are described in these documents in the context of potential environmental and social impacts.

Observations and responses to the Project ESIA submission were received from the Mining Secretariat at the end of Q3 2019, including requests for clarification and further information on some environmental aspects (62 observations). The Company's response document to the Secretariat of Mining was submitted in Q1 2020, including reports on additional studies and aspects such as the conceptual closure plan, an initial geotechnical investigation of the Salar de Arizaro surface bearing capacity, and an assessment of alternative waste landfill sites, amongst others.

In August 2022, complementary information was submitted by FQM to be incorporated into the Project ESIA. This information included an SRK Consulting study of alternatives and conceptual designs for the TSF and WRD (SRK, 2022), in addition to a report by Piteau Associates on hydrogeological studies (Piteau, 2022).

[3] In the Item 22 Economic Analysis, the Salta provincial royalty is deemed to be a profit-based income tax.

A second round of observations (numbering 173) was received from the SME in June 2023. These observations were responded to in October 2023 (FQM, October 2023) including new information on water abstraction from the Valle de Arizaro, Socompa, Valle de Chaschas and Valle de las Burras basins.

In October 2024, a collaborative workshop was held with SEGEMAR[4] and the provincial authorities including the Mining and Energy Secretariat (SME). This workshop included a site visit, technical briefings and presentations from Company staff, followed by interactive questioning on numerous aspects of the Project.

These submissions and presentations remain under review by the relevant authorities.

The ESIA process requires a final report from the SME on their observations and requests for further information. Once the observations process is satisfactorily concluded, there needs to be a public hearing (*audiencia publica*) prior to the ESIA approval.

In accordance with environmental laws governing the Project, the Project ESIA will need to be updated every two years. Updates to the Project Description will be incorporated into subsequent ESIA submissions accordingly.

Additional ESIA documents have been prepared for the feasibility approval of the 345 kV transmission line connecting the national grid to the Project site (FQM, March 2020). Pre-feasibility for the transmission line was submitted in 2021 to the SME and approved in November 2022. The subsequent feasibility ESIA is under evaluation with the authorities after its submission in September 2025.

There is a separate ongoing pre-feasibility ESIA, submitted in 2020, for the proposed access road diversion (FQM, September 2020). There is a new proposed diversion considering a segment of 40 km length to avoid the provincial road RP Nº 27 traffic passing through the mine area. This information is being updated in the permitting application file.

Abstraction permits will be required for the fresh water borefields located at Valle de Las Burras, Valle de Arizaro, Valle de Chaschas and Socompa. In the case of brine abstraction, permit application for borefields and trenches over the Taca Taca and Arizaro salars will be submitted after the results of the ongoing campaign are assessed.

In addition to water permits, other approvals are required for the construction and operation of the mine and ancillary facilities. Specific building permits, waste and chemical handling authorisations, are granted by the provincial and national authorities.

4.10 Factors and risks which may affect access or title

The QPs of this Technical Report are unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the property.

[4] SEGEMAR (*Servicio Geológico Minero Argentino) is* a national scientific and technological body responsible for the production of geological, technological, mining and environmental geological knowledge and information on the territory of the Argentine Republic and the continental shelf.

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

5.1 Topography, elevation and physiography

The Taca Taca Project is located on the eastern side of the Sierra de Taca Taca mountain range, and on the western side of the expansive Salar de Arizaro which lies at an altitude of 3,470 m RL. The Project site is located at a median elevation of 3,625 m RL, and at latitude 24.7°S and longitude 68.0°W.

The local topographic relief is low to moderate, with two prominent peaks in the immediate vicinity, Cerro de Cobre and Cerro Agua del Desierto. The volcano Cerro Aracar is located about 30 km north of the Project site. From Wikipedia: No historical activity is recorded, but in March 1993 inhabitants of Tolar Grande observed a high ash or steam column rising from Aracar, which could have been either an eruption or the result of landslides.

5.2 Seismic conditions

The Project is located in the Andes, near the Chilean border in the Circum-Pacific Belt, which is an active seismic region. The large-scale regional tectonic framework is governed by the interaction of the Nazca and South American plates. The main tectonic features in this region, namely the Andes and the Peru-Chile Oceanic trench, are related to the high seismic activity, and are a result of the two converging plates. The most notable result of this collision is the contemporary orogenic process constituted by the Andes formation.

Seismic criteria and regulation in Argentina are promulgated by the *Instituto Nacional de Prevención Sísmica* (INPRES). The Argentinian seismic code INPRES-CIRSOC 103 regulates general construction, however for special constructions such as for a TSF embankment, there is no specific regulation in force. INPRES divides the country into five zones, with the highest seismicity concentrated in the west-central section of the country, including the Mendoza and San Juan provinces. The Project is located in zone 2, rated as having "moderate" seismic risk with a maximum ground acceleration of 0.18 g. This corresponds to a 10% probability of exceedance in a 50-year period.

According to the United States Geological Survey (USGS) seismic hazard map (2018 update), the peak ground acceleration for a 475-year return period event, which corresponds to a 10% probability of exceedance in a 50-year period, is approximately 0.32 – 0.33 g for the Taca Taca site. In their Project design work, Fluor Australia Ltd (Fluor, 2025 A and B), has adopted an acceleration figure of 0.32 g or 3.1 m/sec^2.

Salta falls within zone 3 of the INPRES division, which is rated as having a higher seismic risk, and hence a peak ground acceleration of 0.45 g or 4.4 m/sec^2. This figure has been considered by Fluor in the design of the Salta Operations Centre facilities.

In view of the seismicity risk for the area, and despite the differing INPRES and USGS criteria, the Project will be designed according to engineering parameters derived from a detailed engineering risk analysis. A site-specific seismic hazard study will be performed during the next phase of project engineering development and will be used for designing facilities including the TSF. This would include a review of the historical and instrumental seismicity, the regional tectonic features, and the definition of seismic sources with their respective recurrence.

Although general seismic design parameters from past projects were incorporated into the conceptual design of the TSF, the Maximum Design Earthquake (MDE) and stability analyses for these facilities were not performed for this level of study.

5.3 Vegetation

Vegetation is sparse to non-existent in the Project area. The dry puna vegetation is characterised by grasses, alpine herbs and dwarf shrubs. Salty *vegas* are present where freshwater enters into the various salars of the region. There are also man-made *plumas verde* present in the Salar de Taca Taca.

5.4 Climate

Project climatic information has been drawn from recordings at three stations installed by the Company, over various durations and at several locations as follows:

- Caipe Alto station, MET-01, 2018 to 2021
- Arizaro station, MET-02, 2018 to 2021
- Taca Taca station, 2011 to 2020

5.4.1 Temperatures

The highest temperatures are recorded during the summer months (December, January, and February) and the lowest temperatures are recorded during the winter months (June and July). The average minimum temperature is around 3.7°C during the winter months, while the maximum temperature reaches 29.8°C during the summer period. Table 5-1 lists the monthly average, maximum and minimum temperatures recorded at the Taca Taca station.

Table 5-1 Average monthly temperatures (source: Piteau, 2022)

Temp (°C)	Jan	Feb	Mar	Apr	May	June	July	Aug	Sept	Oct	Nov	Dec	Average
Median	14.6	14.0	12.8	9.7	5.9	3.8	3.6	5.3	8.4	9.7	11.9	14.1	9.5
Minimum	1.2	-1.0	-2.0	-5.1	-12.7	-11.3	-13.1	-13.1	-6.8	-7.4	-2.9	-1.3	-6.3
Maximum	29.8	28.8	26.7	25.3	19.2	17.6	18.8	22.2	23.3	24.7	25.9	26.9	24.1

5.4.2 Precipitation

Despite the hyper-arid surface conditions that characterise the Puna region, the regional hydrological system is strongly influenced by precipitation occurring at higher elevations in the surrounding mountain ranges. Rainfall and snowmelt in these upland areas infiltrate fractured bedrock and unconsolidated sediments, contributing to groundwater recharge over long timescales. Regional hydrogeological studies indicate the presence of potentially significant groundwater-bearing units, which, in certain basins, may extend to depths of up to approximately 1,000 m below surface, depending on local geological and structural conditions.

In many endorheic basins of the Altiplano–Puna, groundwater and surface inflows converge toward topographic lows (salars), where discharge is predominantly through evaporation and evapotranspiration. At a regional scale, the hydrological system can be described as a long-term dynamic equilibrium between recharge from higher elevations, groundwater storage, and evaporative discharge in low-lying basins; consequently, surface water expression is limited and groundwater processes dominate the regional hydrological system. Current indications are that these are isolated water basins.

The average annual precipitation for the three weather stations and for the available recording years is:

- Caipe Alto station, 40 mm
- Arizaro station, 93 mm
- Taca Taca station, 49 mm

Table 5-2 lists the monthly average precipitation recorded at the Taca Taca station (elevation 3,536 mRL).

This table also lists the median of daily recorded precipitation at the Arizaro (elevation 4,440 mRL) and Caipe (elevation 4,730 mRL) stations. To note are the different precipitation patterns at these higher elevations.

Table 5-2 Average monthly precipitation (source: FlowHydro, 2024)

Prec (mm)	Jan	Feb	Mar	Apr	May	June	July	Aug	Sept	Oct	Nov	Dec	Total
2011										0.0	0.0	28.0	**28.0**
2012	18.0	50.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**68.0**
2013	16.0	19.0		0.0	5.0	3.0	0.0	0.0	0.0	0.0	0.0	1.0	**44.0**
2014	21.0	2.0	0.0	0.0	0.0	0.0			0.0	0.0	0.0	0.0	**23.0**
2015		4.0	18.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0		0.0	**22.0**
2016	0.0	0.0	0.0	2.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	**2.0**
2017	17.0	40.0	0.0	0.0	7.0	5.0	0.0	0.0	0.0	0.0	0.0	0.0	**69.0**
2018	1.0	18.0	1.0	0.0	0.0	0.0	0.0				0.0		**20.0**
2019	46.0				0.0							0.0	**46.0**
2020			19.0	0.0	0.0	0.0							**19.0**
2021					2.4	0.0	10.8	0.2	0.0	0.0	0.0	12.4	**25.8**
2022	30.6	0.0	0.0	0.0	0.0	0.4	0.0	0.0	0.0	0.0	0.0	16.4	**47.4**
2023	0.0	31.6	8.2	0.0				0.0	0.0	0.0	0.0	0.0	**39.8**
2024	0.0	31.6											**31.6**
TT median	**15.0**	**19.6**	**5.1**	**0.2**	**1.3**	**0.8**	**1.4**	**0.0**	**0.0**	**0.0**	**0.0**	**5.3**	**48.7**
Arizaro median	0.0	0.0	0.0	0.0	0.0	5.0	10.0	32.1	11.9	0.0	0.0	0.0	**59.0**
Caipe median	0.0	0.0	1.6	0.6	0.0	0.0	0.0	4.6	4.5	0.1	0.0	0.0	**11.4**

5.4.3 Relative humidity

Table 5-3 lists the monthly average relative humidity for the Taca Taca station. Mild seasonal variations are observed, with higher relative humidity values in the summer months from December to March, and lower values in winter. This coincides with the months experiencing the greatest number of storms (in summer), and hence the higher humidity in those months. The average annual humidity is approximately 18.7%.

Table 5-3 Monthly and annual relative humidity (source: Piteau, 2022)

RH (%)	Jan	Feb	Mar	Apr	May	June	July	Aug	Sept	Oct	Nov	Dec	Total
Median	28.8	33.2	25.9	19.5	20.1	20.1	16.7	12.8	11.6	10.4	9.5	15.4	18.7

5.4.4 Solar radiation

The solar radiation records shows highlight a trend of seasonality, with the highest intensity recorded during the spring-summer months (September-March) and lowest intensity recorded during the autumn-winter months (April-August).

Monthly variability is determined by the number of hours of sunshine and the intensity of radiation resulting from the sun's inclination relative to the Earth. On a daily basis, the maximum radiation levels occur around the time of solar culmination. The average annual solar radiation is 291 Watt/m^2.

Table 5-4 shows the values corresponding to the average and maximum monthly records for the Taca Taca meteorological station.

Table 5-4 Average and maximum monthly and annual solar radiation levels (source: Piteau, 2022)

W/m^2	Jan	Feb	Mar	Apr	May	June	July	Aug	Sept	Oct	Nov	Dec	Average
Median	349	327	305	255	208	186	200	243	302	353	385	382	291
Maximum	1,688	1,580	1,420	1,313	1,134	1,053	1,005	1,157	1,317	1,503	1,619	1,508	1,358

5.4.5 Evaporation

The evaporation measured at the Taca Taca station is shown in Table 5-5. The highest daily values occur in the spring and summer months, and the lowest in the autumn and winter.

Table 5-5 Average monthly and annual evaporation (source: Piteau, 2022)

Evap (mm)	Jan	Feb	Mar	Apr	May	June	July	Aug	Sept	Oct	Nov	Dec	Total
Median	182	167	169	174	205	145	118	70	84	124	160	172	1,770

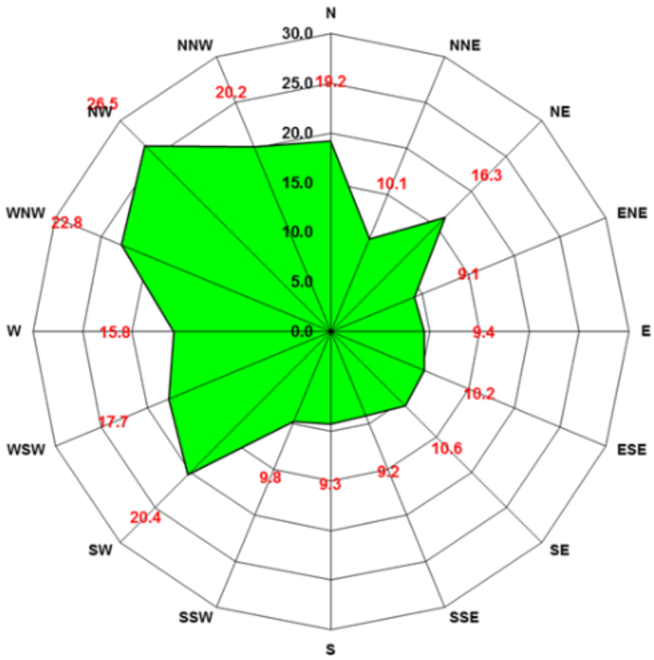
Evaporation significantly exceeds both precipitation and recharge, leading to a net loss in water storage. Further discussion on recharge and water storage is provided in Item 24.

5.4.6 Wind

Typical wind speeds range from 3.8 m/s to 23.2 m/s, blowing predominantly from the northwest. Figure 5-1 shows wind speed (average) and direction at the Project site.

In a review of climatic conditions impacting on Project water supply, Montgomery & Associates (M&A, November, 2018) advised that wind gusts in the Puna region can exceed 90 km/h. Sustained wind speeds during the austral winter can range between 20 and 40 km/h. The speed is less during the austral summer, although pronounced after mid-day and calming during the night.

Figure 5-1 Average wind speed (km/h) and direction, measured at the Taca Taca station



5.4.7 Dust

In view of the prevailing wind direction illustrated in Figure 5-1, dust generation will emanate from the waste rock dump, presenting a climatic and environmental alteration to the operations setting and to the surrounding environment.

SRK Consulting (SRK, August 2025) completed a dust emission study utilising the services of a specialist subconsultant, Agreenco Environmental Projects Pty Ltd (Agreenco). Agreenco adopted an average wind speed of 9.2 m/sec (33 km/hr) as recorded at the Arizaro station, in addition to a modelled speed of 5.73 m/sec (20.6 km/hr) at the location of the waste rock dump.

From their modelled worst-case scenario, Agreenco advised that dust fallout will be limited to within 10 km downwind of the waste dump, whilst the risk of dust affecting surrounding receptors such as the Tolar Grande community, the closest settlement located at 35 km distant, was advised to be nil.

5.5 Hydrological and hydrogeological setting

The Project area is located in an arid environment characterised by very low rainfall and high evaporation rates. Taca Taca lies on the western margin of the Siete Curvas basin, an extensional basin in which the 80 km long and 30 km wide Salar de Arizaro occupies the western third. This basin is bordered to the north and south by major northwest trending volcanic lineaments (Ausenco, May 2016). The basin is a closed hydrologic system in which all of the water that enters it stays within the basin unless lost through evaporation.

The Salar de Arizaro is reportedly the sixth largest salar in the world and covers an area of about 1,600 km. Rainfall and snow at higher elevations are the major source of water run-off in the region. This run-off infiltrates the subsurface and flows down into the extensive salars, or in some cases into thick gravel basins where it has accumulated over time.

Groundwater in the Project area occurs in aquifers located in alluvial environments (i.e., gravel beds) located in the valleys and basins surrounding the Salar de Arizaro. With the groundwater flowing into the salar, brine is commonly found in the clastic (lacustrine) sediments and in the overlying evaporate deposits (M&A, November 2018).

5.5.1 Water quality

The process plant will be the largest water consumer for the Project. About 22% of its requirements will be met with brine water sourced from the Salar de Arizaro and to a lesser extent, from open pit dewatering and depressurisation of the pit slopes[5]. It has been assumed that brine water at up to 300,000 mg/L TDS (total dissolved solids) could be used for milling and rougher flotation.

Table 5-6 summarises the results of brine water quality testing from Salar de Arizaro samples, from *plumas verde* basin samples, and from borehole samples collected within the general open pit mine area.

Table 5-6 Summary of brine water analyses

Parameter	Unit	Salar de Arizaro	Plumas Verde	Pit Water
pH	pH	7.1	8.3	7.05
Conductivity	uS/cm	>200 000	17 420	241 700
TDS	mg/l	255 500	10 700	317 596
Alkalinity	mg/l	49	129	-
Bicarbonate	mg/l	59	158	-
Calcium	mg/l	2 510	169	1558
Magnesium	mg/l	1 350	82	706
Chloride	mg/l	153 000	6 120	184 600
Sulfate	mg/l	3 900	310	
Nitrate	mg/l	600	30	
Sodium	mg/l	84 400	4 120	70 277
Potassium	mg/l	3 030	77	

As part of the geotechnical investigations for the proposed TSF location, three trenches were excavated on the Salar de Taca Taca in 2018. These trenches were dug to a depth of 2.5 m to 3.0 m and over a 3.8 m length.

[5] Relative to the 2021 Technical Report, the brine water requirement has reduced, and the freshwater requirement has increased.

Following excavation, brine inflow filled the trenches to within 0.4 m of the surface over a period of several hours.

In addition, trial pits were dug along the length of a proposed new airstrip on the Salar de Arizaro, approximately 2.5 km east of the proposed brine borefield. A total of 35 trial pits were excavated and these showed the intersection of brine to be at a consistent 1.0 m to 1.1 m depth, which is considered to be representative of depth-to-brine in the salar.

5.6 Environmental setting

Further to the location and climatic conditions described above, the environmental setting for the Taca Taca Project can be summarised as follows:

- The site is located in a cold and arid zone, exposed to strong solar radiation and winds.
- The site has hydrographic characteristics typical of the Andean regions, with little permanent surface water, although minor ephemeral and seasonal streams are formed from summer thaw, rainfall, hail and snowfall.
- Despite the elevations of the site, there are no forms of a glacial or periglacial environment.
- Soils have no agricultural value in the salt flats of Arizaro, although at Taca Taca and the humid area of Valle de Arizaro there are vegetal borders.
- At nearby Caipe there is azonal vegetation which grows in wetlands, associated with water contribution.
- Otherwise, in dry areas, there is zonal vegetation, pediments and alluvial cones, depending on climatic conditions and soil type.
- The Project is located within the "Los Andes Natural Wildlife Reserve." It is one of the three largest protected areas in Argentina and the largest in the Salta province. The Reserve has been classified for multiple use, including for mineral exploration and development activities.
- Although fauna is scarce, vicuñas, certain lizards and birds (yellow winged pigeons, goldfinches, crested duck, peregrine falcon, Baird´s sandpiper, rufous-bellied seed snipes, puna miner and puna ground tyrant) and mammals (yellow-rumped leaf-eared mouse, Bolivian grass mouse, vicunas) have been found in the area adjacent to the Project.
- A critical habit study is currently underway to determine whether, if any, impacts from the Project would influence these habitats.
- Habitats can be classified into three categories: modified, natural and critical habitats:
  - the critical areas are Valle de Arizaro (of greater flora richness), *plumas verdes* (artificial dug-out pools) and Caipe
  - these habitats are not in the immediate Project area; they are habitats located at potential water supply sources.
- In respect of archaeology, no structures of significance have been found in the Project area and the site lacks relevance due to the scarce sedimentary outcrops and absence of fossils.
- An Integrated Management Plan for the area permits development and exploitation of natural resources including open pit mining and associated infrastructure.

5.7 Access to the Project by road

The Project site is readily accessible by road from the city of Salta, which is approximately 230 km to the east and is the nearest major population centre in Argentina. The road distance from Salta, via San Antonio de los Cobres, Cauchari and Tolar Grande, is about 400 km. Figure 5-2 shows the access route on the Argentine side of the border, along Provincial Route (RP) N°27, turning off National Route (RN) N° 51 at Cauchari. Route 51 is paved from Salta until San Antonio de los Cobres. There is a relatively good dirt road from San Antonio de los Cobres to Paso de Sico is (there is a plan at the provincial and national level to pave this stretch, first until Cauchari, and then up to Paso de Sico). Route Nº27 is a good dirt road that has been improved from Cauchari (where this route starts), until Salar de Pocitos, with plans to continue improving this stretch along with the rest of the road.

From the Chilean side, the Project site can be accessed via routes 23 and 27, passing through Calama and San Pedro de Atacama to reach Paso de Jama at the border, and then south via (RN) N°52 to Cauchari and onwards to Tolar Grande and the Project site (Figure 5-2). Alternatively, Route 23 continues from San Pedro de Atacama and the Argentine border could be crossed at Paso de Sico before continuing to Cauchari, Tolar Grande and the Project site (Figure 5-2). All of these roads are fully paved. The more direct road access to the site from the west is on lesser roads, either via route B-55 and Paso Socompa (paved up to Escondida, then unpaved) or via Baquedano, Peine, Socaire and then south before Paso de Sico (paved) (Figure 5-2).

5.7.1 Road access through Argentina

East of Salta, the Project is accessible to the Argentinian port city of Zárate, which is near to the capital, Buenos Aires. The journey from the port, for vehicles of less than 5 m width and height, is via a circuitous route from Zárate to San Antonio de Areco along RN Nº 193 and 8, then to Pergamino and Melincué (RN Nº 93), onwards to Chabas and Rosario (RN Nº 33), and then to Rafaela, Santiago del Estero and eventually to Salta (RN Nº 34 and 9). A report by *Transportes Universales* SA (TUSA, June 2011) shows the route and informs that the travel time is three to five days, depending on the number of truck trailers.

For overweight loads in excess of 100 tonnes, and due to many load limited bridges, the road journey between Zárate and Salta is longer, at seven to eight days (TUSA, June 2011). There is also a load limitation on RN Nº 51, for the continuing journey between Salta and San Antonio de los Cobres. TUSA (June 2011) advises of a long deviation for overweight transport, extending from Salta to San Salvador de Jujuy and Purmamarca (RN Nº 9 and 34), and from there south to San Antonio de los Cobres (Figure 5-2).

Beyond San Antonio de los Cobres, at Cauchari, the Project access road branches off onto Provisional Route RP Nº 27, continues to Tolar Grande and the Project site, and then onwards to Socompa on the Chilean border.

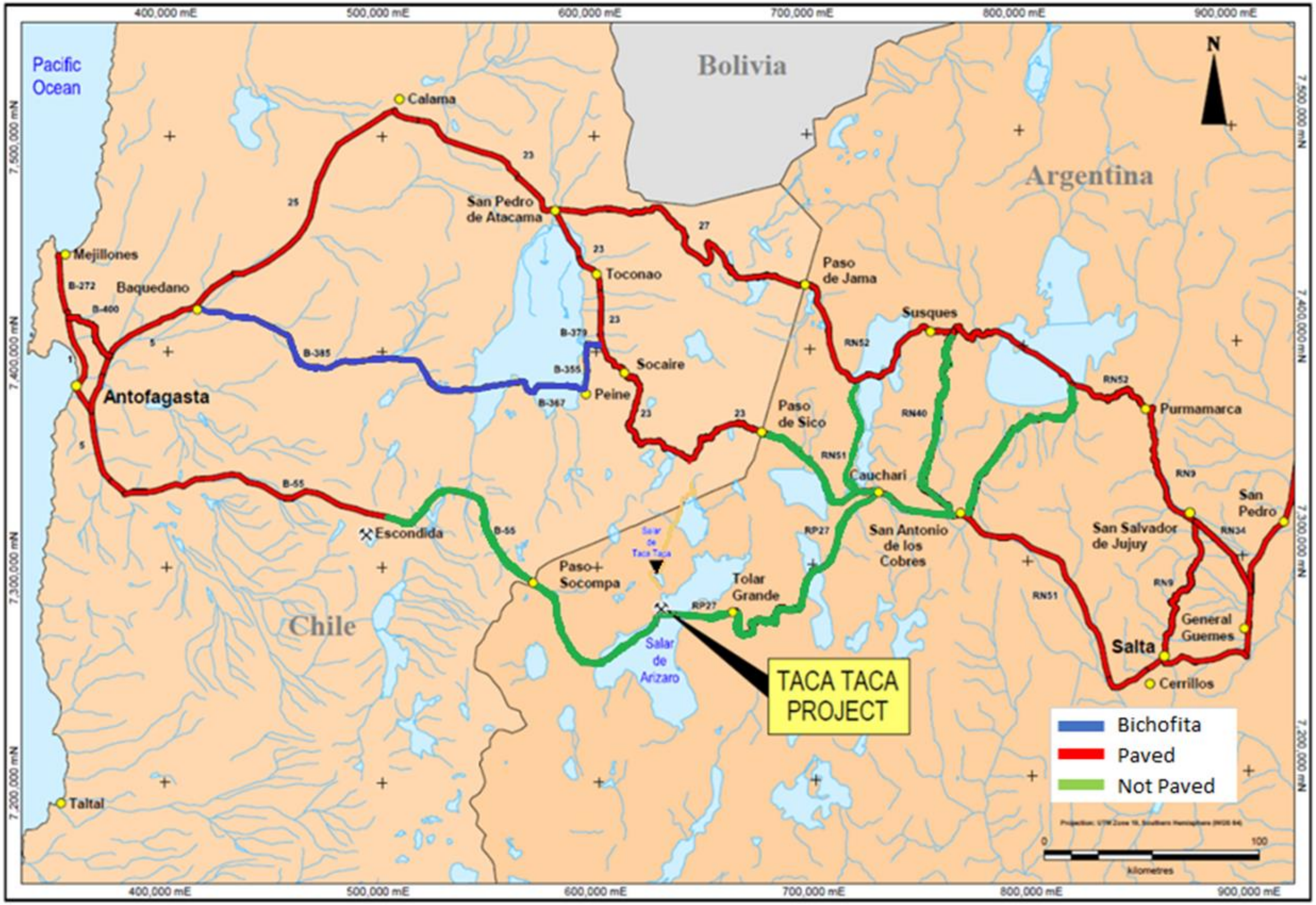
5.7.2 Road access through Chile

Regarding road access to the Project site from potential Chilean ports on Mejillones Bay, the journey for load and width/length limited trucks is via Calama, San Pedro de Atacama, Paso de Jama, and then across the border to Cauchari and south to the Project site, through Tolar Grande (Figure 5-2). TUSA (June 2011) advises that the transit time is about two days.

The shortest routes to the site from the west are on lesser roads, either via route B-55 and Paso Socompa or via Baquedano, Peine, Socaire and then south through Paso de Sico (Figure 5-2). These lesser roads would require upgrade and the border crossings at Paso de Jama, Paso de Sico and at Paso Socompa are at altitudes which make them impassable after heavy snowfall. Figure 5-2 shows another route, extending off from San Pedro de Atacama to Socaire and then to an intersection before Paso de Sico on the border, where a minor road branches off to the south and arrives into the Project site on the west side of the Salar de Taca Taca (Figure 5-2). The advantage of this particular route on the Chilean side is the lower altitude terrain.

Customs clearance for road transport entering/leaving Argentina is available at Paso de Sico. This enables direct logistical access to/from site via Baquendano, Peine and Socaire in Chile. Although this route is paved, it includes narrow width stretches, sharp curves and subsidence areas over the Salar de Atacama (Figure 5-2). For the route approaching the Project site through the border post at Paso Socompa, there is currently only a police control point. There is an existing customs clearance facility which is currently not operational due to low traffic volume.

Figure 5-2 Road access to the Project site



5.8 Access to the Project by rail

The Project site is located within 5 km of a narrow gauge (1 metre) railway line between Salta and Antofagasta. After a time of near dereliction in certain parts, this line is being refurbished to allow resumed services over its full length. The existing rail line has been refurbished to the extent that since 2016, reagents are being railed from Chilean ports to lithium mining companies operating in Salta Province. Lithium carbonate and lime are being railed from Argentina across into Chile. The volumes are currently insignificant in comparison with the future freight requirements for Taca Taca, should they be railed.

Figure 5-3 shows the railway access route on the Argentine side of the border, between Salta and Taca Taca, whilst Figure 5-4 shows the route across the border from Socompa, and then on to Antofagasta and the port city of Mejillones in Chile. Rail traffic on the Argentine side is limited to an axle loading of 17.5 tonne/axle, whereas the line on the Chilean side can be loaded to 16 to 17.5 tonne/axle.

5.8.1 Rail access through Argentina

On the Argentine side of the border, the railway is operated by state-owned *Belgrano Cargas y Logistica SA* (Belgrano). The line itself and related infrastructure, is owned by *Administración de Infraestructuras Ferroviarias Sociedad del Estado* (ADIFSE). Whilst there are numerous stations along the line between Salta and Taca Taca, only the stations at Salta, San Antonio de los Cobres and Tolar Grande are manned as track maintenance depots. Repair and maintenance workshops for locomotives and rolling stock are located at Güemes, east of Salta.

5.8.2 Rail access through Chile

On the Chilean side of the border, the line between Socompa on the border and Augusta Victoria is operated by *Empresa de Transporte Ferroviario SA* (Ferronor SA), a private company. Another private company, *Ferrocarril de Antofagasta a Bolivia* (FCAB, a subsidiary of Antofagasta PLC), operates the line between Augusta Victoria and Antofagasta/Mejillones. Both companies have a track access agreement, each allowing the other to use its track, for payment of a fee.

5.9 Other transport links

A regional airport exists at Salta from where there are regular daily flights to and from the Argentine capital. From this airport there are also flights to and from other cities in Argentina, as well as regular international flights to and from Lima and Panamá City.

Other mining operators in the region have established airstrips near their operations, and there is growing trend to consider air transportation for personnel to and from site. The nearest existing airstrip to Taca Taca is operated by Fortuna Mining for the Lindero Gold heap leach operation. It supports small planes and is approximately one hour by road from Taca Taca.

As part of the Taca Taca Project development it is proposed to construct a 4 km long x 150 m wide airstrip as an elevated, brine sealed pad on the Salar de Arizaro. This airstrip, together with a modest terminal, will support chartered flights using a Dash 8 turboprop plane, servicing four return flights per day, each transporting 65 passengers with their hand luggage. Transit time from the Taca Taca camp to the terminal is expected to be approximately 15 minutes, and the flight time from Taca Taca to Salta is expected to be in the region of 20 to 25 minutes.

Figure 5-3 Rail access to the Project site, through Argentina

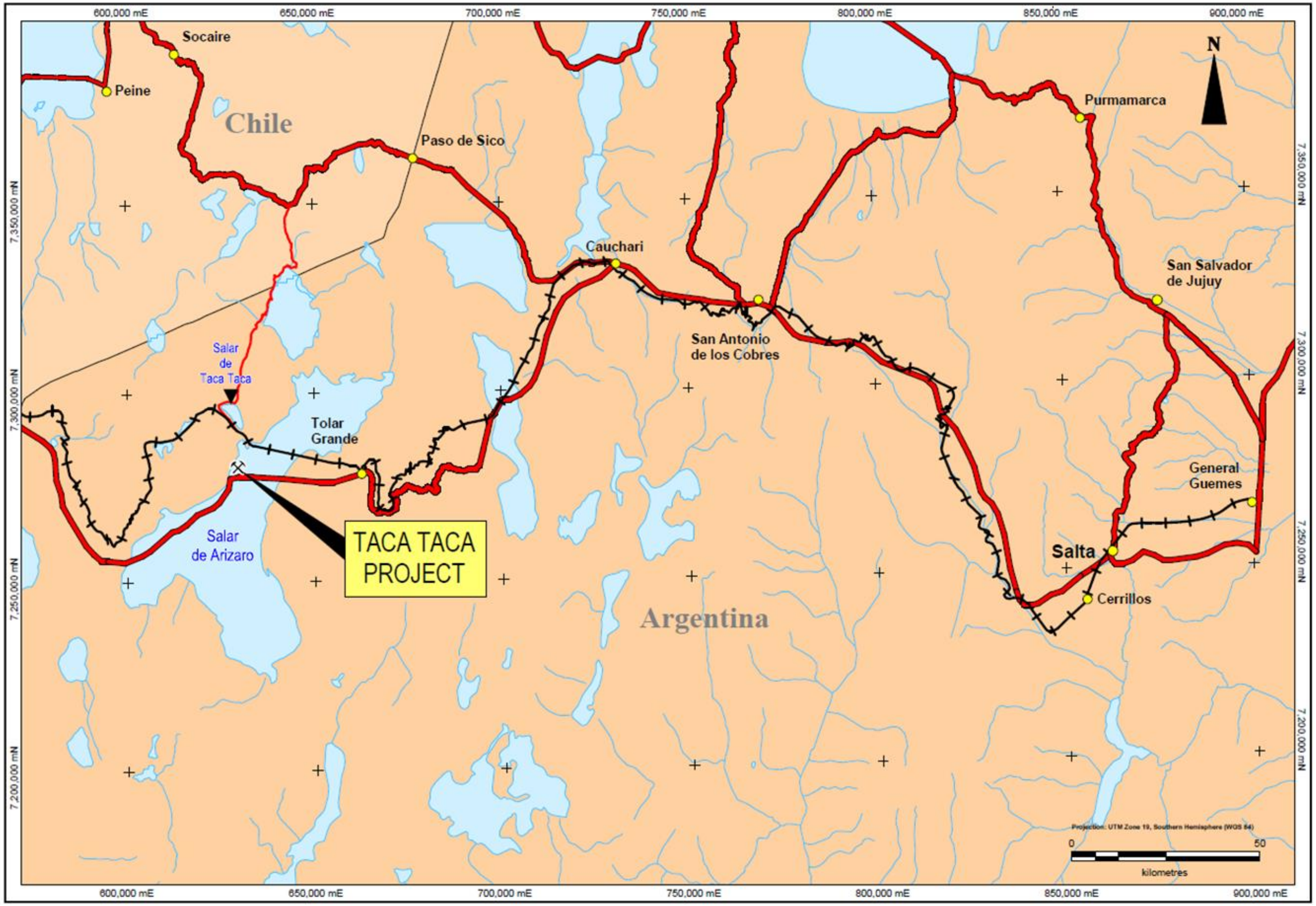
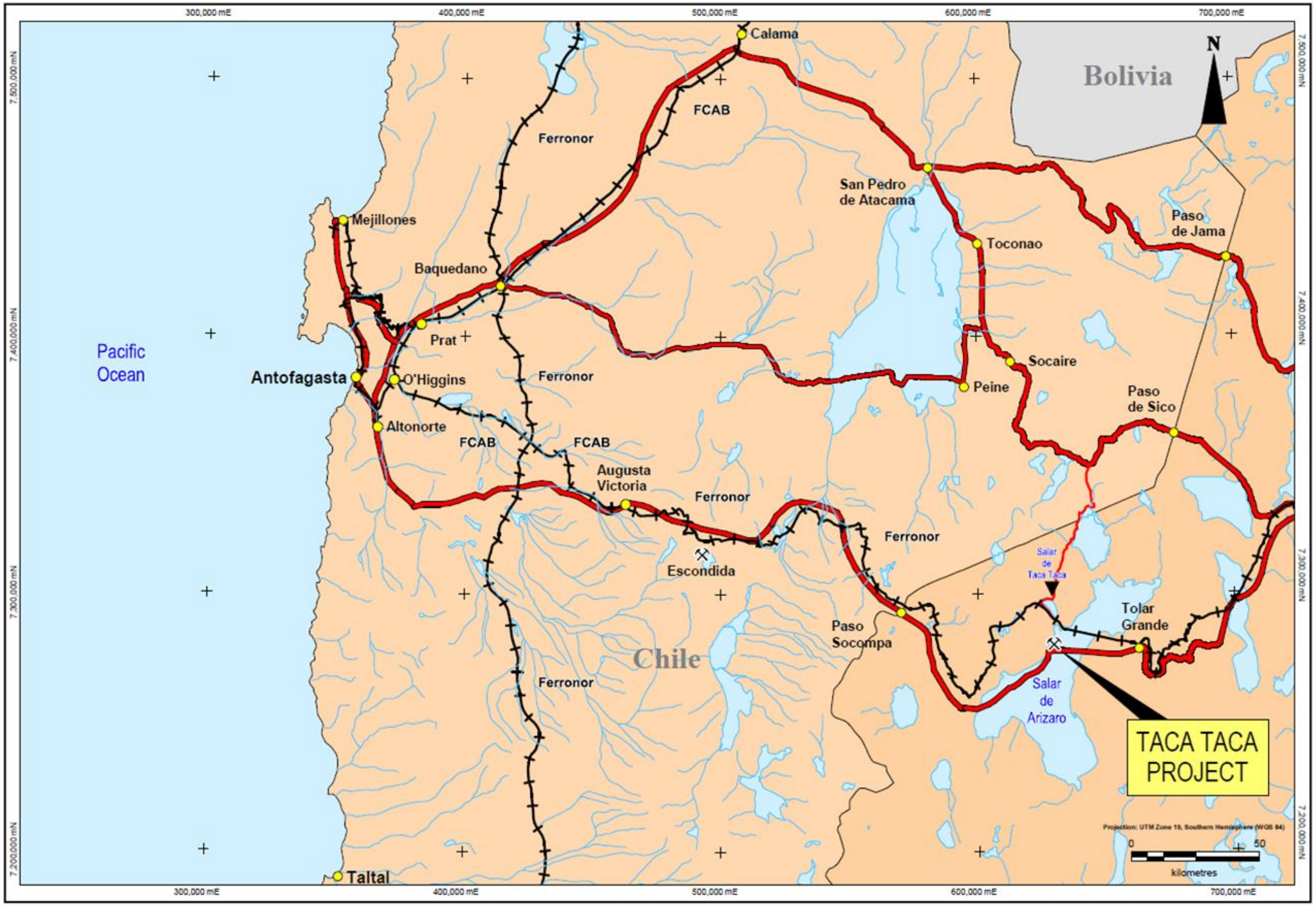


Figure 5-4 Rail access to the Project site, through Chile



5.10 Proximity to population centres

The nearest population centre to the Project site is Tolar Grande, located 35 km to the east. Tolar Grande was established to provide services to the railway line between Antofagasta and Salta; the village has a population of around 250 people. With approximately 558,000 people according to the 2022 Census, the city of Salta is the nearest major population centre in Argentina. Businesses in Salta Province could provide basic goods and services for Project development and during operations.

5.11 Power supply

The nearest power transmission line to the Project site is to the north in the vicinity of Olacapato, near Cauchari (Figure 5-5). This is a 345 kV line from the Cobos generating station in Salta Province, extending to Los Andes in Chile. The line is privately owned and operated by Termoandes SA (Termoandes).

Compañía Administradora del Mercado Eléctrico Mayorista SA (CAMMESA) is an Argentine company which operates and co-ordinates the wholesale energy market in the country, and specifically, the Argentine Interconnection System (SIN). CAMMESA is responsible for coordinating power generation by several separate entities, including Termoandes, and for regulating the supply and wholesale market for electric power.

5.11.1 345 kV transmission line

Hugo Gil Figueroa & Asociados (Hugo Gil, 2011) completed a prefeasibility energy supply report for Lumina in 2011 (with an update in 2013). At the time of the Hugo Gil report, *Termoandes* could only export power to Chile and was therefore not part of the SIN national supply grid.

A power supply study and report were completed by *Tecnolatina SA* in November 2019 (*Tecnolatina*, 2019) and updated in 2023 and 2024 in which it was concluded that a straight-forward connection to the existing 345 kV transmission would not compromise the existing transmission and would add stability to the national grid.

5.11.2 Solar power generation

Transmission capacity in the region is currently over-supplied owing to two recently commissioned solar power farms, one of which is a 300 MW photovoltaic solar power farm at Cauchari, and the other a 200 MW photovoltaic solar farm close to the La Puna sub-station. Recent developments in the country have been aimed at deregulating the electricity supply market and reducing CAMMESA's intervention in contracts between private companies. These reforms could open the door for new dedicated transmission lines and for securing additional solar power farms providing renewable energy to the Project.

5.11.3 Natural gas pipeline

The Atacama Gas Pipeline (AGP) extending between Salta Province and Mejillones in Chile passes approximately 210 km to the north of the Project site. This pipeline has a capacity of 8.5 Mm^3/day. The La Puna Gas Pipeline branches off the AGP near Susques (Rio Las Baras) and extends to Salar de Pocitos, which is about 130 km from Taca Taca. The La Puna pipeline is owned by the Province of Salta and is operated by *Conta SRL*. The La Puna pipeline has a capacity of 0.3 Mm^3/day.

Hugo Gil (2011) is of the view that the La Puna natural gas pipeline will be of inadequate supply capacity for potential on-site power generation. The existing pipeline route, in relation to the Project site, is shown in Figure 5-5.

5.12 Water supply

Figure 5-6 shows the major catchment areas within the Siete Curvas basin, four of which (Valle de Arizaro, Socompa, Valle de Chaschas and Valle de las Burras) have been identified for fresh water supply to the Project. The supply of water to the Project is planned to be from borefields, yielding freshwater from these prospective sedimentary basins, in addition to high salinity brine water abstracted from borefields or from trenches in the adjacent Salar de Arizaro.

There are additional fresh water basins further afield than the four mentioned above. Investigation drilling and modelling is continuing as the Project engineering phase proceeds, to confirm the sustainability of the identified supply sources.

5.13 Availability of personnel

Labour requirements for the Project would largely be sourced from within Argentina, although a cohort of management, engineers, and the construction and operations workforce would comprise skilled personnel with experience gained at other Company sites and projects globally.

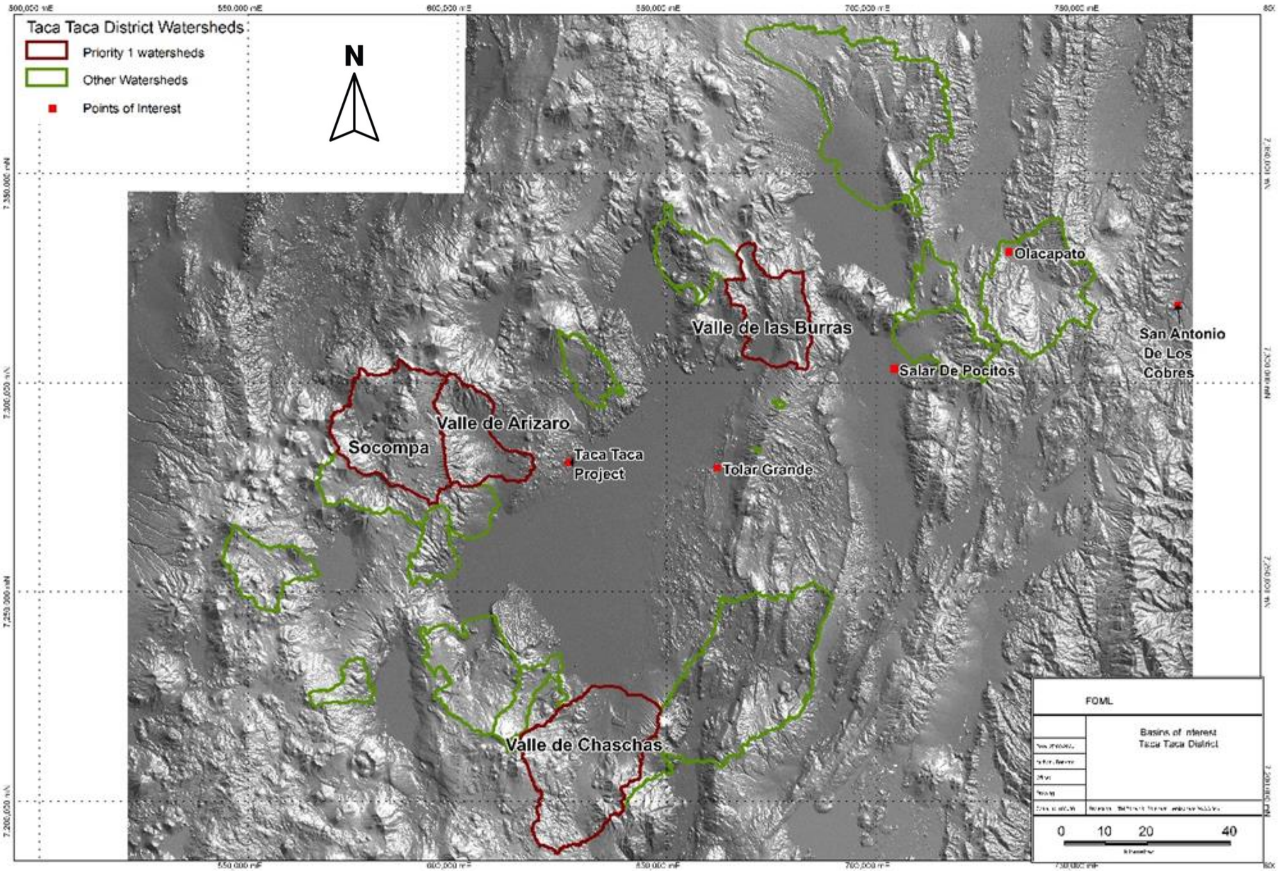
Within Salta Province, and other than at the nearby Lindero gold project, there is little direct work experience with metallic mines. Nevertheless, and where possible, personnel and selected contractors would be recruited from within the Province, including from Tolar Grande, Pocitos, Olacapato, San Antonio de los Cobres, and Salta City.

Elsewhere, qualified Argentine engineers and geologists have been involved in hard rock mining projects and operations in the country since the 1990’s, as well as in neighbouring Chile and Bolivia. Large scale open pit copper mining operations began at Bajo de la Alumbrera in the Catamarca Province in the late 1990’s, and there is considerable experience in gold and silver mining elsewhere in the country, particularly in Jujuy, Catamarca, Santa Cruz and San Juan Provinces. The Company may also recruit experienced personnel from lithium brine projects in the country.

Additionally, the southern provinces where there is robust oil and gas industry, could be a potential source of professionals with technical expertise adaptable to a large scale open pit mining and processing operation. Argentina has a strong academic foundation supporting the mining sector. Universities offering mining-related programmes in different provinces serve as key recruitment sources for young professionals. The country’s NOA region (*Noroeste argentino*) further contributes to the available expertise, offering a diverse range of technical degree programmes, and also supporting long-term workforce development for the mining industry.

Figure 5-5 Power line and gas pipeline routes

Figure 5-6 Regional catchment areas near to the Project



5.14 Communications

The Project site has no fixed telecommunications infrastructure. The only available connectivity options are satellite services and microwave links. While these have been functional during the permitting and exploration phases, they will be insufficient to support the increasing communications demands of the construction, commissioning, and operational phases.

5.15 Sufficiency of surface rights

For the most part, the entire proposed Project infrastructure including the open pit, the waste dump, ore stockpile, processing plant and TSF are all located within the CASA mining concessions.

A small, north western segment of the open pit crosses onto the *Mina Francisco* joint venture concession. A small area in the centre of the TSF is covered by concessions held by third parties. The affected area is considered small enough for non-protracted agreements with those parties.

ITEM 6 HISTORY

6.1 Prior ownership

Copper-gold-molybdenum porphyry-style mineralisation was discovered at Taca Taca in the late 1960s. Lumina first acquired an interest in the property when shareholders of Global Copper Corporation approved a corporate reorganisation on 1st August 2008, ultimately resulting in the acquisition by Lumina of 100% of the shares of CASA and a 100% interest in the property.

In August 2014, the Company acquired Lumina and its Taca Taca asset which was then in an advanced exploration phase.

6.2 Exploration and development work undertaken by previous Project owners

Fabricaciones Militares reported the discovery of porphyry-style, copper-gold-molybdenum mineralisation at Taca Taca in the late 1960s. After three diamond holes were drilled into the leached cap by Falconbridge in 1975, prospecting on the property remained dormant until 1990.

Between 1994 and 2008, Gencor, BHP, CASA, and Rio Tinto each conducted exploratory drill programmes, outcrop and trench mapping, and geophysical surveys (Table 6-1). Four main types of mineralisation were targeted:

1. Remnant oxide and supergene copper within the leached cap.
2. Supergene porphyry copper enrichment underneath the leached cap.
3. Gold-copper bearing veins to the north and west of the porphyry.
4. Exotic copper mineralisation beneath the Salar de Arizaro.

Significant supergene mineralisation directly beneath the porphyry leached cap was discovered by BHP in 1997. Rio Tinto successfully intersected deeper hypogene mineralisation in 2008 but results did not meet their corporate criteria to warrant further expenditure.

In 2008, Lumina acquired the property and completed a Titan 24 geophysical survey to aid with early target identification. After early drilling intersected relatively high-grade shallow mineralisation, a more systematic drill programme was undertaken. Between 2010 and 2012, 155 diamond (DD) holes and 128 reverse circulation (RC) holes were drilled resulting in the delineation of the copper-gold-molybdenum resource.

During an exploration history spanning 45 years, a total of 167,375 metres has been drilled in 450 holes. Following the acquisition of Lumina and the Taca Taca asset in 2014, the Company has continued with the collection and interpretation of geological data for the purpose of ensuring confidence in subsequent Mineral Resource estimates.

Table 6-1 Exploration activities of previous Project owners

Year	Active Company	Company Agreements	Target Type	Drill Programs	Non-Drilling Activities	Campaign outcomes
1967	Fabricaciones Militares	NOA Minero (government sponsored program)	Cu porphry deposit	-	Mapping	Discovery of porphyry copper mineralisation
1975	Falconbridge	-	Cu porphry deposit	3 DD holes	-	Drilled into the leached cap. Abandoned the property.
1990-1995	Gencor (GAMSA)	Taca Taca S.A. (TACSA) in agreement with Recursos Americanos Argentinos (RAA) who explored the property with GAMSA, a subsidiary of Gencor	Epithermal Au mineralisation (to the north) and Cu porphyry	18 RC holes	Petrographic Studies	Mineralisation considered narrow and discontinuous. Gencor returned the property to RAA, who returned it to TACSA in 1995
1995	Corriente	Corriente Resources Inc. (Corriente) signed an exploration agreement with TACSA	-	-	-	-
1996-1997	BHP/Corriente	Corriente entered a joint venture with BHP minerals	Supergene Cu mineralisation below the porphyry leached cap	35 DD holes	36.8km TEM survey, IP survey, topographic survey, geochemical sampling, petrography, mapping	Discovery of supergene mineralisation under the leached cap to the northwest side of the deposit. Target did not meet BHP's corporate criteria. Property was returned to Corriente.
1998-1999	Corriente ASA	Corriente and TACSA merge into Corriente Argentina S.A. (Corriente ASA)	Shallow exotic and supergene Cu in areas peripheral to the porphyry and below Salar de Arizaro	14 DD holes, 80 RC holes	Ground magnetics and radiometric surveys, trenching, geochemical sampling	Mineralisation intercepts were narrow and discontinuous.
1999	Rio Tinto	Rio Tinto options property from Corriente ASA	Remnant in situ copper oxides in the porphyry leached cap and exotic Cu below Salar de Arizaro	9 RC holes	136km ground magnetics, radiometrics (K/Th), and mapping	Target did not meet Rio Tinto's corporate criteria. Property was returned to Corriente ASA.
2003	Lumina Copper	Acquires 100% interest in Corriente ASA	Near surface Cu oxides	-	Mapping and sampling of surface oxide copper zones	Target generated but not followed up
2005	Global Copper	Acquires property after reorganisation of Lumina Copper	-	-	-	-
2007-2008	Rio Tinto	Rio Tinto options property from Global Copper	Deeper hypogene porphyry mineralisation	8 DD holes	Mapping, radiometric dating, spectral analysis, petrographic studies	Results deemed unfavourable and property returned to Global Copper
2008-2012	Lumina Copper/ Corriente ASA	Lumina Copper acquires property from Global Copper	Supergene enrichment and deeper hypogene porphyry mineralisation	155 DD holes, 128 RC holes	Titan 24 survey to provide targets for early drilling	Delineation of porphyry Cu-Au-Mo resource bringing the project into an advanced exploration phase

6.3 Previous Mineral Resource estimate

Table 6-2 lists the Taca Taca Mineral Resource estimate that was included in the 2021 Technical Report (FQM, March 2021). It is reported using a 0.13% copper equivalent cut-off and the classification was guided by a life of mine pit shell. The Mineral Resource estimate was inclusive of the Mineral Reserve estimate. The tabled October 2020 Mineral Resource estimate below, is now superseded by this estimate dated 31st December 2025.

Table 6-2 Mineral Resource statement as at October 2020, using a 0.13% Cu_{eq} cut-off grade

Classification	Volume (Mbcm)	Tonnes (Mt)	Density (t/m³)	Cu grade (%)	Mo grade (%)	Au grade (g/t)	Cu metal (kt)	Mo metal (kt)	Au metal (koz)
Measured	157.7	421.5	2.67	0.60	0.016	0.14	2,542.8	67.02	1,852.6
Indicated	671.6	1,781.8	2.65	0.39	0.011	0.07	6,908.0	197.52	4,199.5
Measured & Indicated	**829.3**	**2,203.3**	**2.66**	**0.43**	**0.012**	**0.09**	**9,450.7**	**264.54**	**6,052.1**
Inferred	269.4	716.9	2.66	0.31	0.009	0.05	2,206.0	65.15	1,182.7

The reported Mineral Resource excluded gold mineralisation located with the uppermost leached horizon.

6.4 Previous Mineral Reserve estimate

Table 6-3 lists the Mineral Reserve estimate that was included in the 2021 Technical Report (FQM, March 2021) and which is now superseded by the Mineral Reserve estimate that is the subject of this Technical Report. The associated mine plan was developed using the Measured and Indicated Mineral Resource, whilst Inferred Mineral Resource was allocated to waste. Mining assumed conventional open pit operations using truck-and-shovel technology. The estimated Mineral Reserve was determined using metal prices of $3.00/lb for copper, $12.00/lb for molybdenum, and $1,200/oz for gold, with a supporting production schedule derived from the ore and waste mining inventory within a practical pit design produced from a selected pit optimisation shell.

The actual marginal cut-off grade for the Mineral Reserve varied according to the copper recovery assigned to the various mineralogical groupings. However, the overall average marginal copper cut-off grade was in the order of 0.13% Cu_{eq}.

Table 6-3 Taca Taca Mineral Reserve statement, at October 2020

Classification	Tonnes (Mt)	Cu grade (%)	Mo grade (%)	Au grade (g/t)	Cu metal (kt)	Mo metal (kt)	Au metal (koz)
Proven	408.3	0.59	0.016	0.13	2,401.6	63.3	1,749.8
Probable	1,350.2	0.39	0.011	0.08	5,333.1	150.2	3,336.9
Proven & Probable	**1,758.5**	**0.44**	**0.012**	**0.09**	**7,734.7**	**213.5**	**5,086.7**

As part of the ultimate pit design for the Project, there was a small area in the north west that crossed over into the *Mina Francisco* joint venture concession. This encroachment, included in the Table 6-3 inventory, amounted to approximately 1.7 Mt of ore at an average grade of 0.38% Cu.

6.5 Production from the property

To date there has been no production from the property.

ITEM 7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional geology

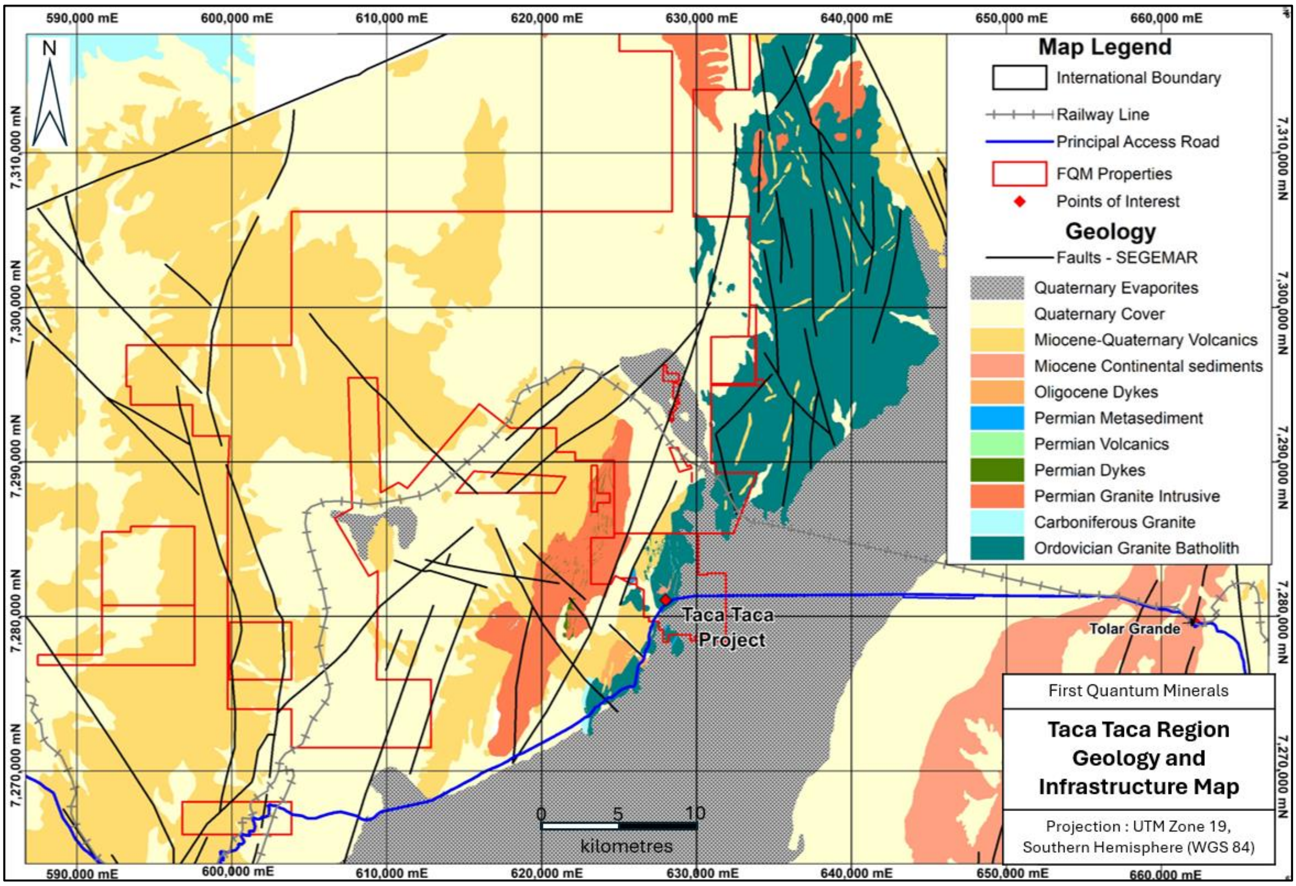
The Taca Taca deposit is in the Puna region of Argentina, an area affected by multiple continental collision and extension events. The region now forms a back-arc basin bounded by high-angle reverse faults developed during uplift of the present-day Andean mountain chain.

The geology comprises granitic batholiths and dykes of granitic composition intruding the crystalline and metasedimentary basement of the Puna, associated with coeval volcanics (Figure 7-1 and Figure 7-3). These units are overlain by Miocene to present-day back-arc basin sediments and volcanics related to Andean uplift and erosion.

The porphyry mineralisation is hosted in the southern part of a more than 50 km long Ordovician batholith, which forms part of a northwest-trending intrusive and volcanic arc extending more than 700 km through northwest Argentina. Later Permian intrusives, volcanics and sediments are related to continental magmatism and back-arc basin formation during a period of passive-margin tectonics.

Oligocene intrusives of the Santa Inés Formation introduced alteration and mineralisation at Taca Taca. These intrusives are interpreted to lie in the back-arc of the Chilean Eocene-Oligocene porphyry belt, with emplacement controlled by regional west-northwest trending cross-arc structures.

Figure 7-1 Regional geology and infrastructure of Taca Taca



7.2 Local property geology

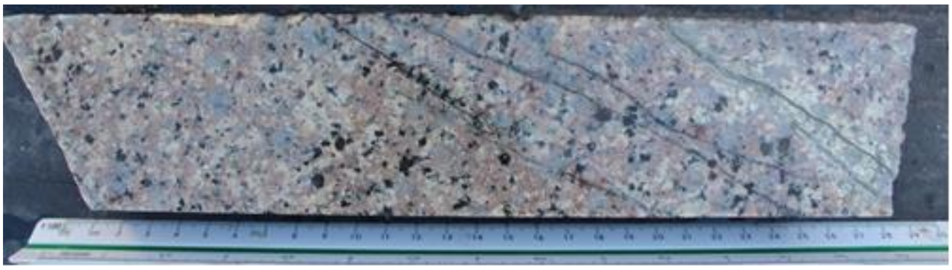
The geological evolution of the Taca Taca deposit area (Figure 7-1 and Figure 7-3) may be summarised as follows:

- During the Ordovician (~440–463 Ma), the Taca Taca batholith, a more than 50 km² granitic intrusion, was emplaced into the Puna basement.
- Late in the batholith's emplacement, aplite dykes and minor dolerite dykes intruded the coarse-grained granite.
- In the Permian (~263 Ma), a second granitic batholith intruded along the western margin of the Taca Taca batholith, accompanied by coeval volcanism from approximately 257–272 Ma. Steeply dipping Permian dykes later intruded both volcanic and intrusive rocks.
- During the late Permian (268 Ma and younger), a mixed sedimentary sequence of shale, sandstone and basal conglomerate was deposited in a small structurally controlled basin along the western side of the batholith.
- In the Oligocene (29.3 Ma), steeply dipping rhyodacitic dykes introduced porphyry copper mineralisation and associated alteration at Taca Taca. Mineralisation is interpreted to have been introduced in three pulses related to texturally distinct dyke phases.
- Regional evidence indicates that the intrusive rocks were then uplifted during the Oligocene and Miocene, as part of the development of the modern Andes, forming the Sierra de Taca Taca range.
- During the Miocene–Pliocene–Pleistocene–Holocene, large areas of the Ordovician batholith, Permian granite and Permian volcanics were covered by lava flows, volcanoclastics and pyroclastics to the north and west of the deposit. The region remains seismically and volcanically active, with basaltic plugs and flows younger than 1 Ma.
- From the Quaternary to the present, deposition of salts and sands in intermontane basins has formed the evaporitic salars of the region.

7.2.1 Rock types

The Taca Taca deposit's mineralisation is hosted within the Ordovician granite (Figure 7-2) batholith and co-magmatic aplite and dolerite intrusives. A smaller, less explored deposit (Taca Taca Alto) is known to exist approximately 4 km to the west and lies outside the Company's concession holdings.

Figure 7-2 Drill core example of Ordovician granite host rock

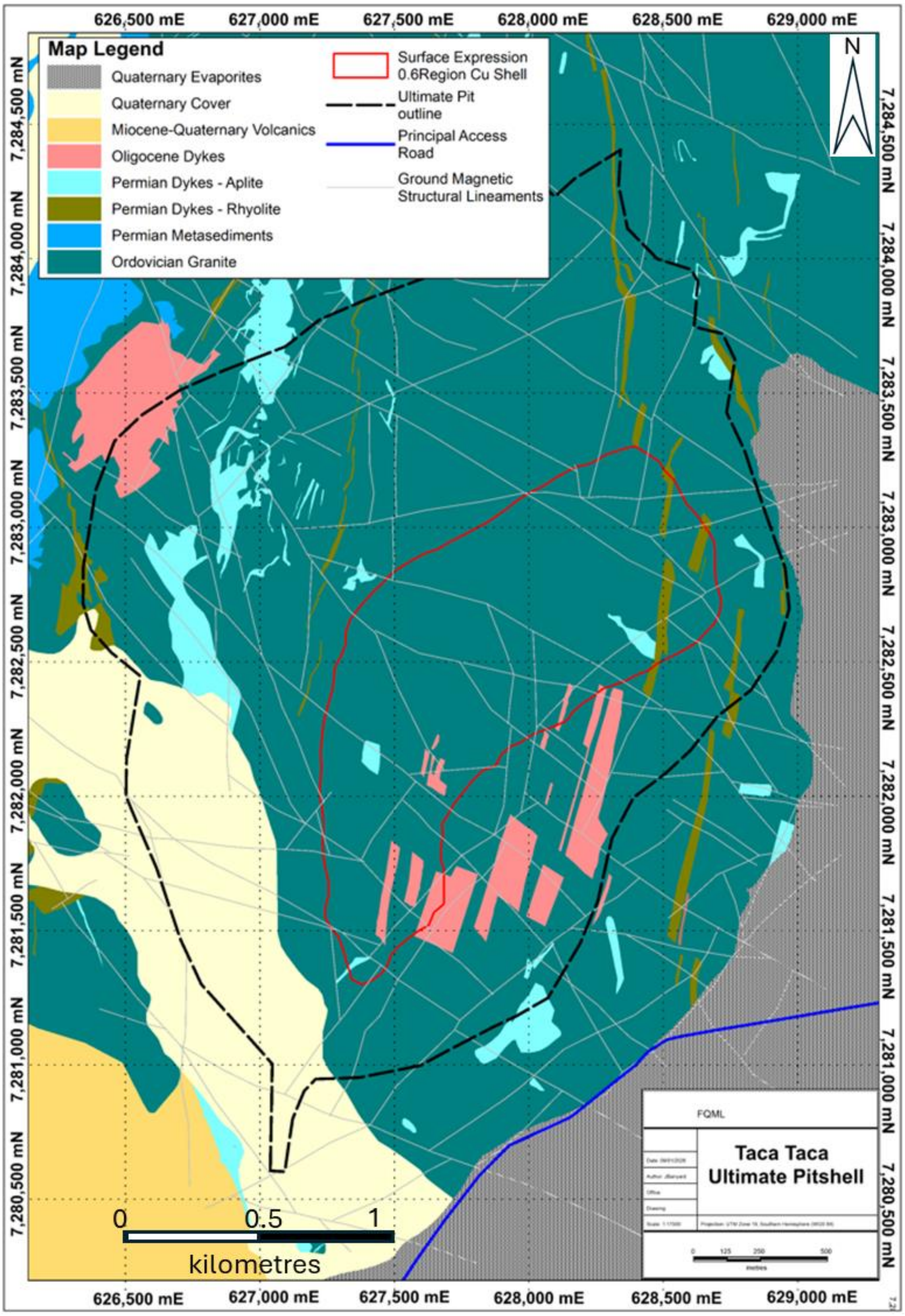


Ordovician Taca Taca batholith

The Ordovician batholith is of granitic to granodiorite composition (Figure 7-2 and Figure 7-3) and forms a prominent range more than 50 km long on the northwest margin of the Salar de Arizaro. It is a medium to coarse grained, equigranular to moderately porphyritic intrusive rock with phenocrystic plagioclase, quartz, K-feldspar, biotite and amphibole. The batholith is cut by co-magmatic aplite sills/dykes and less common, steeply dipping dolerite dykes (Figure 7-3).

Its western margin is in contact with a northeast-trending, Permian granite body, which is locally obscured by less than 50 m thick recent lava flows adjacent to the Aracar Volcano.

Figure 7-3 Taca Taca deposit geology



Permian Granite

A medium- to coarse-grained pink granite outcrops west of the Ordovician granite. It exceeds 20 km in strike and is up to 5 km wide. This granite forms the Taca Taca Massif, which rises to about 4,300 m elevation and hosts the Taca Taca Alto mineralisation. It is intruded by multiple north–south to north-northeast trending rhyodacite and dacite porphyry dykes of Permian age.

Permian volcanic rocks

Permian volcanic rocks in the Taca Taca area comprise a dacite–rhyodacite dominated suite coeval with the Permian intrusive granite. The volcanic package includes volcanoclastics, lava, crystal tuff and ignimbrite, with subordinate dykes, domes and other subvolcanic intrusive bodies.

Permian metasedimentary rocks

Metasedimentary rocks outcrop to the west of the deposit and consist mainly of dark purple shales and siltstones, passing downward into a volcanic breccia at the base. The basal volcanic breccia has a transitional conformable contact with underlying dacite crystal tuff that returned a U–Pb zircon age of 268 Ma.

Permian dykes

Permian-age dykes are present within the Permian volcanic package across the Project area. Around the Taca Taca deposit, these dykes are rhyolitic, several metres wide and trend north–south.

Oligocene dykes

North-northeast striking, steeply dipping dacite-rhyodacite dykes (29.3 Ma) occur locally at the Taca Taca deposit. This orientation is different to the northeast-southwest structural trend of most other dykes in the region and appears restricted to the deposit area. Individual dykes range from less than 1 m up to 100 m wide, as per drilling and mapping data.

In the deposit area they outcrop poorly and in places are difficult to discriminate from the granite host due to intense A-vein quartz stockwork development. The geometry of the dykes is difficult to determine, and some may be more circular, plug-like intrusions.

At least three different Oligocene intrusive events have been recognised:

1. Early-stage rhyodacite associated with early mineralisation. It is characterised by a crowded porphyritic texture of plagioclase and quartz phenocrysts hosted in a shreddy, secondary biotite groundmass.
2. A similar porphyry also associated with early mineralisation but differentiated by a less crowded crystal texture.
3. Late-stage rhyodacite porphyry with a similar phenocryst composition to the earlier events but a less-crowded porphyritic texture in an aplitic to secondary biotite-rich groundmass.

All rhyodacitic phases are strongly altered and are associated with low-grade copper mineralisation.

Miocene-Pliocene-Quaternary volcanics

To the north and west of the Taca Taca deposit, Ordovician and Permian granite and Permian volcanics are covered by recent andesite–basalt lava flows, volcanoclastics and pyroclastic deposits derived from the Aracar and Arizaro volcanoes.

Quaternary evaporites

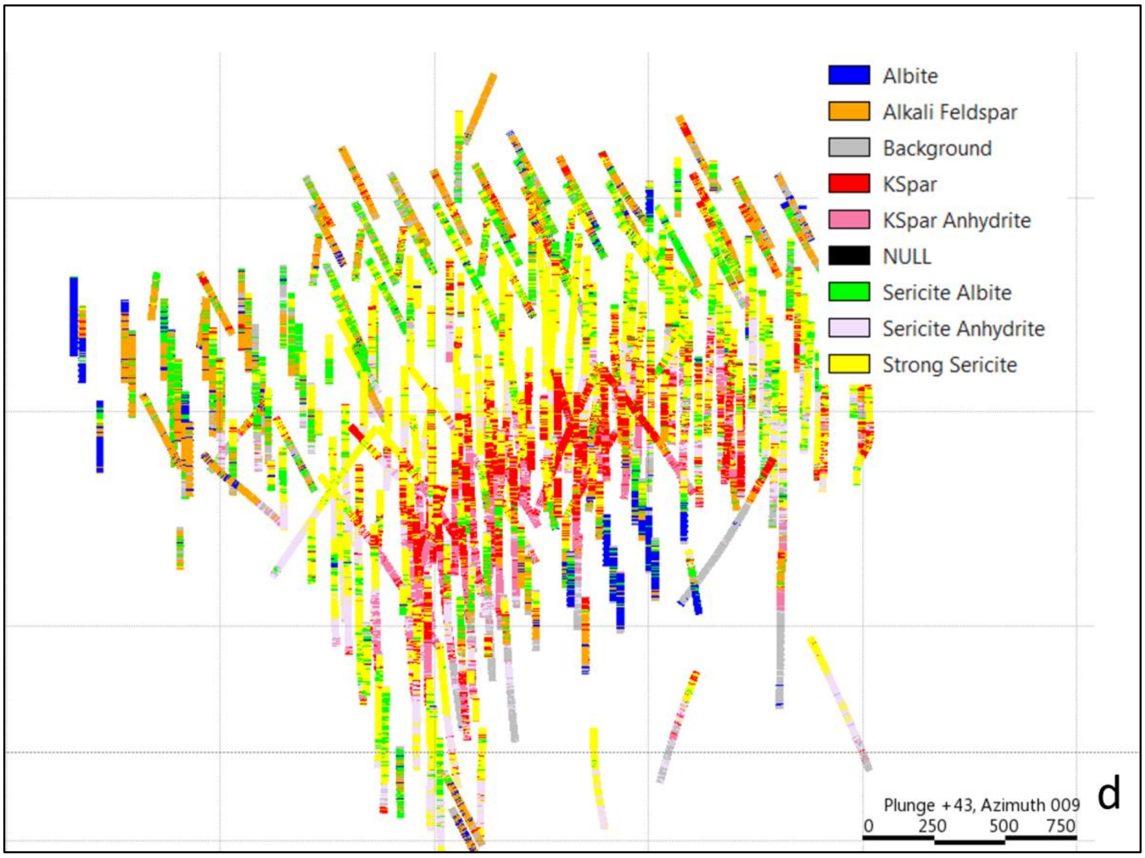
The Salar de Arizaro is a ~1,600 km² salar occupying a structurally controlled closed basin immediately east and southeast of the Project. It locally overlies the Taca Taca host granite. Proximal to the deposit, the upper ~10 m of the salar consist mainly of halite with interspersed sands, capped by a 2–3 m thick surface salt crust.

7.2.2 Alteration

Alteration at the Taca Taca deposit is characteristic of an Andean porphyry Cu-Au-Mo system. Large hydrothermally altered zones range from a central potassic core to peripheral phyllic and argillic zones, with a relatively restricted propylitic halo for a deposit of this size.

At deposit scale, pervasive phyllic (quartz-sericite-phengite-pyrite) alteration commonly overprints earlier assemblages. Geochemical modelling of multi-element data (Figure 7-4) indicates a remnant potassic (biotite to K-feldspar) core coincident with a series of rhyodacite porphyry dykes. Laterally, the phyllic zone grades into propylitic (chlorite ± epidote) alteration at the deposit margins. Supergene argillic (kaolinite-alunite-chalcedony-chalcocite) alteration affects the upper parts of the system and extends to depth along structures.

Figure 7-4 Alteration zonation interpreted from multi-element assay data



Potassic alteration

The innermost potassic core is characterised by abundant coarse secondary biotite replacing mafic minerals and rare K-feldspar in vein selvages. Although largely overprinted by phyllic alteration, it is defined by a distinct geochemical signature and preserved as remnant rafts.

Potassic alteration is associated with weakly mineralised, pervasive 'A-type' quartz vein stockworks. 'B-type' quartz–molybdenite veinlets are common around the outer margin of this quartz-rich core where phyllic alteration overlaps and becomes dominant.

Phyllic alteration

Phyllic alteration is most extensive and is closely associated with mineralisation. Two stages are recognised:

- An early phase of pale green phengite, typically within sericite–andalusite±anhydrite selvages to quartz–copper sulphide 'D-type' veinlets. This phase is associated with an intermediate sulphidation assemblage where chalcopyrite and bornite are more abundant than pyrite. It is associated with higher copper grades and above-average gold grades.
- A later phase of more pervasive white sericite and quartz that overprints potassic, early phyllic and propylitic assemblages. A change from intermediate to high sulphidation led to increased pyrite as disseminations and veinlets, with pyrite–bornite and pyrite–chalcocite–covellite sulphide assemblages.

Phengite and white sericite occur broadly intermixed. Pyrite is present throughout the mineralised zone but increases in abundance outward.

Propylitic alteration

Propylitic alteration is developed at the periphery of the deposit and is largely overprinted by late phyllic alteration. It is characterised by illite–chlorite assemblages with minor epidote and coincides with a strong pyritic halo around the outer edges of the deposit.

Supergene argillic alteration

A 150 m to 300 m thick leach cap overlies the mineralised zone. It is characterised by secondary kaolinite and hematite–jarosite fractures replacing earlier sulphide veins. Copper oxides occur in rare lenses, and a perched secondary sulphide horizon is present on the eastern side of the deposit.

Copper removed from the leach cap was partially redeposited directly beneath along structures within the host rock as secondary sulphides in zones of supergene enrichment. Steeply dipping structures focus localised supergene alteration to depths exceeding 1 km. Secondary kaolinite, chalcedony, alunite and chalcocite veins are associated with these structures.

Alteration and metallurgy

Pyrite is present throughout the deposit. Hypogene sulphide zonation is expressed as increasing pyrite and decreasing chalcopyrite–bornite outward from the centre. A strong pyritic halo with up to 10% sulphur rims the deposit. In the supergene zone, chalcocite commonly overgrows pre-existing sulphides. Pyrite content in plant feed will require monitoring to manage recovery.

Along the east to southeast margins of the proposed pit, plant feed may contain elevated quartz veining. In the absence of SiO_2 assays, quartz vein intensity can only be classified qualitatively into broad relative zones. When treating material from the potassic core, quartz vein intensity will influence grinding and power consumption.

Within the mineralised zone, phyllic alteration contains intermixed white sericite (muscovite) and green sericite (phengite). White sericite is pervasive and associated with pyrite being more abundant than copper sulphides. Green sericite, typically in selvages to quartz–copper sulphide veins, is associated with copper sulphides being more abundant than pyrite. Secondary biotite is preserved where phyllic alteration is less intense.

Existing data does not allow estimation of the relative abundance of white, green or black micas. Metallurgical test work has not identified a clear relationship between mica speciation and performance, suggesting negligible associated risk. As mining progresses, bulk mineralogical composition and potential associations with metallurgical performance should be routinely assessed with the objective of optimising metal recovery at plant scale.

7.2.3 Weathering

Weathering of the upper portions of the deposit has led to the development of a 150 m to 300 m deep copper-depleted leach cap and has facilitated supergene copper enrichment directly beneath.

No saprolite is present in or around the deposit. Alluvial and colluvial fans of gravel-dominated regolith are common around the deposit, with subordinate mud, silt, sand and occasional boulders.

Drill core logging indicates that the uppermost ~30 m of leached saprock is commonly more broken or crumbly. This material is the source of locally derived small-scale alluvial and colluvial deposits.

7.2.4 Mineralisation

Most of the mineralisation is hosted by phyllic-altered Ordovician granite and associated aplite and minor dolerite dykes. Dolerite dykes tend to have relatively higher copper grades due to the abundance of ferrous iron in mafic minerals, which promotes copper precipitation from hydrothermal fluids. Mineralisation is subdivided into an upper leached zone and underlying mixed supergene and hypogene zones.

Leached horizon

A leached zone (also referred to as the "leach cap" or "leached cap") ranging from 150 m to 300 m thick (Figure 7-5) is almost completely depleted of copper mineralisation and is dominated by limonite assemblages consisting of hematite, jarosite, and goethite (Figure 7-6).

Figure 7-5 A plan view showing the relative position and thickness of overlying leached materials

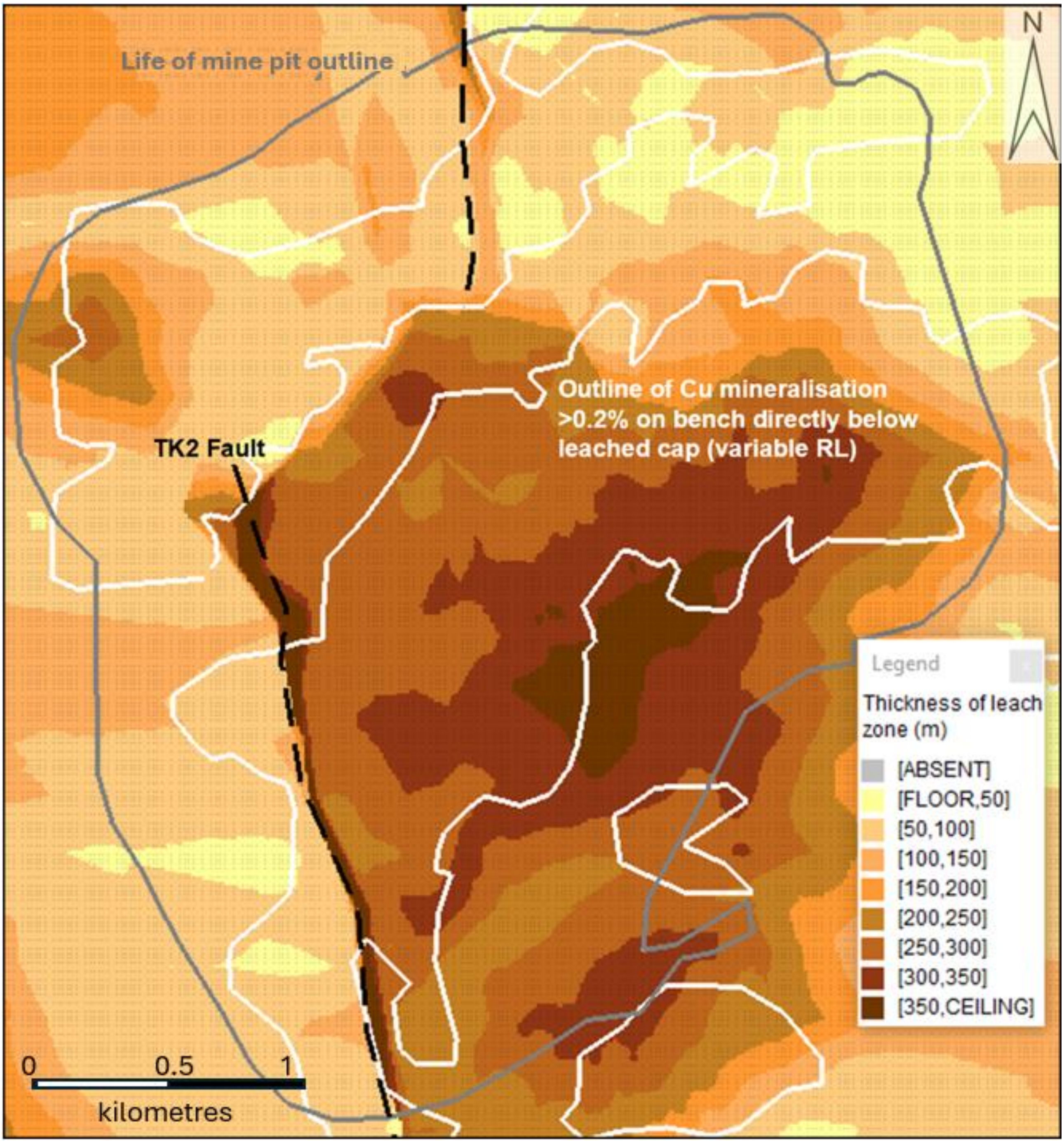
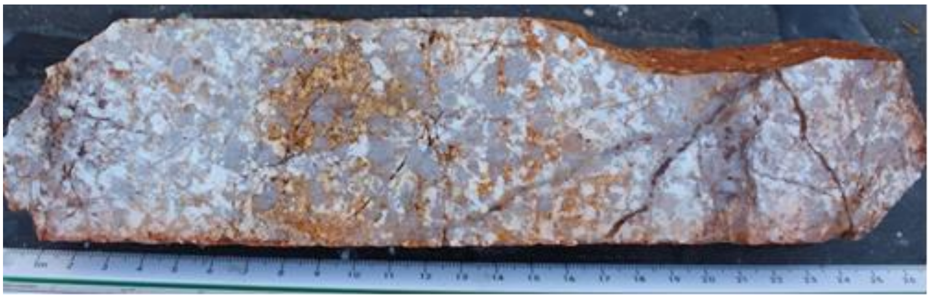


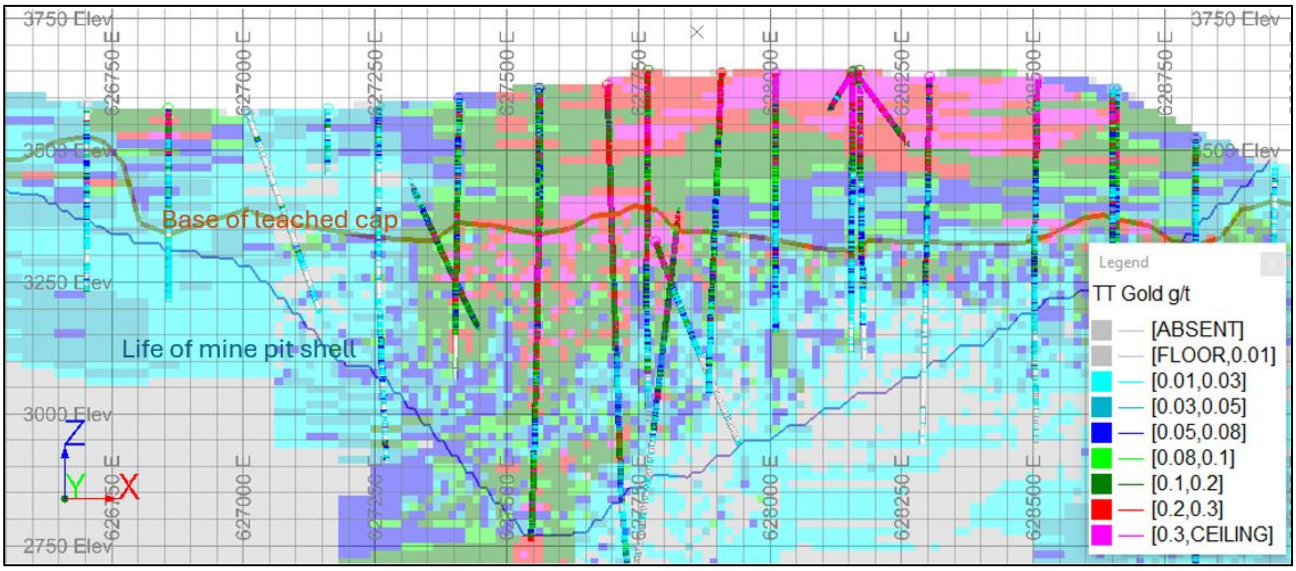
Figure 7-6 Drill core example of granite from the leached cap showing abundant iron oxide



Refractory copper and remnant zones of copper oxide mineralisation are limited to sporadic metre-scale sub-horizontal lenses.

Supergene gold mineralisation is also enriched near the surface above the thickest portions of the leached cap (Figure 7-7). It is not associated or encapsulated with chalcopyrite. Molybdenum enrichment is also evident. Since this zone will be mined during the initial mining of successive pit phases, opportunity exists to stockpile this gold-bearing material for later treatment.

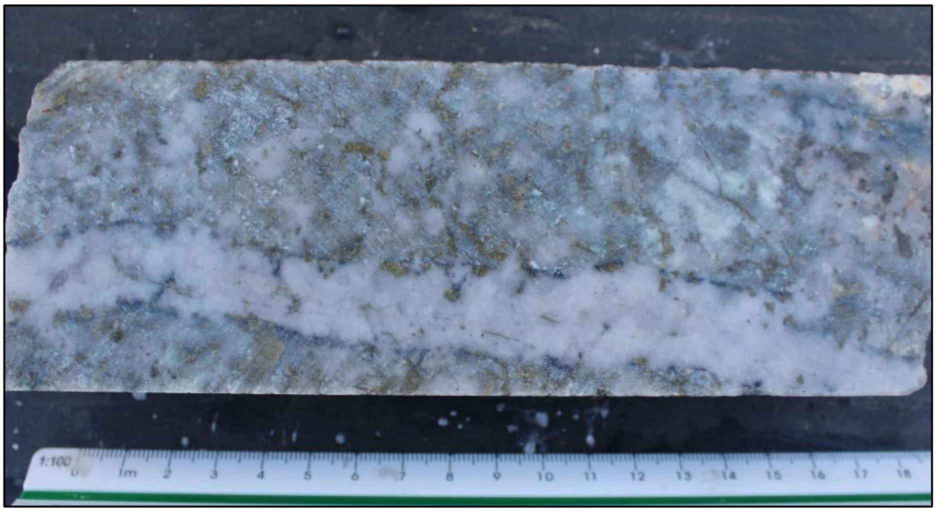
Figure 7-7 A vertical section along 7282620 m N showing the relative position of perched supergene gold mineralisation within the upper portion of leached material



Supergene-hypogene mineralisation

Hypogene copper mineralisation is dominated by chalcopyrite with lesser bornite, chalcocite, covellite and digenite. Copper sulphides occur predominantly as disseminations in sericite vein selvages, in microfractures and intergrown with quartz veins (Figure 7-8).

Figure 7-8 Drill core example of sericitic altered granite with chalcopyrite mineralisation



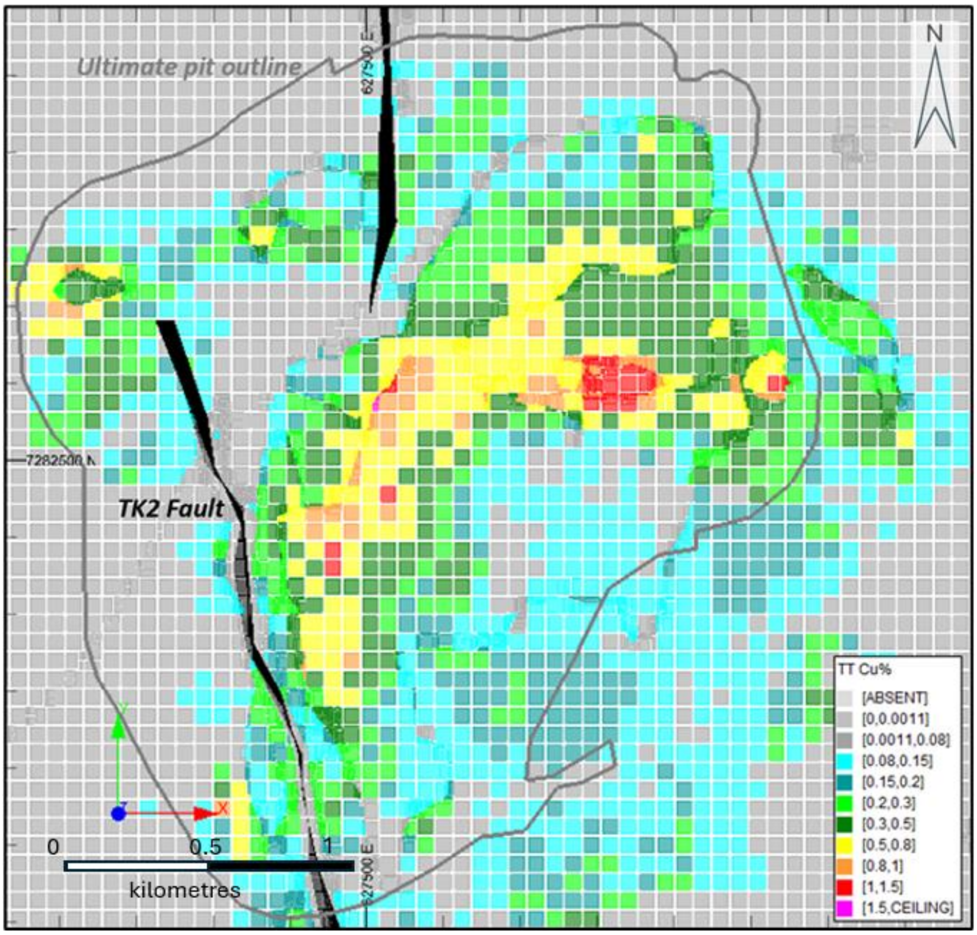
In the potassic zone, minor chalcopyrite and trace bornite are associated with secondary biotite. Most copper mineralisation is hosted in the phyllic zone and is related to the two phyllic alteration stages. Early green-sericite alteration is associated with a chalcopyrite–bornite assemblage, higher copper grades and above-average gold grades. Late white-sericite and quartz alteration is associated with pyrite–bornite and pyrite–chalcocite–covellite assemblages.

Molybdenite occurs as disseminations and in quartz vein stockworks, more commonly in K-feldspar altered granite and aplite dykes.

Fine-grained black chalcocite and lesser covellite are the principal secondary copper sulphides associated with supergene alteration. A discontinuous, sub-horizontal supergene enrichment zone occurs directly beneath the leached cap and has a higher copper tenor than the underlying hypogene mineralisation. This enrichment blanket is best developed in the northeastern part of the deposit. Supergene alteration associated with steeply dipping structures locally extends to depth, generating supergene pockets within hypogene zones. Copper mineralisation is observed as domains where either hypogene or supergene mineralisation is dominant, but overall, the deposit is characterised by mixed mineralogy with a variable chalcopyrite-to-chalcocite ratio.

The mineralisation has an arcuate geometry, reflecting the morphology of the igneous intrusion and the distribution of mineralising fluids (Figure 7-9).

Figure 7-9 Plan view slice of the block model copper grade estimates, showing an arcuate shape to the copper mineralisation



Mineralisation and metallurgy

During early mining, feed to the plant from the upper part of the supergene zone will be mixed with surrounding leached material. In some areas, the upper 0–80 m of the supergene horizon contains discrete metre-scale leached lenses. This mixing is expected to introduce atypical levels of iron oxide minerals into early plant feed, which may require management during plant commissioning, although the volumes involved are small.

Early feed will also include variable amounts of soluble copper associated with localised oxidation in supergene-dominant zones. Depending on oxidation intensity and acid consumption by gangue minerals, this may require appropriate mine planning and processing strategies. Further discussion of metallurgical performance of mixed mineralisation is provided in Item 13.

Supergene-enriched pockets within hypogene domains are common and observed at depth. In such areas, primary sulphide-dominant material contains subordinate but variable amounts of secondary sulphides (chalcocite).

Studies on coarse-crushed exposed material indicate no significant increase in soluble copper content over nine months under site conditions. However, drill core stored under cover at the Salta core shed shows frequent surficial oxidation with copper staining rimming chalcocite (Figure 7-8). This is attributed to hotter and more humid conditions at the core shed and extended exposure (>12 months). Long-term stockpiling or ponding of pit water over broken mineralised material should be avoided.

Trace nickel mineralisation has been reported in drill core. Nickel assay data indicate that grades will be insignificant at mining scale.

7.2.5 Structural geology

North-northeast and northwest trending, steeply dipping discrete mylonite zones are widespread within the Ordovician granite host. These pre-existing zones of structural weakness have controlled emplacement of Oligocene rhyodacitic dykes, porphyry-related quartz veining, fractures and small-scale faults.

Mapping, drill core observations and geophysical data indicate that the deposit is structurally influenced by two main sets of steep faults: a dominant northwest–southeast trending set and a subordinate north–south set. These structures act as conduits for supergene alteration and are commonly associated with secondary quartz, gypsum, alunite and chalcocite.

The TK2 (West) fault

The TK2 (West) Fault (Figure 7-9) is a deposit-scale, north–south trending structure along the western edge of the design pit, dipping steeply to the west. It offsets the base of the leached cap and underlying copper mineralisation, with the leached–mineralised contact significantly deeper on the eastern side. Where exposed at surface, the fault is expressed as breccia zones up to 5 m wide. Centimetre-thick, chalcocite-rich veins occur proximal to the fault.

ITEM 8 DEPOSIT TYPE

8.1 Mineral deposit type

The Taca Taca deposit is a porphyry copper–gold–molybdenum system hosted principally within granitic plutonic rocks, with subordinate dacite, dolerite and rhyolite intrusions. Hydrothermal alteration forms kilometre-scale zones that grade outward from a central potassic core through phyllic and argillic assemblages. The propylitic zone is comparatively restricted for a deposit of this size. Phyllic alteration is the most pervasive and hosts most of the mineralisation. Late-stage argillic supergene processes have locally upgraded copper tenor.

Mineralisation comprises an upper leached horizon overlying a mixed supergene–hypogene zone. Copper occurs disseminated and within fractures, veinlets and quartz stockworks. Sulphide zonation consists of a chalcopyrite–bornite–molybdenite core yielding outward to a strong pyrite-rich halo. The overall sulphide assemblage is mixed and more variable than typically observed in porphyry systems.

Surface weathering and oxidation removed copper from oxide and hypogene copper minerals, producing a 150–300 m thick copper-depleted leached cap. The leached copper was remobilised and partially re-precipitated as secondary sulphides beneath the leached horizon, forming discontinuous supergene enriched zones dominated by fine-grained black chalcocite with lesser covellite. The boundary between hypogene-dominant and supergene-dominant mineralisation is highly irregular and reflects alteration to depth along structures and within the host rocks. Copper grades within the supergene zones are typically higher than in the hypogene mineralisation.

8.2 Guiding principles for exploration and modelling

The geometry and extent of the granite host are well constrained through drilling, geological mapping, aerial photography and geophysical datasets. Early drilling followed outcrop control, with later drilling designed to define mineralisation limits. Drillhole orientations were planned to maximise intersection angles relative to the dominant mineralisation trend.

Detailed logging of drill core, together with analysis of multi-element geochemistry, supported the definition and modelling of deposit-scale geological domains. These domains were based on combinations of lithology, alteration, weathering and mineralisation characteristics. Geological and mineralisation domains were constructed by interpreting sectional strings and linking them into three-dimensional wireframes.

Copper mineralisation is associated with Oligocene rhyodacitic porphyry intrusions. The spatial distribution of copper sulphides is controlled by quartz–sulphide stockworks related to these dykes, as well as by intrusive contacts. Hydrothermal and intrusive breccias, intersecting fracture sets and veining proximal to faults commonly correspond with elevated metal grades. Geological logging, copper grades and multi-element geochemistry were collectively used to delineate mineralised domains.

The leached cap contains narrow discontinuous copper-oxide lenses and discrete perched gold-rich horizons toward the east. The base of the leached zone was defined by the absence of copper grades, increased hematite development and logged weathering observations.

Secondary sulphides formed by supergene enrichment occur within a discontinuous blanket directly underlying the leached cap and, to variable extents, are intermixed with hypogene mineralisation at depth where structures and host rocks have focused fluid flow. These supergene zones show a distinct increase in copper grades.

Modelling of mineralisation within the geological domains was guided by sequential copper assay data, which assisted in identifying the dominant copper species present within each altered rock type. The position of the TK2 Fault was defined using combined information from outcrop mapping, geophysical surveys, drill-core logging, core photographs, and supporting assay data.

Mineralisation remains open at depth and locally along the deposit margins to the south and east under Salar de Arizaro.

ITEM 9 EXPLORATION

9.1 Historical exploration

Fabricaciones Militares first noted copper mineralisation at Taca Taca in the late 1960s which was followed by multiple historical drill programs and geophysical surveys. A summary of these drilling campaigns is provided in Table 6-1. Historical geophysical surveys included:

- Transient electromagnetic (TEM) and Induced Polarisation (IP) surveys completed by BHP Minerals in 1997, focussed on delineating extents of sulphide mineralisation. The survey totalled 36.8 km (10 lines at 500 m spacing).
- Ground magnetic and gravity surveys completed by CASA in 1999, which targeted copper on the northern edge of the Salar de Arizaro.
- A 38.5 km radiometric survey was completed by Rio Tinto in 1999 for targeting shallow mineralisation. The survey overlapped the previous CASA ground magnetics.
- A Titan 24 survey (combined DCIP and magnetotelluric data) was conducted on behalf of Lumina during 2010. Results identified several targets having deeper sulphide mineralisation.

Surface outcrop mapping was active during most of the exploration phases with excavator trenching and road cuts. CASA and Rio Tinto completed a 100 m by 100m grid of geochemical sampling of soils and rock outcrops around the deposit peripheries.

The understanding of the Taca Taca deposit geology has been derived from drillhole logging, interpretation of assay data, geophysical surveys, and the mapping of outcrop and trenches.

9.2 Exploration by the Company

Following acquisition of the Project, the Company undertook several small-scale data collection programmes to complete and verify supporting datasets as summarised below:

- New-Sense Geophysics conducted a helicopter-borne magnetic and radiometric survey in 2014 acquiring 4,424.1 line-km of data at 300 m line spacing. These results supported anomaly definition, structural interpretation and identification of broader lithological trends.
- Additional geochemical sampling of in-situ soils was completed on a 500 m by 500 m grid around the outer limits of the concessions.
- In 2019, a detailed ground magnetic program was conducted over the Taca Taca deposit area. A total of 924-line km was surveyed at a 20m spacing for improving geological and structural interpretations across the deposit. The survey focussed on displacement within the enrichment blanket and depth to top of ore.
- A hyperspectral survey comprising 2 lines was flown directly over the deposit to better define key alteration minerals at surface in the deposit area, searching for previously untested zones of hydrothermal alteration.
- A high-resolution WorldView-3 topographic dataset was obtained, covering a 12 km by 23 km area over the deposit at 0.5 m resolution, with the surrounding region captured at 3 m resolution.
- In 2021, rock samples were taken to better support historic coverage focussing on the Camp Prospect (43 samples, 50 m by 50 m spacing), Pit South (182 samples, 50 m by 50 m spacing) and Little Taca (66 samples, 100 m by 100 m spacing) and was used to support targeting of brownfields drillholes.
- During 2022, a further series of geochemical rock samples were taken on a 250 m by 250 m grid over all outcrop in the TT license area to provide an update on method and consistent coverage.

- In 2025, a structural mapping campaign was conducted over the immediate deposit and pit design area with the objective of refining the existing fault model for geological, geotechnical, and hydrogeological purposes. Additionally, a soil sampling program was completed, collecting 332 soil samples at 25 m spacing along NE-trending lines designed to intersect the NW and NNE structural systems, along with 69 rock samples from surface outcrops. The results show a strong correlation between anomalous gold values and both the WNW and NNE fault sets.

ITEM 10 DRILLING

10.1 Drilling overview

Drilling across the Taca Taca deposit and its surrounds is mostly from other companies prior to the Company's acquisition of the Project. As such, the information detailed herein is in the context of it being relied upon for the current Mineral Resource estimate. The issuer has verified the drillhole core logging, sampling and assay results with visual drill core inspections and verification, database validations and check re-logging. Drillhole samples have been subject to detail geological logging, sampling and analysis with supporting QAQC. Drill samples have been stored in a safe and secure storage facility in Salta city for further testing a verification. All drill data has been stored within a SQL database system ensuring consistent and validated data.

An additional 23 diamond drill holes were added since the previous March 2021 Technical Report. At the time of this report there were 507 drillholes (Table 10-1) with a total drilled length of 180,133 m available across the Taca Taca deposit and its surrounds (Figure 10-1).

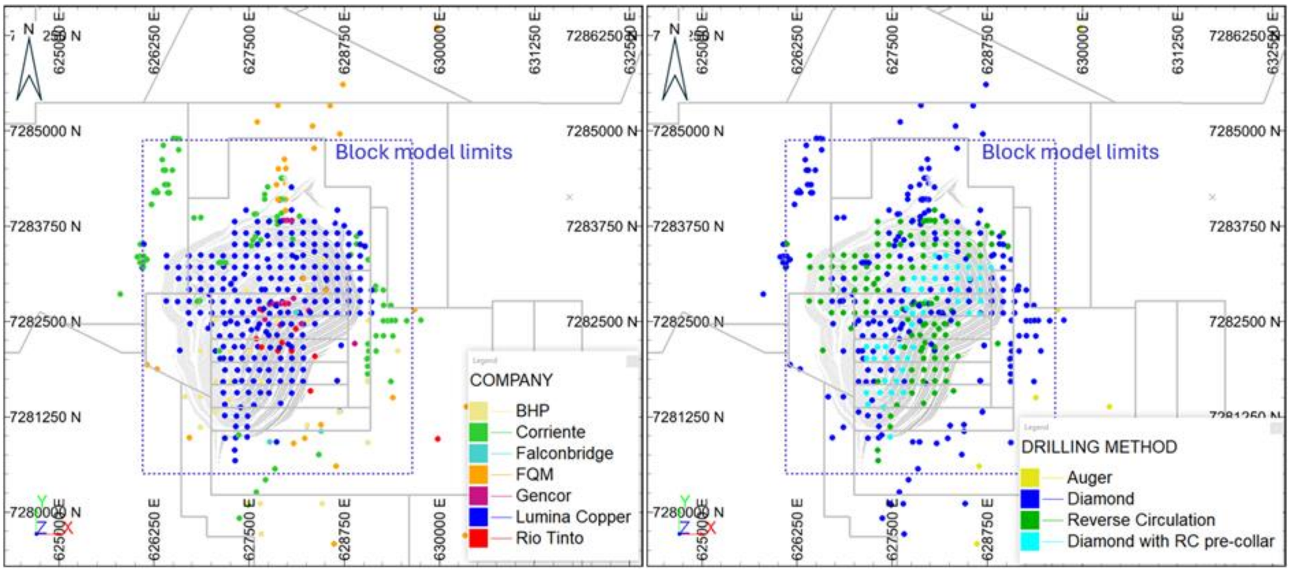
Table 10-1 Current Taca Taca drill hole details per company

Year	Company	Drilling method	Number of holes	Metres drilled
1970	Falconbridge	DD	3	529
1994	Gencor	RC	18	1,606
1996	BHP	DD	4	1,651
1997	BHP	DD	31	9,974
1998	Corriente	DD	14	3,328
1999	Corriente	DD	80	4,428
1999	Rio Tinto	RC	9	3,338
2008	Rio Tinto	DD	8	4,877
2010	Lumina Copper	DD	5	3,437
2011	Lumina Copper	DD	39	22,608
2011	Lumina Copper	RC	17	3,696
2011	Lumina Copper	RD	34	24,311
2012	Lumina Copper	DD	60	36,898
2012	Lumina Copper	RC	111	35,112
2012	Lumina Copper	RD	17	11,609
2018	FQM	DD	10	2,283
2019	FQM	AG	16	160
2019	FQM	DD	8	2,213
2022	FQM	DD	23	8,102
SUB TOTAL		**RC**	**155**	**43,752**
		DD	**336**	**136,249**
		Augur	**16**	**160**
TOTAL		**Holes**	**507**	**180,160**

Of the 507 drillholes, 461 are drilled within Company held concessions (Figure 10-1). Most holes were diamond drilled holes with approximately 24% drilled using Reverse Circulation (RC). Of the 461 holes, 399 are located within the block model volume (Figure 10-1 and Figure 10-2).

Drilled holes were completed for multiple purposes including metallurgy (4), geotechnical (32), water testing (35), exploration (209) and infill (227) drilling.

Figure 10-1 Drillhole collar locations by company and drilling method, relative to the life of mine pit and concession boundaries



10.1.1 Added drilling data

Since the previous March 2021 Technical report, the Company completed a diamond drilling program of 23 holes to delineate limits of known mineralisation and to sterilise areas proposed surface infrastructure areas (Figure 10-2).

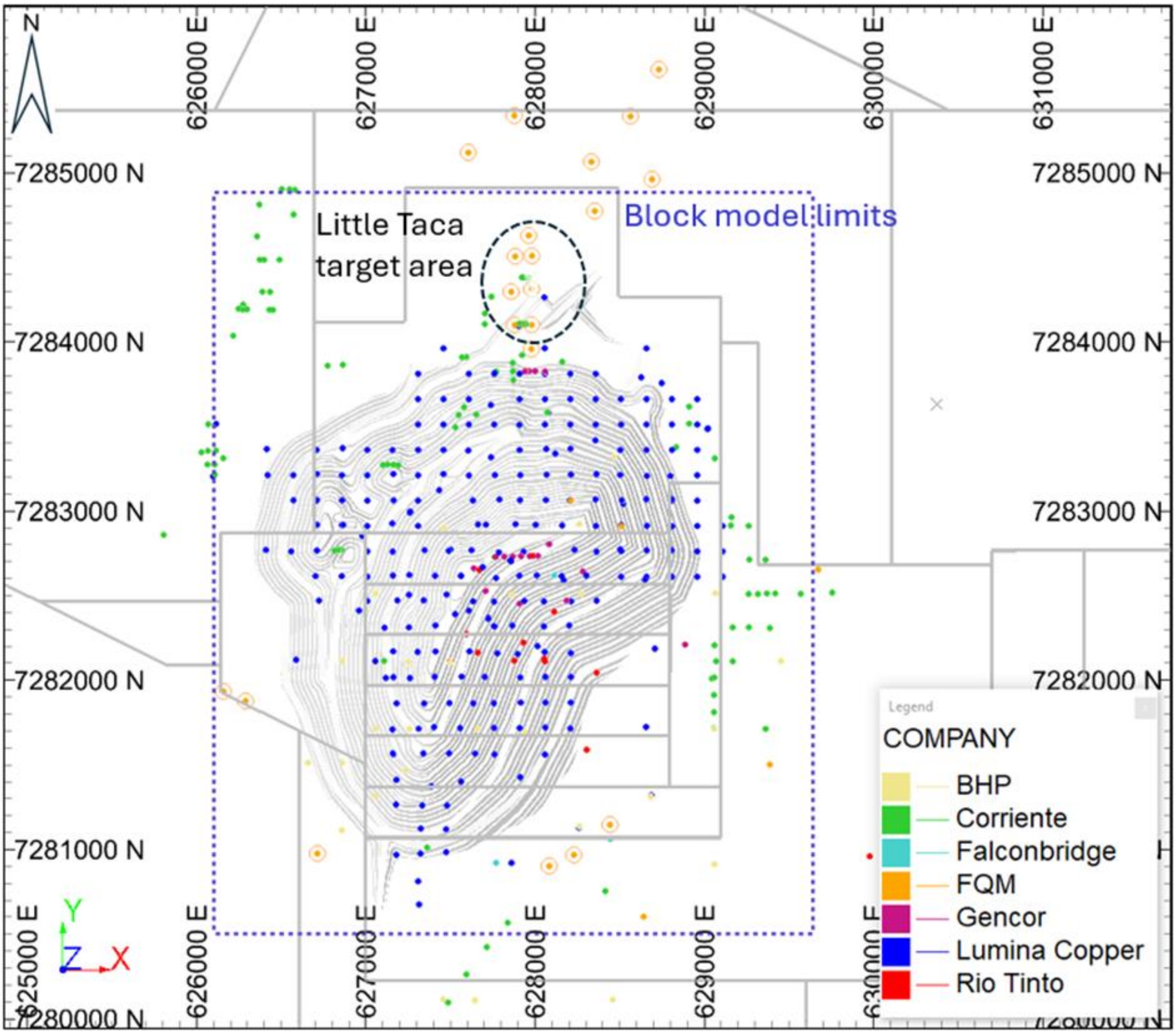
Of the 23 drilled holes, two are located outside the project area and seven are located external to the block model estimate volume. These nine holes are not relevant to this estimate and are in the northern part of the project area and within planned surface infrastructure areas. Due to the absence of visual mineralisation in these holes, only one was assayed but not used in this estimate.

In total, 14 of the 23 drill holes completed in 2022 were assayed. 13 of these holes, comprising 2,590 assays, were incorporated into the drill hole database and used in this estimate.

Five of these holes drilled the Little Taca target (Figure 10-2) but did not identify any continuous copper or gold mineralisation. Isolated narrow copper and gold vein intercepts were encountered at depth; however, these were considered uneconomic due to their limited continuity and depth. As such, the Little Taca exploration target is deemed sterilised.

All assayed drill holes from the 2022 campaign were analysed by ICP-MS for a 48-element suite, including copper and molybdenum, and by fire assay with AAS finish for gold (Figure 10-2).

Figure 10-2 Plan view of the 21 drill holes added to the block model area relative to the life of mine pit shell. The little Taca exploration target area is highlighted in black



10.2 Drilling data used in the Mineral Resource estimate

For Mineral Resource estimation, the drilling database was spatially clipped to include drill holes located within the immediate vicinity of the deposit (Figure 10-1 and Figure 10-2). The clipping boundary coordinates are provided in Table 10-2. All drilling data and related statistics reported herein is based on this clipped dataset.

Table 10-2 Minimum and maximum X and Y coordinates for drill data selection relevant to this estimate

	X coordinate	Y coordinate
Minimum	626,100	7,280,500
Maximum	629,640	7,284,880

Accordingly, 399 drill holes contain assay data relevant to this Mineral Resource Estimate, comprising a total of 164,822 m of samples (Table 10-4). The clipped dataset includes 13 drill holes from the 2022 drilling campaign and 13 historical Corrientes drill holes that were not included in the previous March 2021 NI 43-101 Technical Report due to missing assays now located and included (Table 10-3). The added holes, detailed in Table 10-3 below, are all diamond drilled holes except for a single hole drilled by Lumina Copper using Reverse circulation.

Table 10-3 Summary of drill holes added since the previous March 2021 Technical report

	Drill identity	Easting	Northing	Elevation	Depth	Drilling Purpose	Comment
1	TTBJ22-141	626,290	7,281,877	3,579	600	Mineralisation Delineation – SW LOM Pit Shell	Assayed
2	TTBJ22-142	626,159	7,281,933	3,560	605	Mineralisation Delineation – SW LOM Pit Shell	Assayed
3	TTBJ22-144	628,443	7,281,146	3,505	616	Mineralisation Delineation – SE LOM Pit Shell	Assayed
4	TTBJ22-155	628,084	7,280,902	3,496	700	Mineralisation Delineation – SE LOM Pit Shell	Assayed
5	TTBJ22-162	628,230	7,280,968	3,491	700	Mineralisation Delineation – SE LOM Pit Shell	Assayed
6	TTBJ22-145	627,978	7,283,957	3,197	300	Condemnation Drilling – Little Taca Target	Assayed
7	TTBJ22-146	627,880	7,284,099	3,541	300	Condemnation Drilling – Little Taca Target	Assayed
8	TTBJ22-147	627,980	7,284,100	3,197	300	Condemnation Drilling – Little Taca Target	Assayed
9	TTBJ22-148	627,859	7,284,295	3,573	300	Condemnation Drilling – Little Taca Target	Assayed
10	TTBJ22-149	627,980	7,284,312	3,536	305	Condemnation Drilling – Little Taca Target	Assayed
11	TTBJ22-150	627,883	7,284,504	3,459	375	Condemnation Drilling – Little Taca Target	Assayed
12	TTBJ22-151	627,982	7,284,508	3,541	300	Condemnation Drilling – Little Taca Target	Assayed
13	TTBJ22-152	627,963	7,284,628	3,547	300	Condemnation Drilling – Little Taca Target	Assayed
14	TK-35	627,869	7,283,825	3,540	198	CRR - 1998	Assayed
15	TK-36	627,771	7,283,824	3,541	250	CRR - 1998	Assayed
16	TK-37	627,872	7,283,774	3,542	213	CRR - 1998	Assayed
17	TK-38	627,868	7,283,875	3,539	209	CRR - 1998	Assayed
18	TK-39	627,549	7,283,568	3,561	296	CRR - 1998	Assayed
19	TK-40	627,529	7,283,494	3,565	262	CRR - 1998	Assayed
20	TK-41	627,653	7,283,570	3,552	228	CRR - 1998	Assayed
21	TK-42	628,073	7,283,583	3,551	233	CRR - 1998	Assayed
22	TK-43	628,160	7,283,882	3,533	195	CRR - 1998	Assayed
23	TK-45	627,923	7,283,921	3,538	150	CRR - 1998	Assayed
24	TK-46	627,581	7,283,614	3,558	257	CRR - 1998	Assayed
25	TK-47	628,834	7,283,377	3,471	229	CRR - 1998	Assayed
26	TTRC12-47	626,859	7,283,211	3,614	401	LCC-2012	Assayed

Drill holes completed prior to 2008 generally lack complete procedural documentation and have limited or no associated QA/QC data. These drill holes were assessed through comparison with adjacent drilling, inspection of remaining drill core where available, and review of historical reports. The assay data from these drill holes were found to be consistent with surrounding drilling and were therefore retained for Mineral Resource estimation. Many of these holes intersect shallow portions of the deposit, primarily within the leached cap or peripheral zones, and are not considered to materially influence the Mineral Resource Estimate.

Table 10-4 Details of drilling data used in this mineral resource estimate update

Year	Company	Number of holes	Hole purpose	Hole type*	Hole ID sequence	Meters drilled
1996	BHP	4	Exploration	RD	TK01 - TK04	1,651
1997	BHP	26	Exploration	RD	TK05 - TK25,TK27 - TK33	8,548
1998	Corriente	14	Exploration	DD	TK34 - TK47	3,328
1999	Corriente	64	Exploration	DD	TK48 - TK126	3,665
1999	Rio Tinto	7	Exploration	RC	CCR001 - CCR007,ARI001	2,732
2008	Rio Tinto	8	Exploration	DD	TTBJ0001 - TTBJ0008	4,877
2010	Lumina	5	Exploration	DD	TTBJ10-01 - TTBJ10-5	3,436
2011	Lumina	64	Resource development	DD/RD	TTBJ10-06,TTBJ11-07 - TTBJ11-73	42,364
2011	Lumina	1	Exploration	RC	TTEX01	200
2011	Lumina	4	Exploration	DD	TTEX02,TTEX03,TTEX07	1,851
2011	Lumina	4	Geotechnical	DD	TTGT01 - TTGT-04	2,404
2011	Lumina	16	Resource development	RC	TTRC11-01 - TTRC-17,TTRC12-88/94/95	3,496
2012	Lumina	5	Exploration	RC	FR12-05 - FR12-10	1,878
2011	Lumina	1	Water monitoring	DD	TTB11-44	300
2012	Lumina	11	Geotechnical	DD	TTTV1-TTTV11	5,905
2012	Lumina	66	Resource development	DD/RD	TTBJ11-72 - TTBJ11-76,TTBJ12-77 - 136	42,602
2012	Lumina	81	Resource development	RC	TTRC12-18 - TTRC12-97	28,559
2012	Lumina	2	Water monitoring	RC	AV-SP4S/SP5D	260
2019	FQM	3	Metallurgical	DD	TTBJ19-138 - TTBJ19-140	1,065
2022	FQM	13	Exploration	DD	TTBJ22-141 - TTBJ22-152/155/162	5,701
TOTALS		**399**				**164,822**
* DD=diamond drilled holes, RC=reverse circulation holes, RD=diamond drilled hole with RC pre-collar						

10.3 Historical drilling

Prior to Lumina Copper, five different companies had carried out exploratory drill campaigns across the Taca Taca deposit and surrounds (Table 10-4).

Earliest drilling of the copper porphyry was by Falconbridge in 1975. Results from three diamond holes showed a relatively thick, metal depleted, leached cap. No further drilling was conducted until 1994 when Gencor tested for shallow gold-copper bearing veins to the north of the porphyry and remnant copper mineralisation within the porphyry leached cap (Figure 10-1).

Between 1996 and 1997, BHP drilled a total of 35 diamond holes (including two re-drills) at an approximate 400 m by 400 m grid spacing into the porphyry (Figure 10-1). Results partially delineated the supergene dominant zone of mineralisation directly below the leached cap.

During 1998 and 1999, CASA drilled 14 diamond holes and 80 RC holes focusing on shallow and exotic copper mineralisation peripheral to the porphyry (Figure 10-1). Rio Tinto conducted two separate campaigns in 1999 and 2008. Drilling in 1999 mainly targeted shallow oxides within the porphyry leached cap (seven RC holes) and exotic mineralisation below the Salar de Arizaro (two RC holes). In 2008, Rio Tinto returned to test for deeper hypogene copper-molybdenum mineralisation at the core of the porphyry with eight diamond holes (including two re-drills).

Despite most historical drillholes intersecting mineralisation, previous owners considered the intercepts to be narrow and discontinuous and that preliminary exploration models did not recognise a potential for the present-day mineralisation extents. Drilling results typically did not meet then corporate criteria to warrant further expenditure in the historic economic climate.

Assay data is not available for historical drillholes drilled by Falconbridge and Gencor and for 12 of the 14 diamond holes drilled by CASA. Nevertheless, drillholes completed by BHP, Rio Tinto and those CASA drillholes yielding assay data, were assessed and deemed to be sources of sufficient quality data to support the Mineral Resource estimate for this Technical Report.

10.4 Drilling database

Drilling data are stored in the Company's central, secure SQL database. Historical datasets were migrated into this database and subjected to validation checks to ensure data integrity and consistency. Collar coordinates were validated against a topographic surface, with additional field spot-checks completed by Company geologists. A representative subset of assay records was verified against original laboratory certificates to confirm data transcription accuracy. During Mineral Resource estimation, the database was further validated using built-in geological software validation tools, with no material data issues identified.

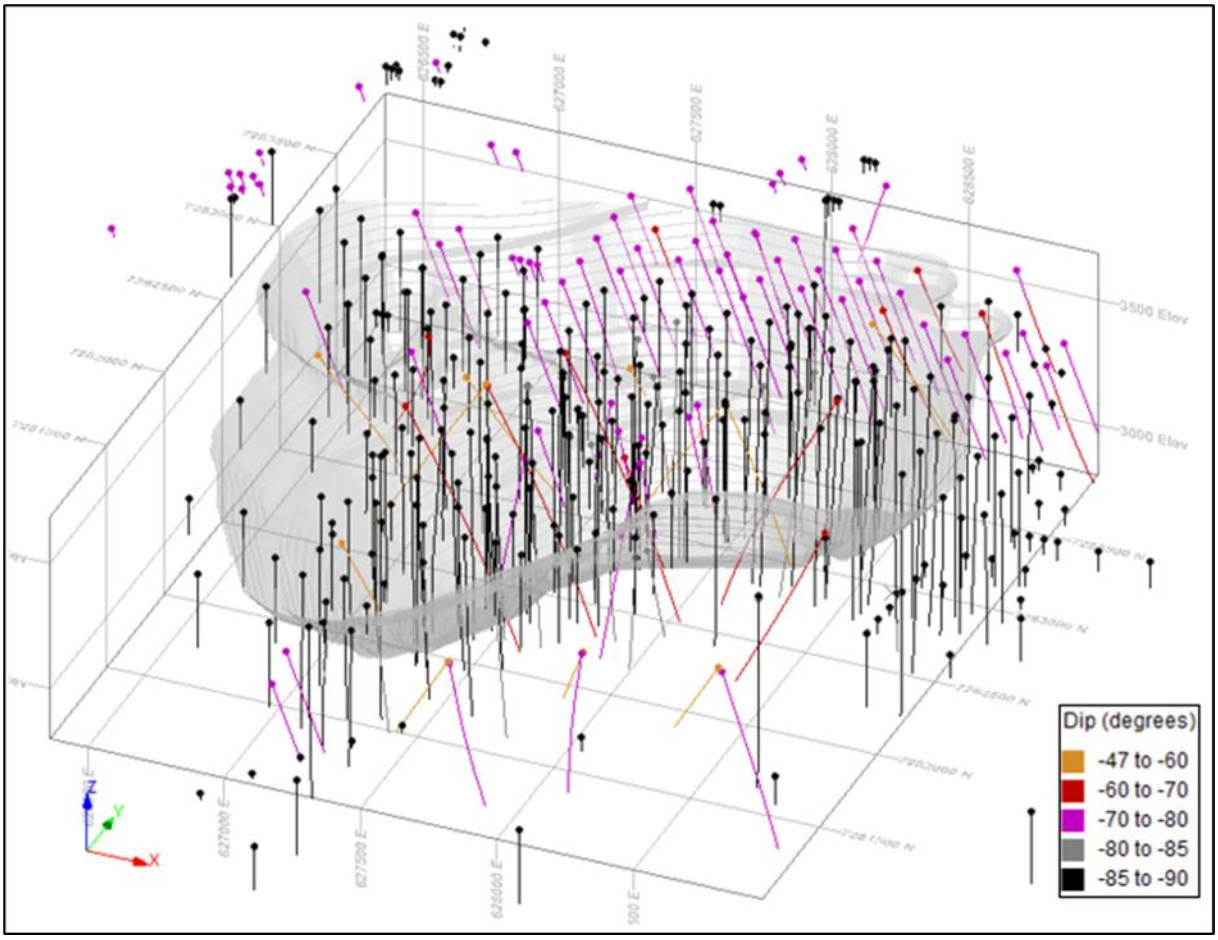
10.5 Drilling orientation

Approximately 70% of the drill holes used in the MRE were drilled vertically (Figure 10-3). Shorter RC drill holes, primarily located in the northern portion of the deposit, were typically inclined at approximately −70° to the east to optimise intersection angles with mineralisation associated with sub-vertical, north–south–trending structures. Geotechnical drill holes were completed at variable inclinations and azimuths, depending on the orientation of the target structures.

Diamond drill holes generally range from approximately 350 m to 900 m in depth, while shorter RC drill holes are predominantly located around the periphery of the deposit. The deepest drill hole on the property was completed by Rio Tinto in 2008 to a depth of 1,153 m.

Due to the largely amorphous shape of the Taca Taca deposit lithologies, alteration and mineralisation, sub vertical holes accurately represent the in-situ geology with close to perpendicular angles of intersection. Higher grade intervals are typical within the supergene zones immediately below the leached cap. As such, sub vertically drilled samples accurately capture the geological detail along these contacts. Sample intervals were mostly at 2 m, which provides good resolution of relevant geological contact detail.

Figure 10-3 Drillhole traces coloured by range of dip



10.6 Local grid

All drillhole collar coordinates were stored in WGS84 UTM Zone 19s grid and all modelling was completed in this grid. Data stored in different grids from previous Project owners was converted during its migration into the Company database.

Data provided by Lumina was collected in UTM coordinates based on the Gauss Kruger zone 2 POSGAR 1994 ellipsoid. The formula applied to convert these coordinates to WGS84 and to adjust ellipsoid elevations to orthometric elevations is summarised in Table 10-5.

Table 10-5 Formulae used to convert X, Y, Z coordinates from grid used by Lumina to WGS84 UTM Zone 19s

Coordinate	Formula
X mE	New X = Old X - 2000051.22
Y mN	New Y = Old Y - 878.33
Z mRL	New Z = Old Z - 40.9

10.7 Collar surveys

Holes drilled after 2010, had collar co-ordinates initially located using a handheld GPS and then surveyed using a differential Trimble GPS after drilling was complete. Most holes drilled prior to 2010 have since been located and resurveyed. All collar elevation coordinates were validated against a recently acquired high-resolution topographic surface with limited discrepancies identified. Twenty-three collar elevations required adjustment greater than 5 m to align with the detailed topography.

10.8 Downhole surveys

From 2008, downhole survey measurements were standard procedure on all diamond drilled holes, using single shot camera REFLEX or Peewee survey tools. Surveys were typically taken every 100 m to 150 m. No drill core orientation measurements were taken.

Of the 399 drill holes used in the Mineral Resource Estimate (MRE), 115 drill holes were completed prior to 2008 and do not have downhole survey measurements. These drill holes account for approximately 12% of the total drilled metreage used in the MRE.

Drilling completed after 2008 includes a subset of drill holes drilled at inclinations other than vertical (90°), predominantly at approximately −70°. This inclined drilling represents approximately 11% of the drilled metreage used in the MRE. Downhole survey measurements were not collected for this inclined drilling.

10.9 Core recovery and Rock Quality Data

Core recovery and Rock Quality Designation (RQD) measurements were recorded by trained technicians from all core drilled since 2010. Point load index tests were also conducted at 10 m intervals on all 2010 to 2012 drill core. This data was used to support sample quality analysis and geotechnical modelling. For holes drilled prior to this, visual cross checks on remaining core kept in storage suggests no significant core recovery issues were encountered.

For the 13 diamond drill holes completed in 2022 and located within the MRE area of influence, recovery and RQD measurements were collected for 11 drill holes, contributing a total of 1,747 recovery and RQD measurements to the database.

Approximately 94% of the recovered core has recovery greater than 85%, with intervals of lower recovery predominantly associated with the weathered leached cap. The overall median core recovery for these drill holes is 98%. These good recoveries provide quality samples with a limited risk to a non-representative sample mass.

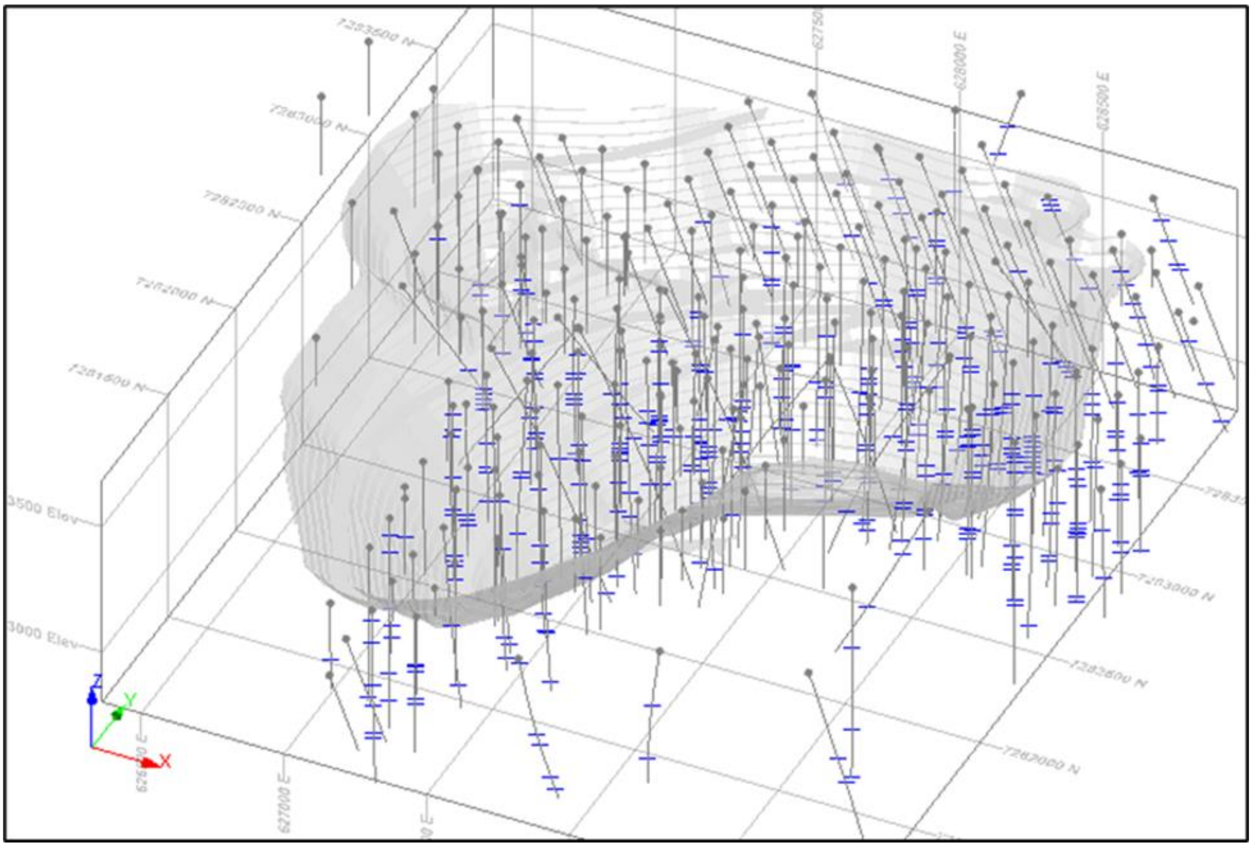
No core recovery or RQD issues are considered material to the three-dimensional position, accuracy, or quality of the geological logging and assay data.

10.10 Density measurements

Rock density was measured at 10 m intervals from all diamond core drilled between 2010 and 2012, using a core volume method and water displacement method. 45 cm or 60 cm whole core sample lengths were taken. A comparison of the two sets of results showed similar results. In line with industry practice, the water displacement results were estimated into the model.

A total of 5,363 density values were available with good spatial coverage of the deposit (Figure 10-4). There were sufficient samples per weathering and key lithology domains for a reasonable density estimate. No density measurements were collected during the 2022 drilling campaign.

Figure 10-4 Spatial distribution of samples, shown in blue, taken from Lumina drillholes for density measurement



10.11 Core/chip handling and storage

Diamond drilled core by BHP, Corriente, Rio Tinto, Lumina, and by the Company was securely transported from site to covered core storage facilities shortly after drilling. All core was stored in wooden core trays with hole information and depth demarcation clearly labelled. For all diamond holes used in the estimate, half or quarter core remains available in a safe and secure Company owned warehouse in Salta city, as a permanent record. Storage facilities are covered and secured.

In contrast, not all RC chip samples remain available from RC holes drilled prior to 2010. Most of these holes are shallow with limited to no impact on the estimate of mineralisation. All RC holes drilled after 2010 have chip logging sheets.

Full core photographs, both wet and dry, were captured by a trained technician prior to being sampled. Photographs are available for most holes drilled since 2010. Photographs have since been retaken on remaining core in storage for any holes missing original records.

10.12 Geological data

Drill core and RC chips were logged by qualified geologists for lithology, weathering, alteration, vein density and mineralisation. In addition, the Company also completed several drill core re-logging campaigns to validate historical logging and add to rock and mineralisation information. Regardless, historical logging was of good quality, reflecting the prevailing geology, and therefore relevant for use in this estimate.

Structural measurements were taken relative to core axis, using a protractor, for geological and geotechnical purposes. Frequently recorded features included fault contacts, breccia zones, fractures, igneous contact angles, and vein angles. Drill core was also logged for RQD, and samples were taken for density, point load testing, and element analysis.

Magnetic susceptibility measurements and Short-Wave Infrared (SWIR) spectral data was collected, at points approximately every 3 m and 10 m, respectively, for the length of each diamond hole drilled between 2010 and 2012. For drilling completed in 2019, high resolution SWIR spectral data was collected from pulp sample residuals after drill core sample preparation.

10.13 Factors materially affecting the Mineral Resource estimate

The QP considers that the diamond and RC drilling programs were conducted appropriately, with drillholes and resulting samples, suitably oriented and sized to capture geological and mineralisation detail. The drillhole grid spacing provides adequate coverage of the prevailing geology and mineralisation, with very limited instances of clustered drilling. In addition, drillhole orientations were sufficiently variable across the deposit to limit potential bias from a dominant preferred orientation. The drill hole depths have adequately captured mineralisation for a Mineral Resource estimate, despite mineralisation at depth still been open. Combined with secure and reliable data management practices, the drill data supports accurate three-dimensional positioning of downhole samples.

From these observations there are no identified drilling related factors that would materially impact on the accuracy or reliability of samples used in this Mineral Resource estimate. Regardless, the QP notes that targeted infill drilling will continue to support improved estimate accuracy and improved Mineral Resource classifications.

ITEM 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Drillhole samples used in this Mineral Resource estimate are dominated by approximately 75% competent, high recovery diamond drill core samples. Reverse circulation methods focussed within the leach cap zone with diamond drilling methods focussed on the supergene and hypogene zones. Most samples were analysed for copper and multiple elements using a 4-acid digest and an ICP_OES/MS finish. Gold was analysed for with traditional fire assay methods. A small proportion (approximately 13%) of copper mineralised samples were subject to sequential copper analysis to aid identification of copper mineralogy. QA/QC was practiced from 2008 onwards with results verified for these campaigns. All sample data has been managed in secure SQL drillhole database. Drill core is safely stored in a Company storage facility in Salta city.

11.1 Diamond drill core and RC chip logging

Geological logging of diamond drill core and RC chips was completed using standardised logging codes applicable at the time of drilling by the respective project owners. Historical logging data were primarily recorded in Microsoft Excel spreadsheets and subsequently imported into the Company's Maxwell DataShed SQL database. As part of this process, the data were formatted, standardised, and validated prior to use in Mineral Resource estimation.

11.2 Diamond drill (DD) core sampling

Documentation of core sampling procedures for drilling completed by BHP (1996–1997) and CASA (1998) is limited; however, database records and visual inspection indicate that sampling practices and resulting data are consistent with industry-standard procedures. Core sampling procedures implemented by Rio Tinto (2008), Lumina (2010–2012), and the Company (2019) are well documented and form the basis of current practice.

Following geological logging, diamond drill core was typically marked into nominal 1 m to 2 m sample intervals. Core was cut longitudinally along the marked centreline using a diamond saw, with one half retained in the core box for permanent reference and the other half bagged for laboratory analysis. In selected holes, samples were also collected from the core cutting table sludge to monitor potential metal loss to fines.

As standard practice, the full length of each drill hole was sampled. In some earlier drill holes, the upper portions of the hole were not sampled in zones of intense weathering and oxidation. All samples were assigned unique sample identifiers. From 2008 onwards, laboratory-generated sample tickets were used, with barcoded sample tags implemented from 2010. Samples were secured, placed into sealed bags and mesh sacks, and dispatched to the sample preparation laboratory upon completion of sampling for each drill hole. Remaining drill core was retained in covered, secure storage facilities in Salta.

11.3 Reverse Circulation (RC) sampling

Written documentation of reverse circulation (RC) sampling procedures for drilling completed by BHP (1996–1997 pre-collars), CASA (1999), and Rio Tinto (1999) is limited; however, available evidence indicates that sampling practices and resulting data are consistent with those applied in later RC drilling programs. The majority of these historical RC drill holes are shallow or located outside the planned pit extents and are therefore considered to have a limited impact on the Mineral Resource Estimate.

RC sampling procedures implemented by Lumina during the 2011–2012 drilling campaigns are well documented and form the basis of current practice. Drill cuttings were collected from the rig-mounted cyclone at nominal 2 m intervals over the full length of each drill hole. Samples were split using a standard adjustable riffle splitter to produce two representative sub-samples weighing approximately 6 kg to 10 kg each. Sampling equipment was cleaned between samples to minimise the potential for cross-contamination.

One sub-sample was dispatched to the laboratory for analysis, while the second was retained for reference or future sampling. In addition, a small (<100 g) portion of coarse chips was retained on a chip tray for geological logging. Where samples were recovered wet, a rotary wet splitter was used; material was allowed to decant prior to splitting, after which one split was submitted for analysis and the remainder retained in storage.

11.4 Sample preparation

BHP (1996–1997) and CASA (1998–1999) programmes

- Detailed records of sample preparation methods are no longer available.
- Available evidence indicates that industry-standard sample preparation procedures were applied and are consistent with those documented for subsequent drilling programmes.

Rio Tinto 1999 RC programme

- Sample preparation undertaken at the Bondar Clegg laboratory, Mendoza.
- Whole samples crushed to −80 mesh (177 µm), with a 1 kg split pulverised using a large pulp preparation procedure.

Rio Tinto 2008 DD programme

- Core samples prepared at the Alex Stewart laboratory, Mendoza.
- Samples weighed, dried, crushed to 80% passing 2 mm, riffle split to approximately 1.2 kg, and pulverised to 85% passing 75 µm.
- Approximately 200 g of pulp submitted for analysis.
- Samples transported under documented chain-of-custody procedures; no irregularities reported.

Lumina 2010–2012 programmes

- Sample preparation completed at ALS Minerals and Alex Stewart laboratories, Mendoza.
- Samples weighed and barcoded on receipt, crushed to 70% passing 2 mm, riffle split to approximately 1 kg, and pulverised to 85% passing 75 µm.
- Approximately 200 g of pulp submitted for analysis.
- Samples transported under sealed chain-of-custody protocols; no discrepancies reported.

FQM 2019 and 2022 programmes

- Core samples prepared at ALS Minerals Mendoza.
- Sample preparation procedures were consistent with those applied during the Lumina 2010–2012 programmes.

11.5 Sample analysis

A summary of drill campaigns and associated analytical laboratories used for sample analyses is provided in Table 11-1. Original assay certificates were not available for samples submitted prior to 2008; the previous owners' results were in spreadsheet format only.

Both Alex Stewart (Mendoza) and ALS (Lima) analytical laboratories are fully equipped and accredited with ISO/IEC 17025:2017 for mineral and geochemical testing competence as well as being ISO 9001 accredited for quality management covering chemical analysis of geological samples. Both Alex Stewart (Mendoza) and ALS (Lima) laboratories are independent of the Company.

Table 11-1 Analytical laboratory and methods used per drill campaign

Drill Campaign	Analytical Laboratory	Analytical Methods	Elements Analysed
1996–1997 BHP	Bondar Clegg, La Serena; American, Mendoza; SGS, Salta and Santiago	Unknown	Ag, Au, As, Cu, Mo, Pb, Zn
1998–1999 Corriente	ALS, Mendoza	AAS; Fire assay (AAS) for Au	Ag, Au, As, Cu, Mo, Pb, Zn
1999 Rio Tinto	Bondar Clegg, Vancouver	4-acid digest with ICP-OES/MS finish; Fire assay (AAS) for Au	34 elements + Au Ag, Al, As, Ba, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, Pb, Sb, Sc, Sn, Sr, Ta, Te, Ti, V, W, Y, Zn, Zr
2008 Rio Tinto	Alex Stewart, Mendoza	4-acid digest with ICP-OES finish; Ore-grade Cu by AAS; Fire assay (AAS) for Au	39 elements + Au Ag, Al, As, Ba, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, Hg, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, S, Sb, Sc, Se, Sn, Sr, Ta, Te, Ti, Tl, V, W, Y, Zn, Zr
2010–2012 Lumina	ALS, Mendoza; Alex Stewart, Mendoza	4-acid digest with ICP-OES finish; Ore-grade Cu by AAS; Fire assay (AAS) for Au; Sequential Cu leach (subset)	35 elements + Au Ag, Al, As, Ba, Be, Bi, Ca, Cd, Co, Cr, Cu, Fe, Ga, K, La, Mg, Mn, Mo, Na, Ni, P, Pb, S, Sb, Sc, Se, Sn, Sr, Th, Ti, Tl, U, V, W, Zn
2019,2022 FQM	ALS, Lima	4-acid digest with ICP-OES/MS finish; Fire assay (AAS) for Au; Sequential Cu leach	48 elements + Au Ag, Al, As, Ba, Be, Bi, Ca, Cd, Ce, Co, Cr, Cs, Cu, Fe, Ga, Ge, Hf, In, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, Rb, Re, S, Sb, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Y, Zn, Zr

11.6 Quality Assurance and Quality Control (QA/QC) protocols

Quality Assurance and Quality Control (QA/QC) procedures have been verified for drilling completed from 2008 onwards, with results available for review, representing approximately 89% of the samples used in the current Mineral Resource Estimate. QA/QC information for drilling completed prior to 2008 was sourced from historical reports and information provided by previous operators. Although detailed QA/QC results are generally unavailable for these earlier programmes, the associated data largely relate to peripheral or shallow portions of the deposit and show good agreement with proximal, better-documented samples. These data are therefore considered suitable for use in the Mineral Resource Estimate.

BHP 1996 to 1997 programme

- Historic reports indicate that BHP submitted quarter-core duplicates at a frequency of approximately 1 in 20 samples and coarse reject duplicates for check assaying; however, detailed results are not available.
- In 2003, AMEC collected 11 check samples from archived NQ core at matching intervals and submitted them for independent re-assay, with reported good agreement to original results.
- Comparisons with nearby Lumina drill holes (<30 m) show comparable grade distributions, indicating that the data are suitable for use in the Mineral Resource Estimate.

CASA 1998 to 1999 programme

- Historic reports indicate routine submission of pulp duplicates for check assaying; however, detailed results are not available. Most drill holes are shallow or located outside planned pit extents and are of limited relevance to the Mineral Resource Estimate.
- Comparisons with nearby Lumina drill holes (<30 m) show comparable grades, indicating that sample results are of adequate quality to support the Mineral Resource Estimate.

Rio Tinto 1999 RC programme

- Information provided by Rio Tinto indicates that systematic QA/QC procedures were implemented, including insertion of one field duplicate for approximately every 12 samples, together with pulp duplicates, certified reference materials (CRMs), and blanks; detailed insertion rates and results are not available. Most drill holes are in shallow zones with limited relevance to the main mineralised domains.
- Historic reports indicate that the data are of acceptable quality and are considered suitable for use in the Mineral Resource Estimate.

Rio Tinto 2008 DD programme

- Information provided by Rio Tinto indicates that systematic QA/QC procedures were implemented, including certified reference materials (CRMs) for Cu and Au (3–4%), coarse and pulverised blanks (2–3%), field and pulp duplicates (2–3%), and a limited number of half-core field duplicates. Additionally, 133 pulp duplicates from drill hole TTBJ0003 were submitted to ALS Lima for check analysis using four-acid digestion with AAS and Sludge samples from the core cutting table were routinely analysed to monitor potential metal loss to fines.
- Only blank and duplicate results are available to the Company. Blank sample results indicate that contamination during sample preparation was generally well controlled (Figure 11-1). Duplicate samples show acceptable precision for copper, gold, and molybdenum, with increasing scatter at higher grades (Figure 11-2 to Figure 11-4). Field duplicates for gold show poor correlation, largely due to low gold grades and the nugget effect characteristic of gold mineralisation.
- Historic reports and comparisons with proximal samples indicate that the data are of acceptable quality and are considered suitable for use in the Mineral Resource Estimate.

Figure 11-1 Blank samples control chart for Rio Tinto 2008 diamond drill campaign, both coarse crush (top) and pulp material (bottom)

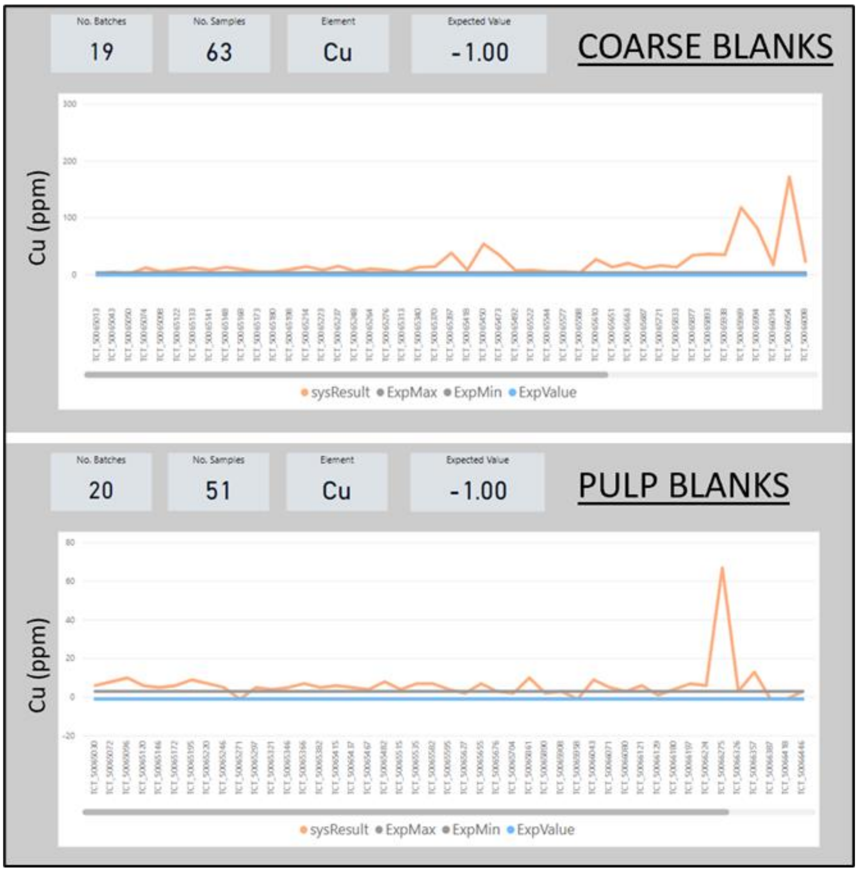


Figure 11-2 Scatter plots showing original copper analysis values against field, coarse, pulp, and laboratory duplicate results for Rio Tinto 2008 diamond drill campaign

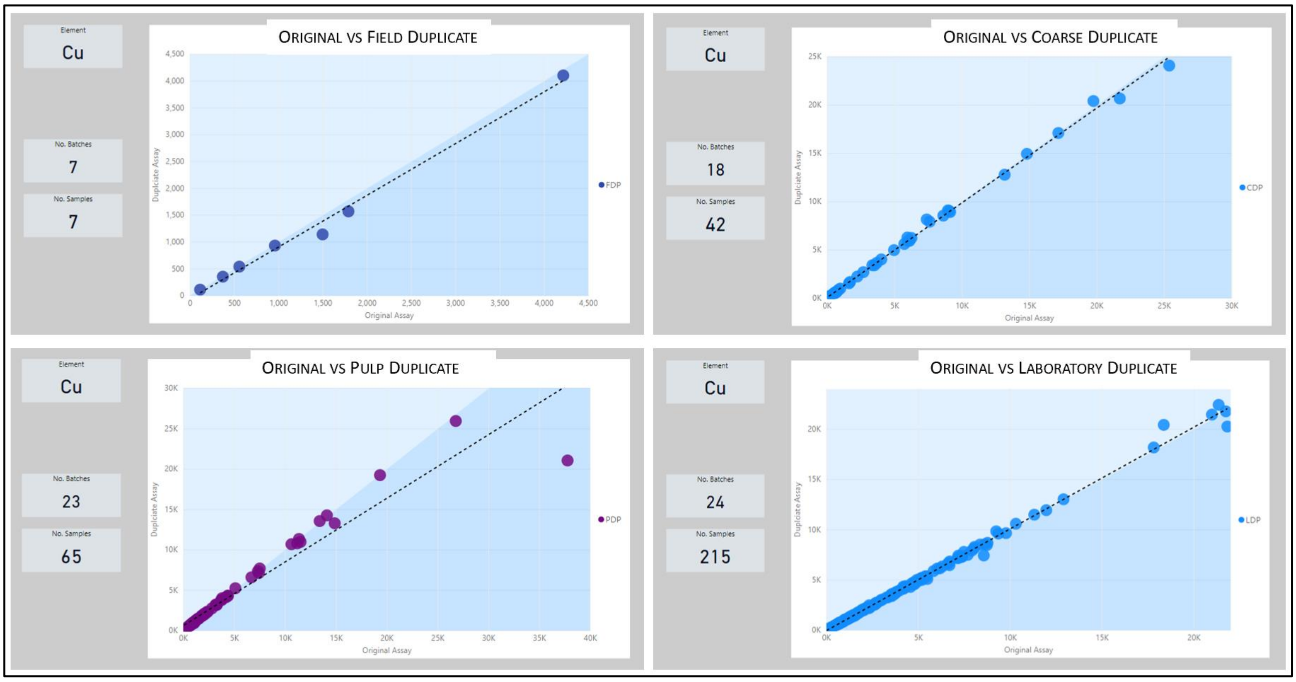


Figure 11-3 Scatter plots showing original gold analysis values against field, coarse, pulp, and laboratory duplicate results for Rio Tinto 2008 diamond drill campaign

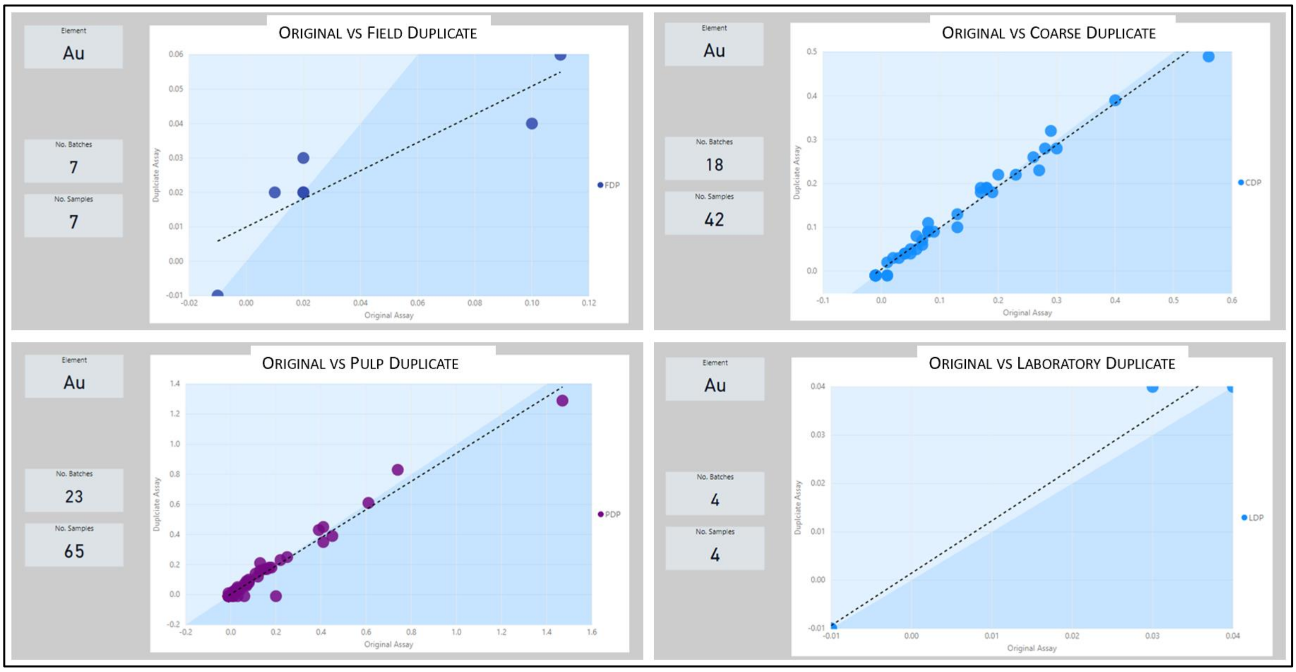
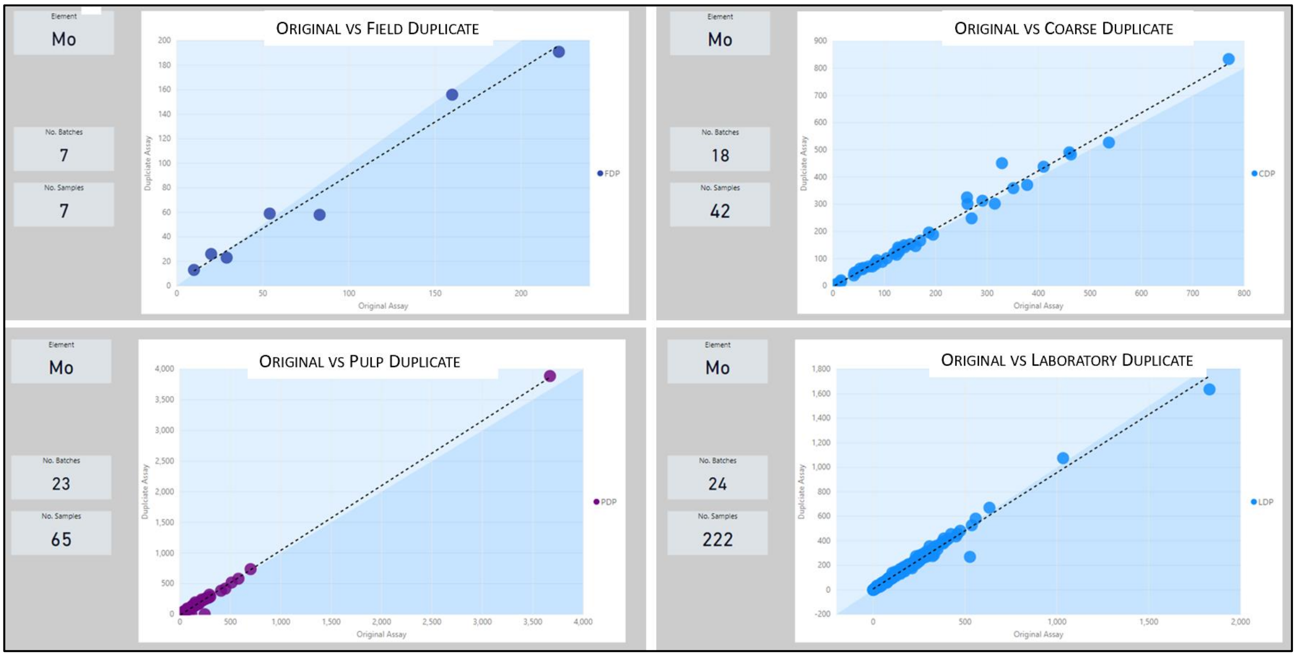


Figure 11-4 Scatter plots showing original molybdenum analysis values against field, coarse, pulp, and laboratory duplicate results for Rio Tinto 2008 diamond drill campaign



Lumina 2010 to 2012 programme

- Systematic, industry-standard QA/QC procedures were implemented throughout the Lumina drilling programme, including insertion of certified reference materials (CRMs) for Cu, Au, and Mo at an overall rate of approximately 2%, with three CRM types used to cover a range of grade levels. Coarse and pulverised blank materials and both coarse and pulp duplicate samples were each inserted at rates of approximately 2%. All QA/QC results are available to the Company.
- Returned coarse and pulverised blank results indicate that contamination was adequately controlled during sample preparation for both diamond core and RC samples (Figure 11-5; Figure 11-6).

- Duplicate sample results demonstrate acceptable analytical precision for copper, gold, and molybdenum for both diamond core and RC samples (Figure 11-7; Figure 11-8).
- CRM assay results indicate acceptable primary laboratory accuracy for both diamond drilling (Figure 11-9) and RC drilling programmes (Figure 11-10), with most results within certified limits and only minor evidence of sample mislabelling.

Figure 11-5 Blank samples control charts for Lumina diamond drill campaign, both coarse crush (left) and pulp material (right)

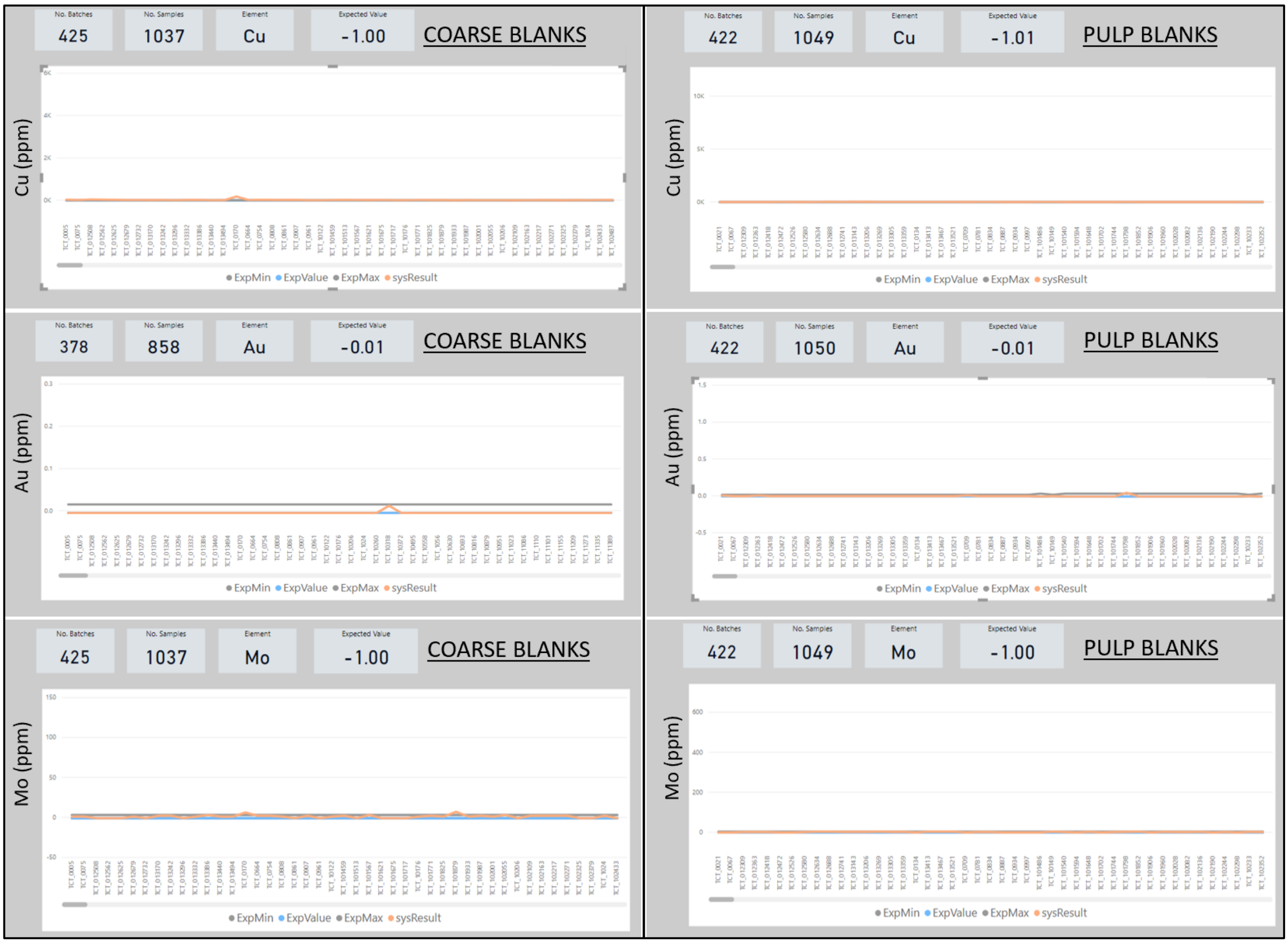


Figure 11-6 Blank samples control charts for Lumina RC drill campaign, both coarse crush (left) and pulp material (right)

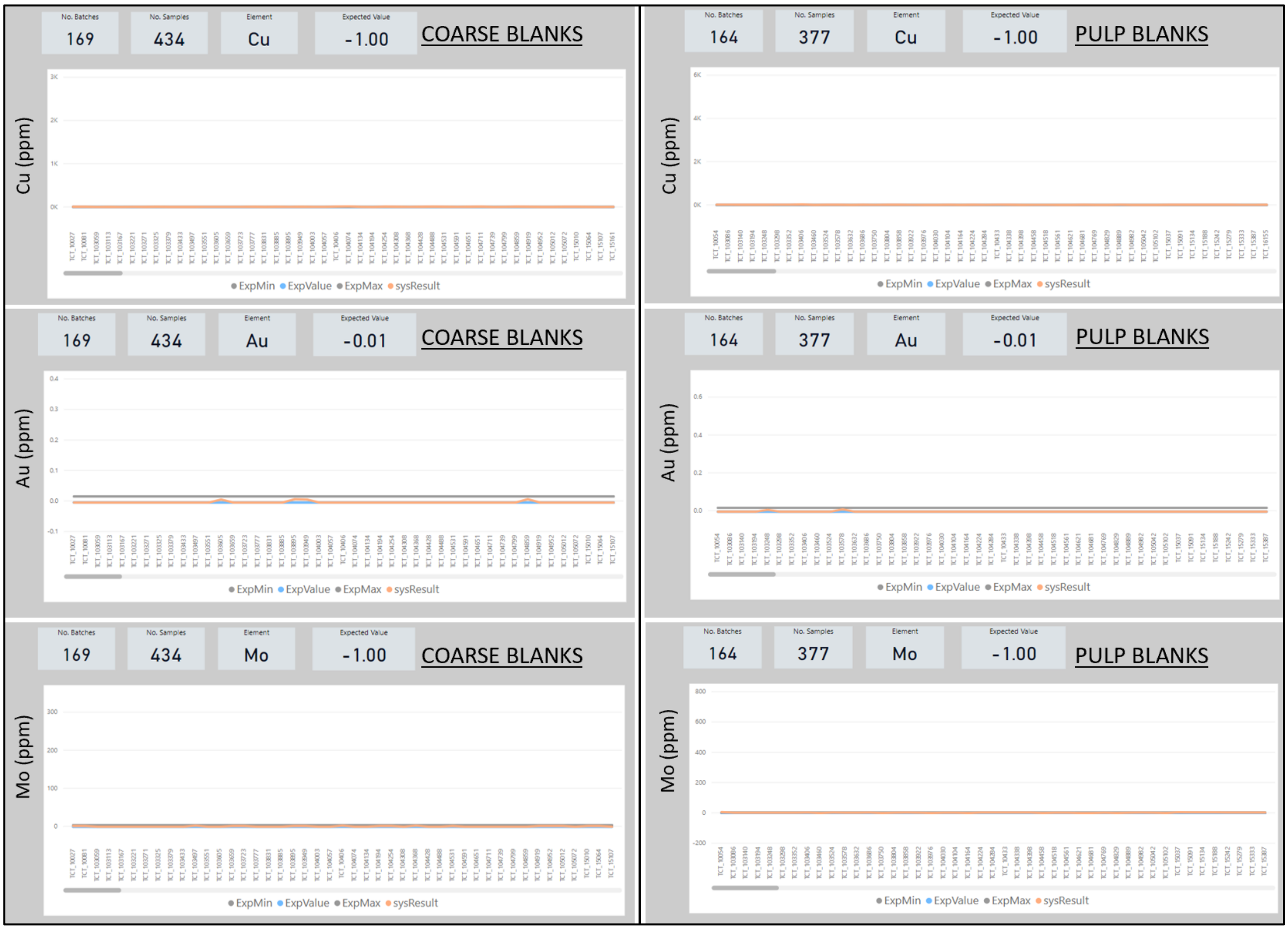


Figure 11-7 Scatter plots showing original Cu, Au, and Mo analyses values against coarse, pulp, and laboratory duplicate results for Lumina diamond drill campaign

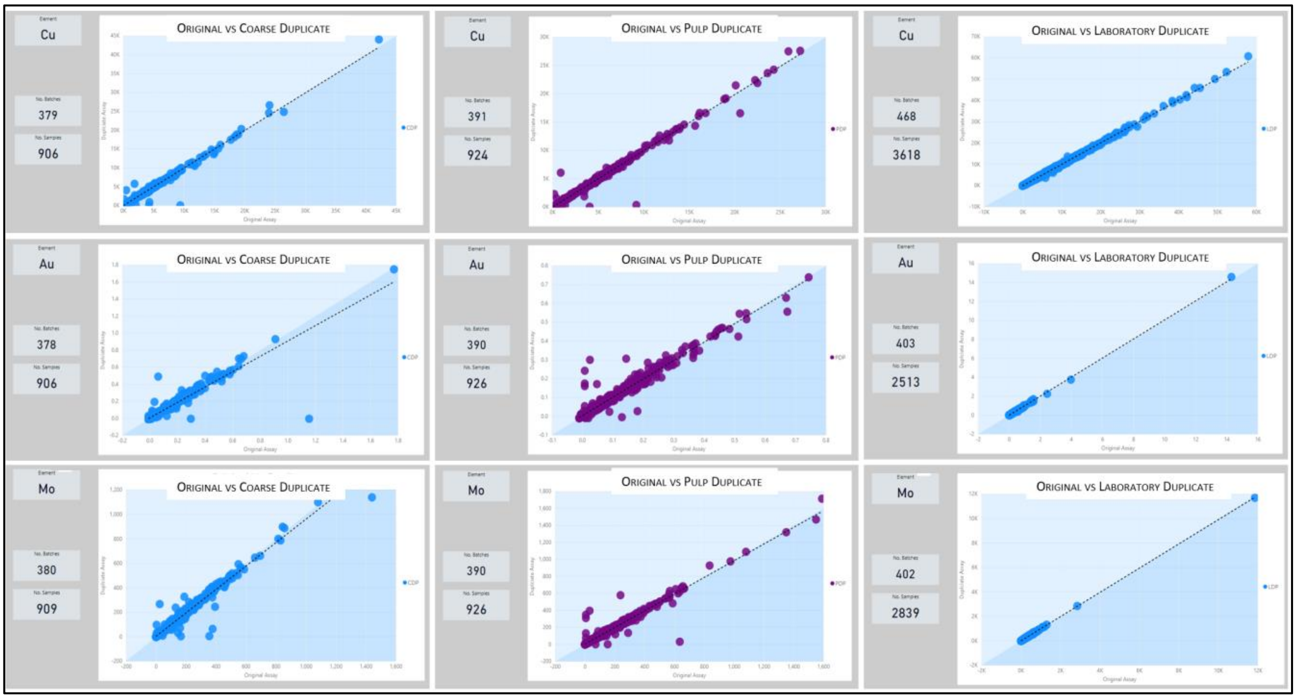


Figure 11-8 Scatter plots showing original Cu, Au, and Mo analyses values against coarse, pulp, and laboratory duplicate results for Lumina RC drill campaign

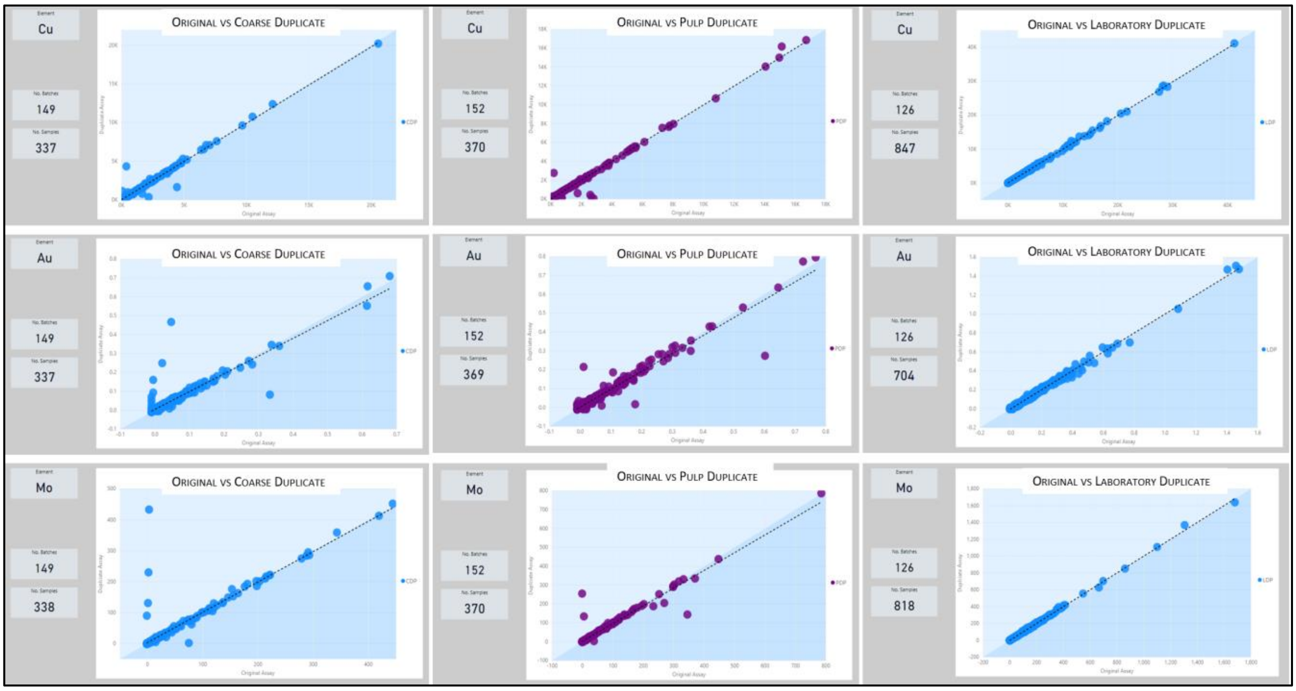


Figure 11-9 Control charts showing Cu, Au, and Mo results from the 3 CRM samples analysed during Lumina diamond drill campaign

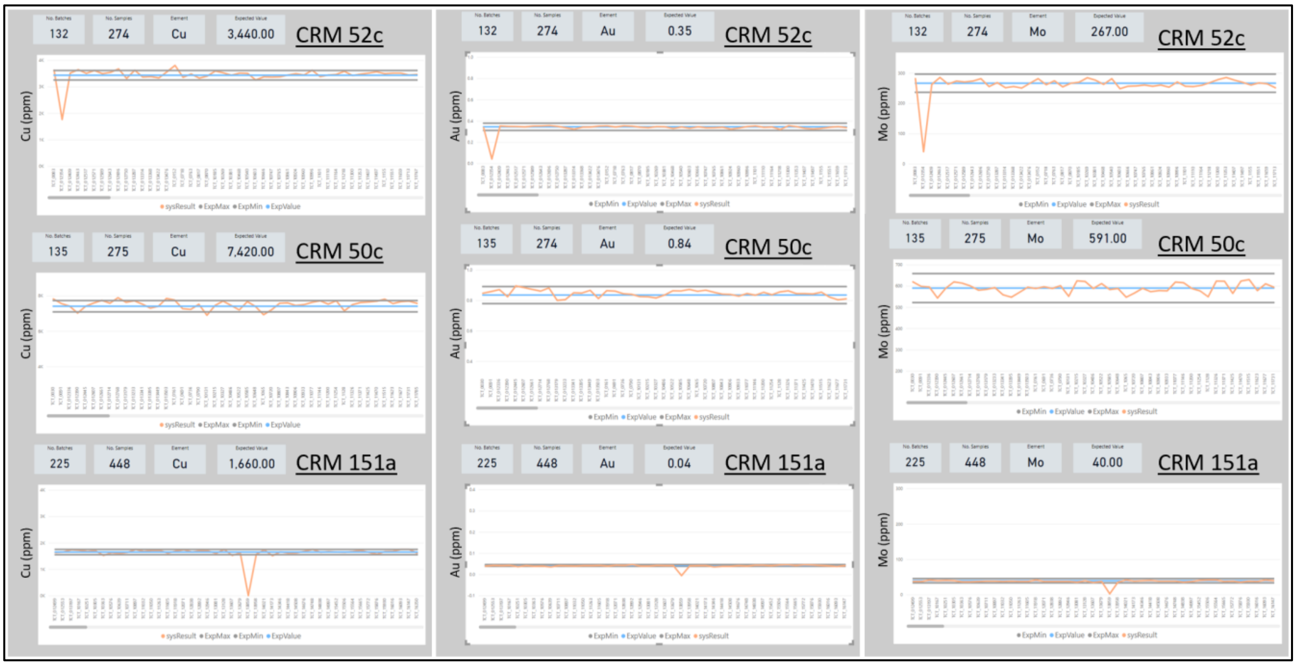
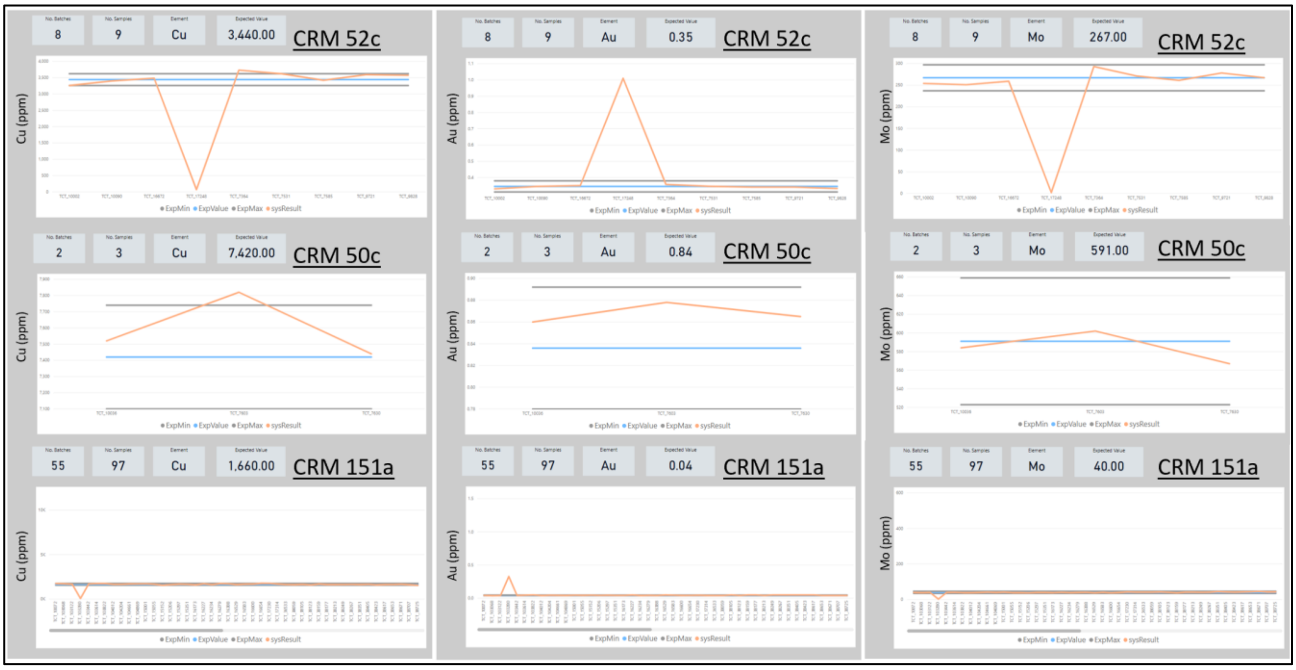


Figure 11-10 Control charts showing Cu, Au, and Mo results from the 3 CRM samples analysed during Lumina RC drill campaign



FQM 2019 programme

- QA/QC procedures included insertion of certified reference materials (CRMs) for Cu, Au, and Mo at a rate of approximately 3%, together with coarse blank material and coarse duplicate samples at rates of approximately 2%.
- Returned coarse blank results indicate that contamination during sample preparation was adequately controlled (Figure 11-11). Duplicate results demonstrate acceptable analytical precision for copper, gold, and molybdenum (Figure 11-12), and CRM results indicate acceptable primary laboratory accuracy, with all values within certified limits (Figure 11-13).

Figure 11-11 Blank samples control charts for FQM diamond drill campaign analysed for Cu, Au, and Mo

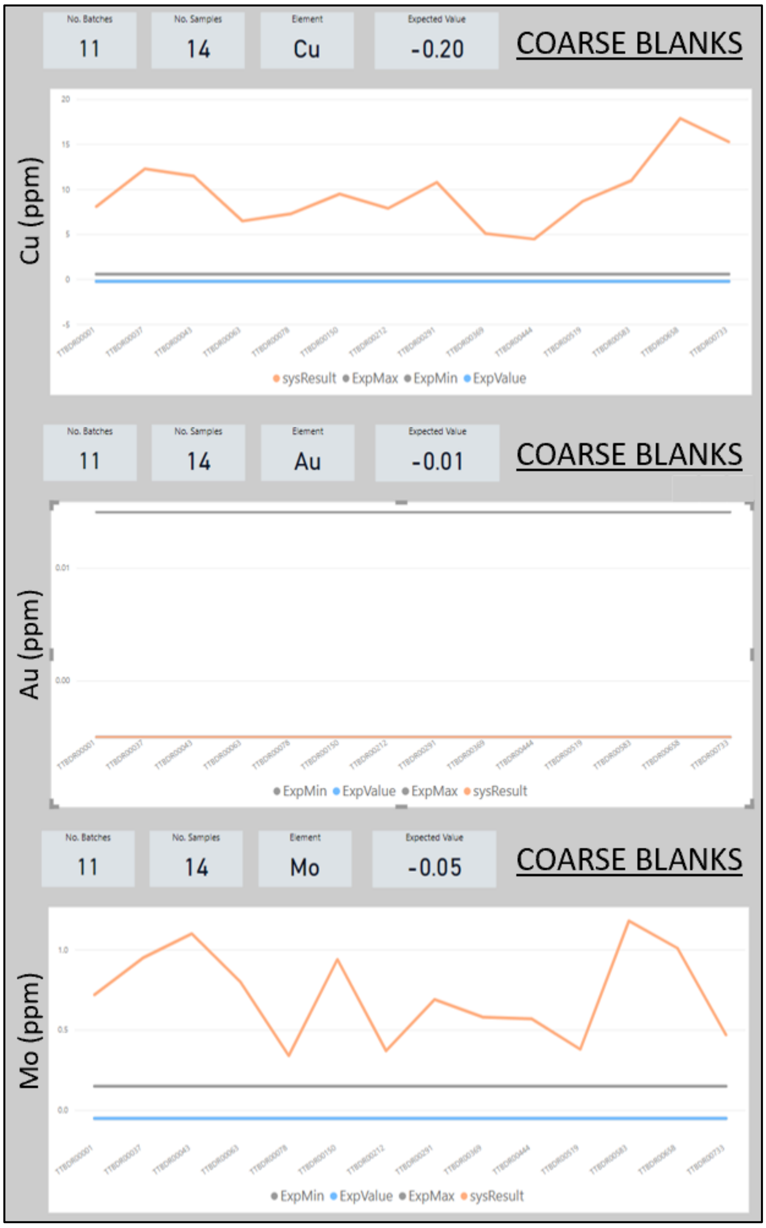


Figure 11-12 Scatter plots showing original Cu, Au, and Mo analyses values against coarse and laboratory duplicate results for FQM diamond drill campaign

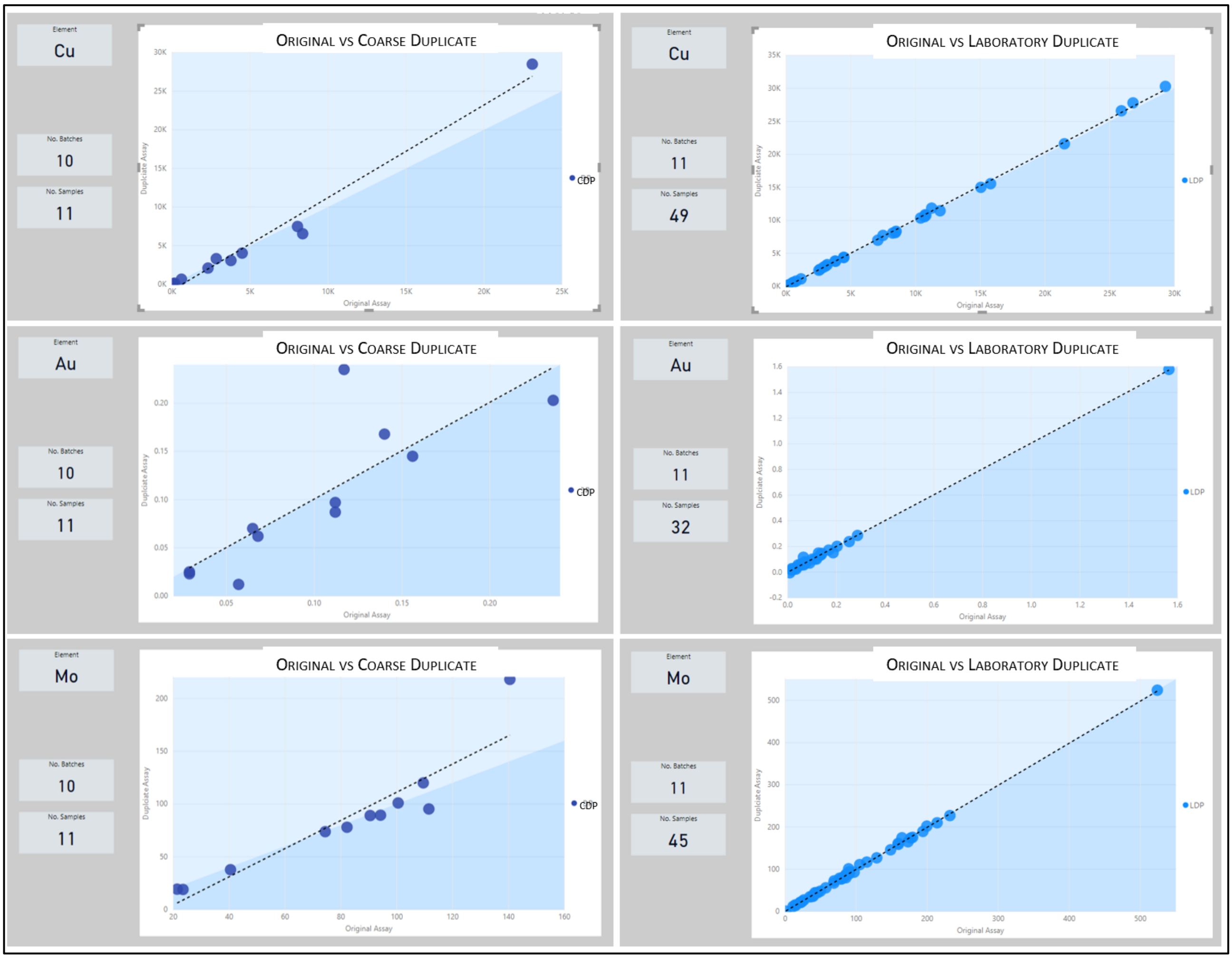
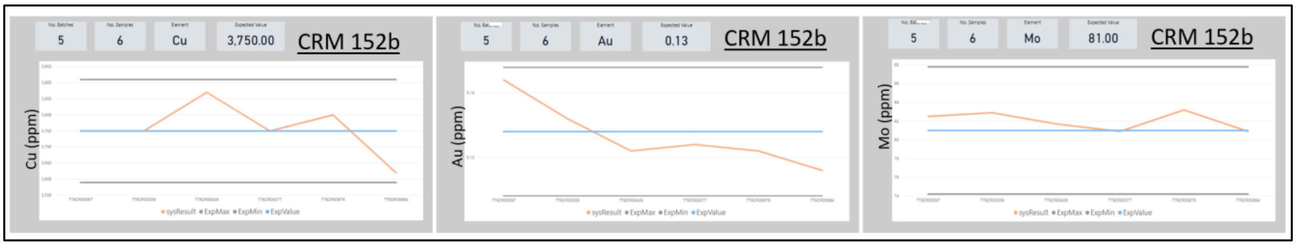


Figure 11-13 Control charts showing Cu, Au, and Mo results from the CRM sample analysed for the FQM diamond drill campaign



11.7 Sequential Copper Leach Analysis (CuSeq)

Sequential Copper Leach (CuSeq) Analysis was used to support interpretation of copper mineralisation style, including oxide, secondary sulphide, primary sulphide, and refractory copper, by quantifying relative proportions of sulphuric acid–soluble, cyanide-soluble, and residual copper.

Sequential Copper Leach Analysis were completed for approximately 13% of the sample population used in the Mineral Resource estimation (Figure 11-14) on pulp samples analysed at ALS Minerals Mendoza and ALS Lima at various times following primary assay (Table 11-2). In 2019, sequential copper analyses were undertaken on metallurgical drill core samples that were vacuum sealed and dispatched immediately to the laboratory to minimise oxidation; these results are considered most representative of in-situ copper speciation.

Earlier analyses of selected Lumina samples in 2012, and subsequent re-analysis by FQM in 2014, demonstrated time-dependent oxidation effects in stored pulps, expressed as increasing acid-soluble copper and decreasing cyanide-soluble copper, particularly under humid storage conditions. Additional analyses conducted in 2016 and 2017 further constrained these effects.

Due to the limited number of samples, restricted spatial coverage, and variable oxidation impacts, sequential copper values were not interpolated into the block model. Instead, CuSeq results were normalised to total copper grades (ICP or ore-grade AAS) and used qualitatively to support definition of dominant copper mineralogy within geological and geometallurgical domains, consistent with the approach described in Chapter 14.

An oxidation-rate study initiated in 2019 using coarse-crushed metallurgical samples indicated that copper mineral oxidation under both laboratory and site exposure conditions is limited over a twelve-month period and unlikely to materially affect copper speciation during mining and processing.

Figure 11-14 Spatial distribution of samples (shown in red) with sequential leach copper assay results

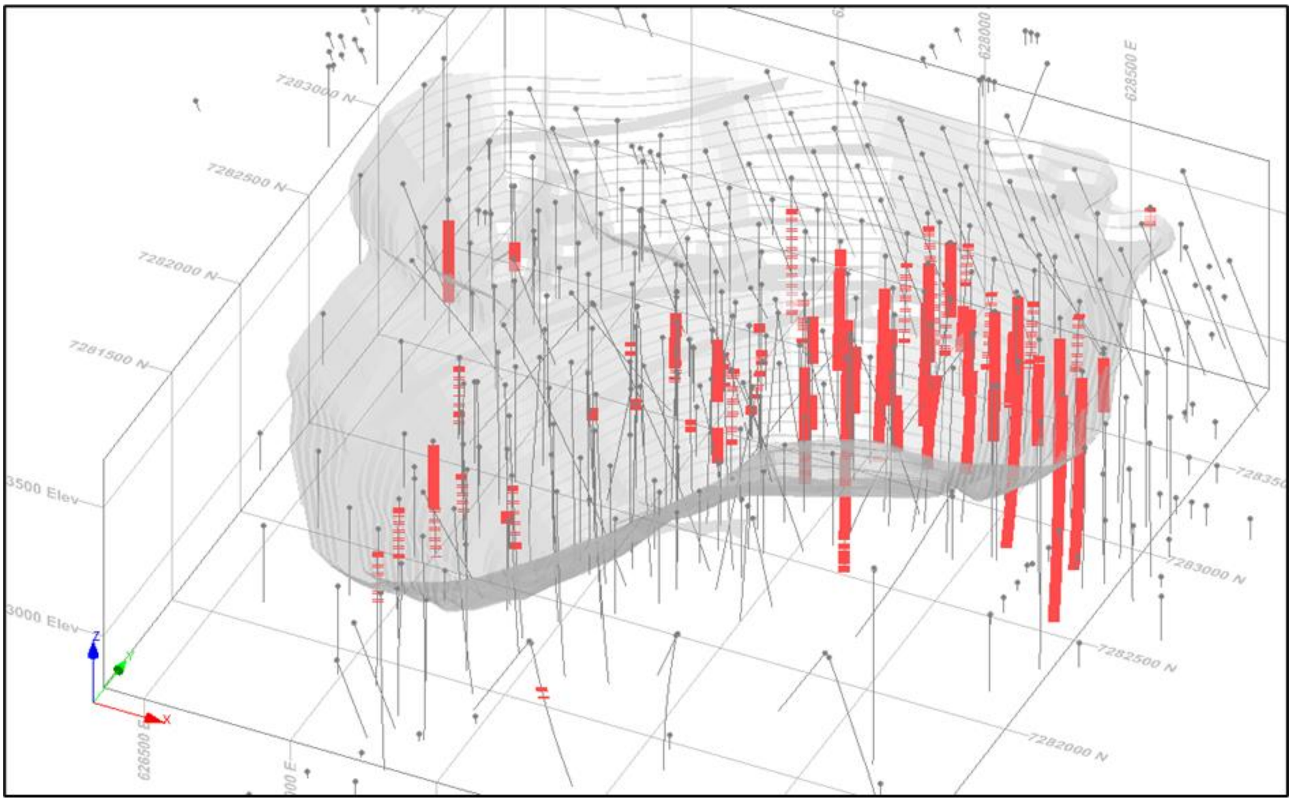


Table 11-2 Phases of dispatch for sequential leach copper analysis

Dispatch company	Year of dispatch	Number of samples	Year samples were drilled	Sample type	Estimated time in storage prior to dispatch
Lumina	2012	2194	2010-2012	pulp residuals	<1-2years
FQM	2014	149	2012	pulp residuals	2-3 years
FQM	2016-2017	4494	2010-2012	pulp residuals	4-7 years
FQM	2019	737	2019	drill core	Negligible

11.8 Factors materially affecting the Mineral Resource estimate

It is the Qualified Person's opinion that sample preparation, analytical procedures, and data management protocols have resulted in consistent and repeatable analytical results for most samples used in the Mineral Resource Estimate. Review of QA/QC data indicates that appropriate controls were implemented and that assay results are reliable. Sample values are considered representative of the mineralisation and suitable for use in the Mineral Resource Estimate. Historical data with limited supporting documentation, predominantly relate to peripheral areas of the deposit and are not considered to materially affect the quality or reliability of the estimate.

ITEM 12 DATA VERIFICATION

Data verification was undertaken by the QP, David Gray, through review of drilling, sampling, analytical methods, QA/QC results, database integrity, and geological interpretation as is consistent with the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines, 2019.

Collar coordinates were reviewed against a high-resolution topographic surface and through field spot checks using hand-held GPS measurements compared with the drilling database. No material discrepancies were identified in easting or northing coordinates. Comparison of collar elevations against the topographic surface identified a small number of holes with elevation differences greater than 5 m. These were corrected before been used in the Mineral Resource Estimate and do not affect estimates.

Geological logging and sampling records were compared against available stored drill core on a selective basis. No material inconsistencies were observed.

Analytical methods and QA/QC protocols were reviewed and indicate that appropriate controls were in place to monitor accuracy, precision, and contamination as per Item 11.

Bias checks between sample types and analytical methods indicate marginal to no systematic bias.

Database validation checks were completed including of assay values against original laboratory certificates (from 2008 onward), review of outlier values, resolution of duplicate or overlapping records, verification of downhole survey data, and confirmation of consistent recording of key metadata.

Residual pulp samples from the Lumina drilling programme that were subsequently dispatched by the Company for sequential copper leach analysis provided an additional check on total copper and sequential copper values. No material issues were identified, although some oxidation of stored pulp samples was noted.

Three-dimensional geological models were developed using integrated datasets and were reviewed against re-logging of stored drill core and available core photography.

Based on the data verification completed, the QP considers the available drilling, sampling, analytical, QA/QC, and database information to be of sufficient quality and reliability for use in the estimation of the Taca Taca Mineral Resource.

ITEM 13 MINERAL PROCESSING AND METALLURGICAL TESTING

Much of the metallurgical testwork performed by the previous owners was carried out at Plenge Laboratories in Lima between 2010 and 2012, under the supervision of Lumina personnel. This testwork was summarised by Pincock Allen & Holt in 2012 (2012) and included in the Preliminary Economic Analysis (PEA) prepared for Lumina by Ausenco dated May 2013.

Between 2012 and 2019 there was minimal additional metallurgical work undertaken on the flotation of copper mineralisation from the Taca Taca deposit. Some work was carried out in 2017 on existing core to assess the possibilities of gold recovery from near-surface mineralisation that could be mined from within each pit phase.

In 2019, four drill holes were completed to provide ten metallurgical samples for flotation testwork and using brine solutions sourced from site. These samples represent the first five years of plant feed mined according to the FQM proposed mining plan. This latest testwork was completed at ALS in Kamloops, Canada in July 2020, and is documented in their report dated September 2020

13.1 Project mineralisation

Further to the commentary in Item 7, there are essentially two ore types that will present for processing, i.e., secondary (supergene) and primary (hypogene) ores. There is an overlying leached cap (auriferous but leached of copper).

Primary ore is defined as cupriferous ore containing more than 50% of the total copper as chalcopyrite. The highest proportion of copper in chalcopyrite in all the samples tested to date was less than 90% of the total copper assay. Primary ores also appear to contain between 5% and 20% of the copper present, according to the sequential copper analysis, as oxide copper minerals.

Similarly, supergene ores contain a maximum of 75% to 80% of the total copper in cyanide soluble copper sulphide minerals (chalcocite and bornite), with up to 5% chalcopyrite, and the remainder in acid soluble copper minerals that are all ill-defined at present.

13.2 Testwork by the previous Project owners

The testwork programmes initiated by Lumina looked at two basic ore types - secondary (supergene) and primary (hypogene) ores. An overlying leached cap containing discrete gold mineralisation[6] was not considered as plant feed, but was metallurgically investigated for potential gold recovery to a scoping study level of detail.

Primary ores also appeared to contain between 5% and 20% of the copper present as oxide copper minerals, according to a sequential copper analysis.

Supergene ores contained a maximum of 75% to 80% of the total copper in cyanide soluble copper sulphide minerals (chalcocite and bornite), with up to 5% chalcopyrite, and the remainder in acid soluble copper minerals that were all ill-defined.

13.2.1 Testwork summary

Comminution testwork indicated that the ore is soft (primary ore) to moderately soft (supergene ore) as defined in the JK Tech (JK) data base. Bond work indices were moderately high at 16.4 and 18.7 kWh/t for primary and supergene ores, respectively. These ores would be amenable to SAG – ball milling. Due to the

[6] Referred to as the “perched” gold horizon and also as Domain 102.

high Axb numbers from the JK testwork, it appeared that secondary crushing of the mill feed would not be required.

Flotation testwork described in the Plenge reports defined optimum conditions for flotation (grind, reagent suite, pH, slurry density, etc.) and provided recovery estimates and achievable concentrate grades. Reported average recoveries over the life of mine would be approximately 82.9% for Cu from supergene ores and 87% from primary ores. Recoveries of 55% for molybdenum and 60% for gold would be achievable for both ore types. The optimum primary grind was defined as 80% passing 150 µm, but a regrind of rougher concentrates to 80% passing 30 µm would be required to achieve high copper concentrate grades.

A majority of the testwork was conducted in Lima tap water, with little testing undertaken using water from site. Testwork using saline water from site for rougher flotation recovery continued to use Lima tap water for cleaner flotation recovery. Limited flotation testwork in brine from site indicated that rougher flotation recovery results were similar to those in tap water, but the recovery in the cleaners and the concentrate grade dropped off significantly when using brine.

The process flowsheet in the 2021 Technical Report thus described rougher flotation in brine, followed by dewatering of rougher concentrates and then regrind in better quality water.

Few samples representing the first years of operation were tested. Two supergene and two primary ore composite samples, one of each representing the Lumina planned first five years of operations and the second pair representing years six to ten, were subject to locked cycle flotation work, but only in tap water.

Consequently, an additional metallurgical testwork programme was undertaken by the Company in 2019 and 2020, focussing on the starter pit as defined by a preliminary FQM mine plan. This testwork was undertaken in water collected from site, using brine from the salar for grinding and rougher flotation testing, followed by cleaner flotation testing using fresh or brackish water sourced from regional boreholes.

13.2.2 Sample mineralogy

Table 13-1 lists the copper mineralogy of the two composite samples used for the original testwork.

Table 13-1 Mineral species (%) in testwork samples

Composite	Cu bornite	Cu chalcopyrite	Cu chalcocite
Primary	11.2	57.5	31.3
Supergene	35.1	13.4	51.5

Two mineral liberation analyses (MLA) were performed by the Centre for Advanced Mineral and Metallurgical Processing at the University of Montana in October 2010. The samples were rougher concentrates, with one sample from supergene ore flotation, and the second sample from primary ore flotation. MLA work was performed on four different size distributions from each concentrate. The coarsest size looked at was +150 mesh (106 µm), and at that size most of the sulphide particles were liberated.

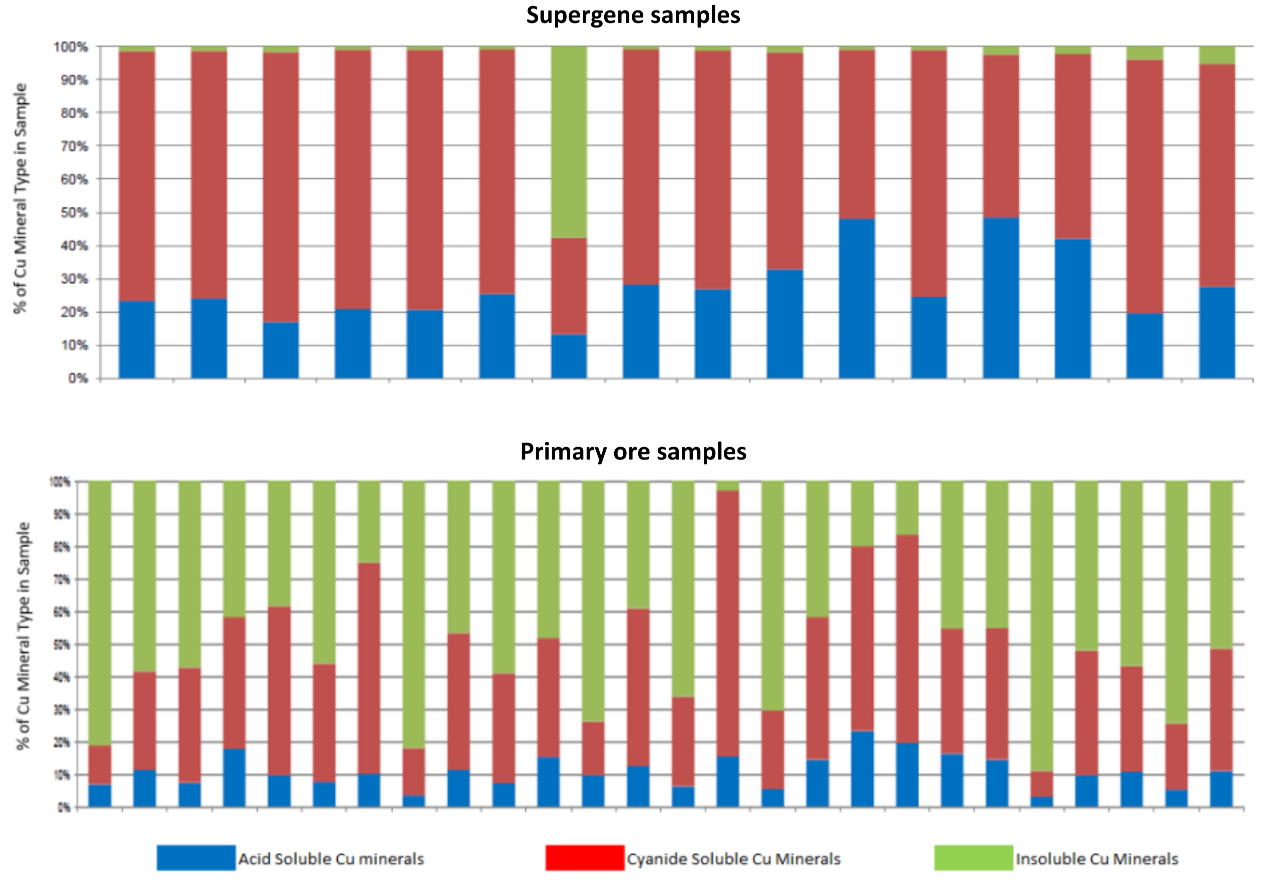
The university reported four different copper minerals present in each ore type, the fourth being covellite (CuS). No oxide copper minerals were reported. Table 13-2 provides the proportion of Cu represented by each mineral in the concentrates.

Table 13-2 Cu distribution in rougher concentrate, by mineral species

Mineral	Supergene	Primary
Chalcocite	50 to 60%	0 to 13%
Chalcopyrite	14 to 17%	50 to 56%
Bornite	21 to 28%	26 to 36%
Covellite	2 to 4%	8 to 12%

Sequential copper analyses were conducted by Plenge on forty samples (fifteen supergene and twenty five primary ore samples) provided for variability testing. Figure 13-1 provides a graphical representation of the proportion of acid soluble copper, cyanide soluble copper and insoluble copper in the forty samples.

Figure 13-1 Sequential copper analyses on supergene and primary ore samples



Two supergene ore samples contained 45% to 50% of the total copper present in oxide copper minerals, and fourteen of the fifteen samples contained over 20% of the copper in oxide minerals. However, the mineralogical evidence (where no oxide minerals were identified) does not support the sequential copper analysis. Similarly, the good flotation performance evidenced in the testwork results does not support the presence of oxide minerals.

Covellite, chalcocite and bornite are highly soluble in cyanide solution; the dissolution rate for chalcocite and covellite is reported as ‘fast’, whilst for bornite it is reported as ‘moderately fast’. All three dissolve slowly in dilute sulphuric acid, but only to between 1% and 5%; hence, these minerals are unlikely to have been the cause of the elevated acid soluble copper assays. Further work is required to fully understand the contradictions between the mineralogical examinations and the sequential copper assays, and to confirm the absence of oxide copper minerals.

There appears to be no ‘pure’ primary ore in the Taca Taca deposit. All primary ore samples received and tested contained a minimum of 20% secondary copper minerals, and all contained acid soluble copper minerals (with 15 out of 25 samples assaying more than 10% AsCu). This appears to have had a positive effect on concentrate grades achievable during processing but may have had a negative effect on recoveries.

No mention was made in any of the metallurgical testwork conducted for Lumina of differing mineralisation styles within the orebody, except for the variability work. This variability work was evaluated by FQM, leading to an interpretation of mineralisation domains. The latest FQM testwork (i.e., 2019 to 2020) has looked at the impact of these domains on metallurgical response.

13.2.3 Comminution

SMC testing was completed on two samples from the deposit, one representing supergene, and the other primary ore. These tests were reported in the Plenge Scoping Study report. Ten samples (four supergene and six primary) were also sent for semi-autogenous grinding (SAG Design) testwork by Starkey & Associates in Ontario, Canada. Additionally, numerous samples were subjected to Bond ball mill work index testing (BWi) during the course of the flotation testwork conducted by Plenge. A summary of all these tests programmes gives the average results listed in Table 13-3.

Table 13-3 Comminution testwork results

Parameter	Units	Supergene Ores		Primary Ores	
		Average Value	Tests	Average Value	Tests
SG		2.70	4	2.71	6
Compressive Strength	Mpa	12.24	4	12.8	6
Crusher Wi	kWh/t	7.35	15	8.04	25
SAG Design Pinion Energy	kWh/t	9.46	4	8.9	6
JK Axb		68.8	2	60.9	2
Rod Mill Wi	kWh/t	14.48	4	13.56	6
Ball Mill Wi	kWh/t	18.69	19	16.41	31
Abrasion Index	g	0.20	4	0.23	6

According to the JK data base, the primary ore can be classified as moderately soft, whilst the supergene ore can be classified as soft. However, the supergene ore has a higher BWi than the primary ore and will require additional ball milling power to achieve the desired grind.

The SAG design results and the JK results are not in agreement; the SAG design numbers indicate relatively high SAG mill power requirements. Further work on the ten more recent metallurgical samples (i.e., FQM testwork) gave much lower Axb results, indicating a much tougher ore than suggested by the JK results in Table 13-3.

13.2.4 Flotation

Initial testwork was carried out at the Plenge Laboratories in Lima, Peru, on two composite samples, one of supergene and one of primary ore. Each had a head grade similar to the expected feed grades for the Project. Initial results indicated good recoveries of copper to high grade concentrates (about 34% Cu), as shown in Table 13-4.

Table 13-4 Scoping Study locked cycle test results

Ore Type	Head Grades (Calculated)			Bulk Concentrate Assays			Recovery %		
	Au ppm	Cu %	Mo, ppm	Au ppm	Cu %	Mo %	Au	Cu	Mo
Supergene	0.11	0.60	207	4.2	33.9	0.55	51.3	86.9	45.8
Primary	0.10	0.43	200	4.86	34.5	0.95	48.7	90.1	54.1

Head grades for the composite samples were similar to the expected feed grades for the Project.

The primary grind was 80% passing 150 µm, with a regrind size of 80% passing 30 µm. It appeared that concentrates were very sensitive to the regrind size, i.e. at 45 µm, copper concentrate grades were only 22.6% Cu. However, at finer regrind sizes, gold recovery dropped off, but molybdenum recovery and grades in the bulk concentrate improved (Table 13-5 and Table 13-6).

Table 13-5 Batch cleaner flotation test results at different concentrate regrind sizes

Regrind Size, µm	Bulk Concentrate Assays			Recovery %		
	Au ppm	Cu %	Mo %	Au	Cu	Mo
Supergene Ore						
70	2.13	17.55	0.335	49.5	80.8	50.2
60	2.81	19.70	0.352	55.5	79.1	45.4
45	2.43	22.60	0.443	49.3	82.0	49.8
30	3.53	31.49	0.395	50.3	80.6	32.0
Primary Ore						
50	2.54	24.4	0.698	38.9	78.2	51.9
40	2.92	26.6	0.819	38.8	77.4	54.1
33	3.84	33.4	0.676	36.7	75.1	35.5
25	4.07	37.2	0.702	37.9	74.5	33.4

Table 13-6 Recovery variability with concentrate regrind size for locked cycle tests

Regrind Size, µm	Head Grades (Calculated)			Bulk Concentrate Assays			Recovery %		
	Au ppm	Cu %	Mo %	Au ppm	Cu %	Mo %	Au	Cu	Mo
Supergene Ore									
40	0.14	0.59	0.020	3.78	25.28	0.633	53.2	83.8	62.3
35	0.15	0.57	0.021	3.68	31.85	0.837	36.7	84.0	59.7
30	0.12	0.57	0.018	4.20	33.89	0.547	51.3	86.9	45.8
Primary Ore									
40	0.12	0.41	0.018	3.88	29.19	0.973	41.3	88.0	66.8
35	0.16	0.40	0.017	4.68	31.54	1.140	33.7	89.6	75.3
30	0.13	0.40	0.018	4.86	34.49	0.946	39.5	90.1	54.1

A bulk flotation test performed on the same composites to provide concentrates for Cu-Mo separation achieved lower recoveries of 85.2% and 85.1% Cu for supergene and primary ores, respectively. Gold recoveries from supergene ores also dropped to 37.3%.

Optimisation testing was performed on one supergene and one primary ore composite. Head grades are listed in Table 13-7.

Table 13-7 Head grades of testwork composites

Composite	Head Grades						
	Fe %	Cu %				Au ppm	Mo %
		Acid Soluble	CN Soluble	Insoluble	Total		
Supergene	0.96	0.14	0.58	0.01	0.75	0.08	0.033
Primary	0.70	0.07	0.23	0.13	0.45	0.16	0.028

Both samples contained relatively low sulphide S and Fe levels and had a higher proportion of secondary Cu minerals when compared with the composites reported in the first two test reports.

Tri-cleaner and locked cycle flotation tests were run at the following conditions:

- primary grind at 80% passing 150 µm
- flotation at 37% solids and pH 9

- concentrate regrind size at 80% passing 30 µm

Testwork was conducted initially in Lima tap water. Comparative tests were then conducted using brine sourced from site for both the rougher and cleaner flotation testing and using a combination of brine for rougher flotation and tap water for cleaner flotation. Recoveries and concentrate grades achieved in this testwork were as listed in Table 13-8.

Table 13-8 Concentrate grades and recoveries in batch cleaning tests using Lima tap water and brine solutions

Ore Type	Water	Bulk Concentrate Assays			Recovery %		
		Au ppm	Cu %	Mo %	Au	Cu	Mo
Supergene	Tap	1.9	34.0	1.10	49.6	86.1	57.5
	Brine at pH 4.4	2.4	34.0	0.75	33.0	59.0	28.3
	Brine at pH 7.6	2.2	40.8	0.84	31.3	68.4	32.0
	Brine pH 7.6 rougher tap water for cleaners	1.6	26.1	0.95	48.0	84.5	68.3
Primary	Tap	6.3	38.6	1.3	51.3	76.2	35.9
	Brine at pH 4.4	5.6	22.7	1.3	20.8	29.3	20.7
	Brine at pH 7.6	8.1	31.7	1.4	42.6	42.4	25.4
	Brine pH 7.6 rougher tap water for cleaners	5.05	26.5	1.68	53.1	82.7	73.9

Note that these were batch cleaner tests; locked cycle tests would be expected to give higher recoveries and lower concentrate grades. This is confirmed by the results for flotation in tap water as listed in Table 13-8 (batch cleaning) and Table 13-9 (locked cycle).

Table 13-9 Concentrate grades and recoveries to bulk Cu-Mo concentrate – locked cycle tests

Ore Type	Water	Bulk Concentrate Assays			Recovery %		
		Au ppm	Cu %	Mo %	Au	Cu	Mo
Supergene	Tap	1.9	31.4	1.15	60.8	89.9	72.2
Primary	Tap	5.4	30.6	1.91	62.4	92.8	84.0

Compared with the previous locked cycle test results (Table 13-4), copper recoveries were higher by about 3%, whilst concentrate grades dropped by 2.5% to 4%. The sample head grades were higher than for the previous test, which may have contributed to the higher recovery.

For supergene ores, batch cleaning gave 86.1% copper recovery at a concentrate grade of 34.0% Cu; the locked cycle tests yielded 89.9% recovery at a 31.4% Cu concentrate grade. Similarly, the primary ores achieved 76.2% recovery (which appears low) at 38.6% concentrate grade in batch flotation compared with 92.8% recovery at 30.6% grade for locked cycle flotation. Copper recoveries dropped significantly when using brine solution in the cleaners, although concentrate grades were not seriously impacted, (apart from one test). As noted, the primary ore results for the test in Lima tap water alone appear anomalous.

Gold recovery for both samples was about 50% in tap water. Using brine in rougher flotation followed by tap water in the cleaner circuit this dropped to about 32% for supergene ores and to 20.8% (brine at pH 4.4) and 42.6% (brine at pH 7.6) for the primary ore sample. Mo recoveries also dropped in brine flotation but were significantly improved when tap water was used in the cleaner flotation, after rougher flotation using brine. In fact, both Au and Mo recoveries in the rougher float using brine, followed by cleaner flotation in tap water, were higher than using only tap water alone. The reasons for this are unclear.

It is clear from these results that the use of brine in the cleaner circuit has a detrimental effect on recoveries, and thus the process flowsheet must incorporate brine for rougher flotation, followed by a dewatering stage and with subsequent cleaner flotation in good quality water. This circuit was replicated in the laboratory by filtering the rougher concentrates and re-diluting with fresh water prior to cleaner flotation tests.

It should be noted, that in practice at plant scale, the rougher concentrates (in brine) would be dewatered in a thickener to between 55% and 60% solids before being reground and re-diluted in low chloride water. In the laboratory, the rougher concentrates were filtered to about 85% solids prior to cleaner flotation and thus contained significantly less entrained brine than would occur in the plant.

The above tests were repeated in the final testwork as reported by Plenge. Results are shown in Table 13-10.

Table 13-10 Locked cycle tests in brine and brine followed by tap water

Ore Type	Water	Bulk Concentrate Assays			Recovery %		
		Au ppm	Cu %	Mo %	Au	Cu	Mo
Supergene	Tap	4.12	38.6	0.41	51.5	86.4	53.9
	Brine & Tap	3.69	29.2	0.30	63.4	86.5	75.0
Primary	Tap	6.91	36.2	1.02	64.7	92.5	72.7
	Brine & Tap	7.35	33.9	0.69	64.9	91.2	52.7
Blend (1:1)	Tap	5.3	35.9	0.70	63.5	87.8	67.4
	Brine & Tap	4.21	29.3	0.64	61.7	88.8	75.0

In these tests, copper recoveries were approximately the same using tap water or brine followed by tap water, but concentrate grades were lower (and the mass pull higher) when brine was used in rougher flotation. For the supergene sample, both Mo and Au recoveries were higher in flotation using brine and tap water, than in tap water alone.

13.2.5 Samples representing plant feed to be mined in Lumina's first ten years of operations

Four composite samples were tested by Plenge that represented the first ten years of operations as then envisaged. Two composites were produced for each of the supergene and primary ores, and the sample head grades are listed in Table 13-11. In the table, Supergene 1 and Primary 1 refers to composite samples representing years 1 to 5, whilst Supergene 2 and Primary 2 are samples for years 6 to 10. The proportion of soluble and insoluble copper in these composites is shown graphically in Figure 13-2.

Table 13-11 Head grades for samples representing plant feed in the first ten years of operations

Composite	Head Grades						
	Fe %	Cu %				Au ppm	Mo %
		Acid Soluble	CN Soluble	Insoluble	Total		
Years 1 to 5							
Supergene 1	1.25	0.17	0.49	0.06	0.72	0.16	0.022
Primary 1	1.8	0.06	0.15	0.38	0.61	0.19	0.022
Years 6 to 10							
Supergene 2	1.72	0.20	0.53	0.10	0.84	0.13	0.017
Primary 2	3.93	0.04	0.13	0.27	0.46	0.11	0.017

Figure 13-2 Graphic representation of sequential copper assays for samples representing plant feed in the first ten years of operations

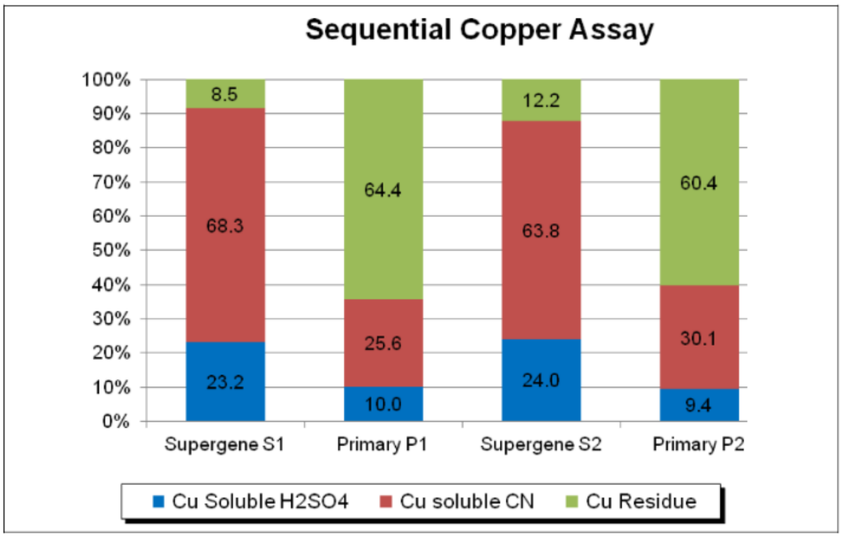


Figure 13-2 shows that the primary ores contain about 10% acid soluble copper (copper oxide minerals), and 20 to 30% cyanide soluble copper (secondary sulphides).

Locked cycle tests were conducted in tap water on large sample masses to produce concentrate for Cu-Mo listed in Table 13-12. These results are superior to the locked cycle tests performed previously, and this is possibly because of the higher proportion of insoluble copper minerals in the feed samples.

Table 13-12 Locked cycle test results on samples representing the first ten years of operations

Ore Type	Bulk Concentrate Assays				Recovery %			
	Ag ppm	Au ppm	Cu %	Mo %	Ag	Au	Cu	Mo
Years 1 to 5								
Supergene 1	20.2	5.38	34.48	0.80	48.6	60.2	90.6	69.7
Primary 1	21.2	6.62	28.74	0.77	46.6	71.7	94.2	86.1
Years 6 to 10								
Supergene 2	12.8	4.56	33.26	0.69	36.8	67.7	90.2	85.8
Primary 2	21.7	4.70	29.70	0.80	21.00	60.9	94.7	73.9

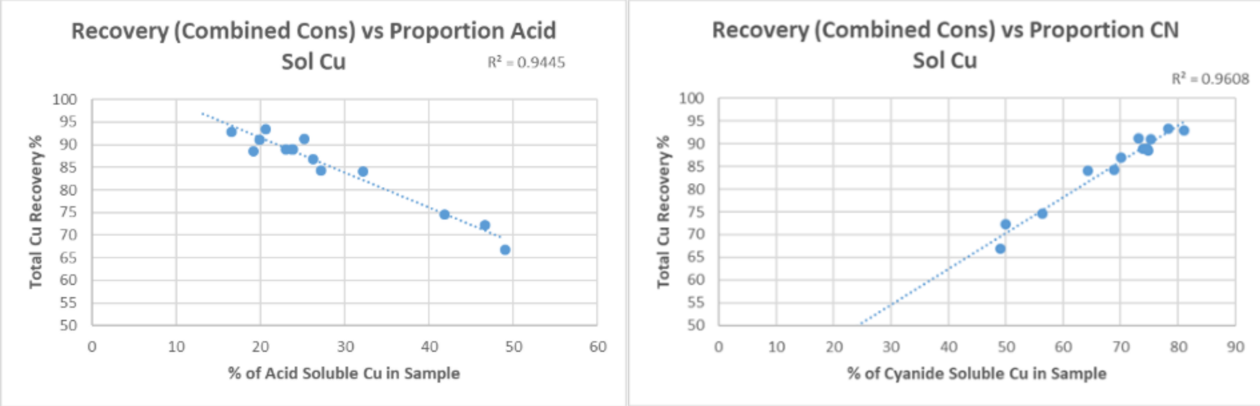
13.2.6 Variability testing

Forty samples (fifteen supergene and twenty-five primary samples) were tested for variability in their response to flotation in tap water at the optimum flotation conditions.

The sequential copper analyses for these samples were presented previously in Figure 13-1.

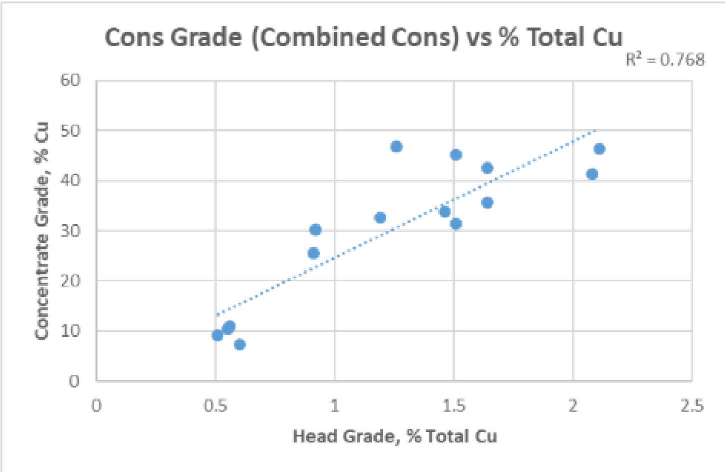
The supergene samples had higher copper grades than the primary samples, ranging from 0.51% to 2.11% Cu, and averaging 1.23% Cu. Recoveries in batch cleaner flotation varied between 66.8% and 94.5%, with an arithmetic average of 85.8%, whilst the concentrate grades were 7.4% to 46.9%, with an average of 30.0%. There appeared to be a strong correlation between recovery and cyanide soluble copper in the samples, and an inverse relationship between recovery and acid soluble copper in the samples (Figure 13-3).

Figure 13-3 Cu recovery from supergene ore samples vs cyanide acid soluble and cyanide soluble content



Similarly, there was a reasonable correlation between concentrate grade and % total copper in the feed samples (i.e., as expected, Figure 13-4).

Figure 13-4 Concentrate grade vs sample head grade



For the primary ore variability samples, the head grade varied between 0.31% and 0.89% Cu, with an average of 0.49% Cu, which is close to the expected production feed grade. Recoveries varied between 77.5% and 95.1%, with an average of 89.3%, and concentrate grades were 13.0% to 32.5% (average 23.0%). There were no clear correlations between head grades and recovery or concentrate grade as there were for the supergene ore samples. However, on repeating the above graphs for the combined supergene and primary samples, the overall trends are similar (Figure 13-5 and Figure 13-6).

Figure 13-5 Cu recovery from combined variability samples vs acid soluble and cyanide soluble content

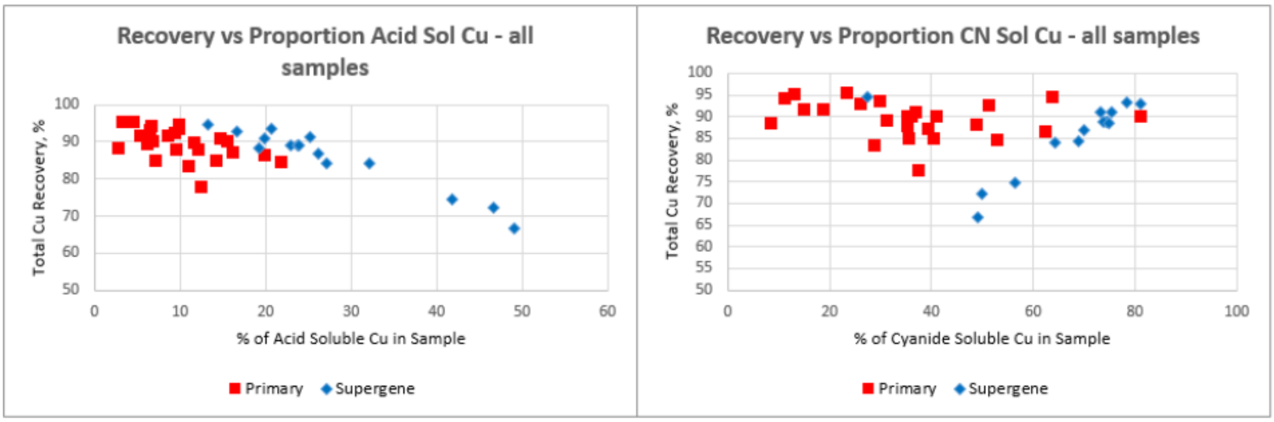
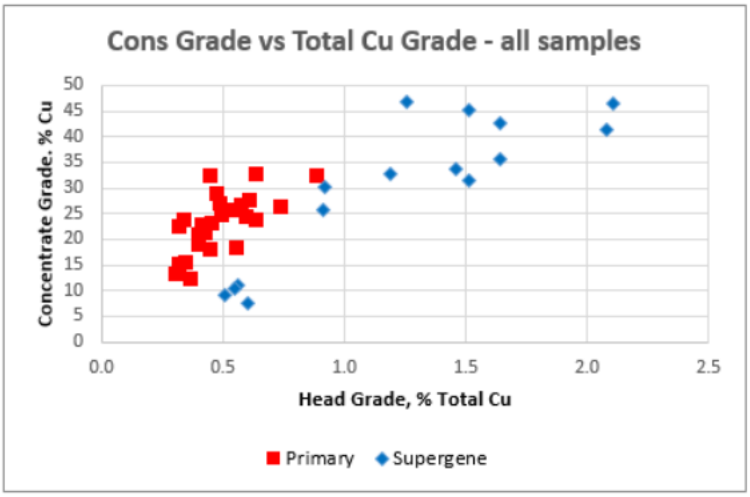


Figure 13-6 Concentrate grade vs combined sample head grade



13.2.7 Copper-molybdenum separation

The initial copper-molybdenum (Cu-Mo) separation testwork was reported by Plenge in 2010.

60 kg each of supergene and primary composite samples tested[7] were subjected to a series of ten locked cycle tests to produce large enough quantities of bulk concentrates so that Cu-Mo separation testing could be performed. The main objective was to determine if saleable molybdenum concentrates could be produced from the bulk concentrates while maintaining high molybdenum recovery to the molybdenum concentrate. For these tests, the optimised flowsheet, primary grind size and reagent schemes determined in the previous testing were utilised; however, the regrind size was reduced from 30 µm to 18 to 22 µm to obtain improved molybdenum liberation.

The separation tests were run in open circuit with six stages of cleaning for molybdenum. Results indicated that molybdenum concentrates assaying at 49% Mo could be produced from each composite. Molybdenum recoveries from the supergene and primary composites were 44% and 53%, respectively. Molybdenum recoveries to bulk concentrates, before separation, were 59% for supergene and 63% for primary samples, meaning that 10% to 15% of the molybdenum contained in the bulk concentrates would not be recovered into the molybdenum concentrate. This level of rejection or loss is typical for similar projects.

Copper concentrate grades were slightly higher in the primary composite tests than in the previous tests while copper recoveries were slightly lower. Table 13-13 presents the results of the Cu-Mo separation tests for these materials.

[7] Refer to Table 13-7 for the head grade analyses

Table 13-13 Copper-molybdenum separation test results

Product	wt %	Concentrate Assays					Recovery %			
		Ag ppm	Au ppm	Cu %	Mo %	Fe %	Ag	Au	Cu	Mo
Supergene										
Bulk Con	1.54	21.12	3.65	31.03	0.76	25.20	43.7	37.4	85.2	59.3
Moly Con	0.02	2.40	0.82	0.94	49.81	1.00	0.1	0.1	0.03	44.1
Copper con	1.52	21.33	3.68	31.38	0.2	25.50	43.7	37.3	85.2	15.2
Primary Ore										
Bulk Con	1.00	31.60	4.58	36.01	1.17	24.80	38.4	38.6	85.1	62.5
Moly Con	0.02	5.00	1.26	1.09	49.19	6.10	0.1	0.2	0.1	52.5
Copper con	0.98	32.15	4.65	36.72	0.19	25.10	38.2	38.4	85.1	10.0

Further Cu-Mo separation testwork was carried out in 2012 and reported by Plenge.

Each of the four composites representing the first ten years of operation as then envisaged was subject to locked cycle flotation to produce a Cu-Mo bulk concentrate for separation testwork. The primary grind was 150 µm, and the concentrate regrind size was 30 µm. Nine stages of molybdenum cleaning were employed, and approximately 1.8 kg/t of NaHS was used for xanthate destruction. Molybdenum cleaning was performed in open circuit. Test results are listed in Table 13-14.

All four composites achieved a molybdenum concentrate containing greater than 40% Mo at between 47% and 73% Mo recovery. Average results were 57% recovery to a concentrate of 47% Mo. The main diluents in the concentrate were iron, copper and insolubles such as carbon and silica gangue. Due to limitations in sample availability, only limited testwork could be performed, but it is believed that with finer concentrate regrind sizes and additional testing, these results could be improved.

It should be re-emphasised that this work was performed in Lima tap water. No Cu-Mo separation tests have been performed in brackish water from site. Furthermore, no additional work on Cu-Mo separation was performed in the subsequent FQM testwork programme at ALS.

Table 13-14 Copper-molybdenum separation tests for samples representing the first ten years of operations

Product	wt %	Concentrate Assays					Recovery %			
		Ag ppm	Au ppm	% Cu	Mo %	Fe %	Ag	Au	Cu	Mo
Supergene S1										
Bulk Con	1.79	20.40	5.50	34.90	0.62	23.00	46.8	58.0	87.9	54.0
Moly Con	0.02	2.40	1.24	1.90	45.30	3.25	0.1	0.2	0.1	46.0
Copper Con	1.77	20.60	5.55	35.30	0.09	23.21	46.7	57.8	87.9	8.0
Supergene S2										
Bulk Con	2.18	13.50	4.48	33.00	0.61	25.60	36.4	64.3	86.2	76.0
Moly Con	0.02	2.00	1.01	0.95	49.60	2.50	0.1	0.1	0.0	62.3
Copper Con	2.16	13.60	4.52	33.30	0.11	25.80	36.3	64.2	86.2	13.6
Primary Ore P1										
Bulk Con	2.11	19.60	6.36	28.40	0.69	30.30	44.9	71.0	95.0	80.6
Moly Con	0.02	6.00	0.88	1.30	53.20	2.00	0.2	0.1	0.0	73.0
Copper Con	2.08	19.70	6.42	28.70	0.06	30.60	44.8	70.9	95.0	75.0
Primary Ore P2										
Bulk Con	1.48	21.80	4.96	0.74	28.00	20.80	59.9	91.4	65.2	62.5
Moly Con	0.02	5.30	0.90	4.40	41.50	6.80	0.1	0.1	0.2	47.2
Copper Con	1.46	22.00	5.01	29.60	0.21	28.30	20.7	59.8	91.2	18.0
Averages										
Moly Con		3.90	1.00	2.14	47.40	3.63	0.1	0.14	0.1	57.1
Copper Con		19.00	5.38	31.70	0.12	27.00	37.1	63.2	90.1	11.8

13.2.8 Oxide copper leaching and flotation tests

Bottle roll leaching of supergene and primary ore samples in sulphuric acid is described in a Plenge report of 2010. The sample mineral contents are listed in Table 13-11.

Leach tests were run for 96 hours in 10 g/L acid, yielding the results listed in Table 13-15.

The extractions recorded in Table 13-15 reflect the mineralogy of the samples, with low copper recoveries obtained from the primary samples due to the large proportion of chalcopyrite present. However, the supergene samples also exhibit low leach recoveries; chalcocite and bornite being slow leaching minerals in acid.

This work was repeated on three oxide copper samples, designated high grade, medium grade and low grade. The testwork is described in a Plenge report dated October 2012 and the results are listed in Table 13-16.

Table 13-15 Bottle roll leaching tests for copper recovery

Sample	Size	Head Grade	Extraction	Gangue Acid
		Cu %	%	Consumed, kg/t
Supergene	100% - 1.7 mm	0.61	29.5	19.4
	80% - 75 micron	0.58	49.1	19.2
Primary	100% - 1.7 mm	0.39	17.3	19.1
	80% - 75 micron	0.40	26.4	19.3

Table 13-16 indicates that the copper recovery from oxide samples does not appear to be dependent on crush size but is generally low. The economic viability of heap leaching this material appears to be marginal. Scoping level flotation testwork on these samples indicated that about 70% Cu recovery to a concentrate grading 43% Cu could be achieved on the higher grade sample. Therefore, no further work on the leaching of oxide copper ores was undertaken.

Table 13-16 Bottle roll leaching tests for oxide copper

Sample	Size	Head Grade	Extraction	Gangue Acid
		Cu %	%	Consumed, kg/t
Low Grade	100% - 1.7 mm	0.13	57.5	7.5
	80% - 75 µm	0.13	51.3	7.4
Med Grade	100% - 1.7 mm	0.17	28.3	7.4
	80% - 75 µm	0.17	39.1	6.0
High Grade	100% - 1.7 mm	0.30	28.0	8.0
	80% - 75 µm	0.30	33.3	6.5

13.2.9 Sedimentation and filtration testwork

Testwork[8] on sedimentation and filtration of various products from flotation of supergene and primary composite samples is described in two testwork programmes conducted by Plenge:

1. Metallurgical Scoping Study, Plenge 7758-62/7763-67, September 2010, and
2. Comminution, Cu-Mo Separation, Variability, Oxide Copper and Oxide Gold, Plenge 9281-9402, October 2012.

The first sedimentation testwork programme is of interest because settling area requirements were determined with and without flocculant addition for three flotation products from each composite. The

[8] The sample details can be found in Table 13-11

flocculant used was Magnafloc 351 (with a consumption rate of about 22 g/t on average) for the tailings and MT-8834 for the concentrates.

The results listed in Table 13-17 suggest that thickening of the cleaner scavenger tails could be problematic, requiring large settlement areas and producing low underflow densities. Settling of the concentrate and production of a clear supernatant liquor proved impossible during this work.

Table 13-17 Copper concentrate and flotation tailings thickener area requirements, with and without flocculant

Sample	Rougher Flotation Tails		Cleaner Scavenger Tails		Bulk Cu-Mo Concentrate	
	Underflow % solids	Settling Area $m^2/(t/d)$	Underflow % solids	Settling Area $m^2/(t/d)$	Underflow % solids	Settling Area $m^2/(t/d)$
Supergene Composite						
With Flocculant	52	0.08	49	0.33	46	0.26
No Flocculant	60	0.51	46	1.32	-	-
Primary Composite						
With Flocculant	54	0.08	30	0.48	45	0.18
No Flocculant	58	0.35	32.4	1.92	-	-

Settling tests were conducted in process water (though this is not defined), with a grind size of 80% passing 150 µm for the tailings and 80% minus 30 µm for the concentrate. Magnafloc 351 was used as the flocculant, with a consumption rate of about 24 g/t for the tailings and 27 g/t for the concentrate. The settling requirements were defined as listed in Table 13-18 and are comparable with the previous testwork results for the flotation tailings, although the copper concentrate settling area requirements are lower by a factor of 2 to 3.

It appeared difficult to obtain high underflow densities on the flotation tailings while achieving acceptable overflow solution clarities.

Table 13-18 Copper concentrate and flotation tailings thickener area requirements

Sample	Flotation Tailings		Copper Concentrate	
	Underflow % solids	Settling Area $m^2/(t/d)$	Underflow % solids	Settling Area $m^2/(t/d)$
Supergene 1	52	0.08	63	0.06
Supergene 2	54	0.05	63	0.04
Primary 1	50	0.06	51	0.10
Primary 2	50	0.08	54	0.06
Average	52	0.07	58	0.07

Filtration testwork on the copper concentrates gave the results listed in Table 13-19.

Table 13-19 Copper concentrate filtration area requirements

Sample	Feed Slurry % Solids	Filter Cake % Moisture	Filtration Rate $m^2/(t/h)$
Supergene 1	50	7	1.12
	60	8	0.95
Supergene 2	50	5	1.12
	60	5	0.81
Primary 1	51	7	1.11
	62	7	0.95
Primary 2	50	8	1.68
	50	11	1.35
Average	52	7	1.14

13.3 Testwork undertaken by FQM

During 2019, four diamond holes were drilled to provide ten composite metallurgical samples for a new round of flotation testwork. These samples were sent to the ALS laboratories in Kamloops, British Columbia, Canada, along with brine and brackish water samples sourced from the Salar de Arizaro (brine), Valle de Arizaro (brackish) and Valle de las Burras (brackish). As previously, the samples were intended to represent plant feed from the initial years of starter pit mining, although this time with the testwork to be undertaken in brine solutions rather than in Lima tap water.

The testwork was undertaken in 2019 and 2020.

13.3.1 Testwork sample details

Figure 13-7 to Figure 13-10 show the location of the drillholes from which the new testwork samples were selected.

The ten composite samples are described in Table 13-20, whilst the proportions of acid soluble Cu, cyanide soluble Cu and insoluble Cu are plotted in Figure 13-11.

Figure 13-7 Plan view of sample drillhole locations

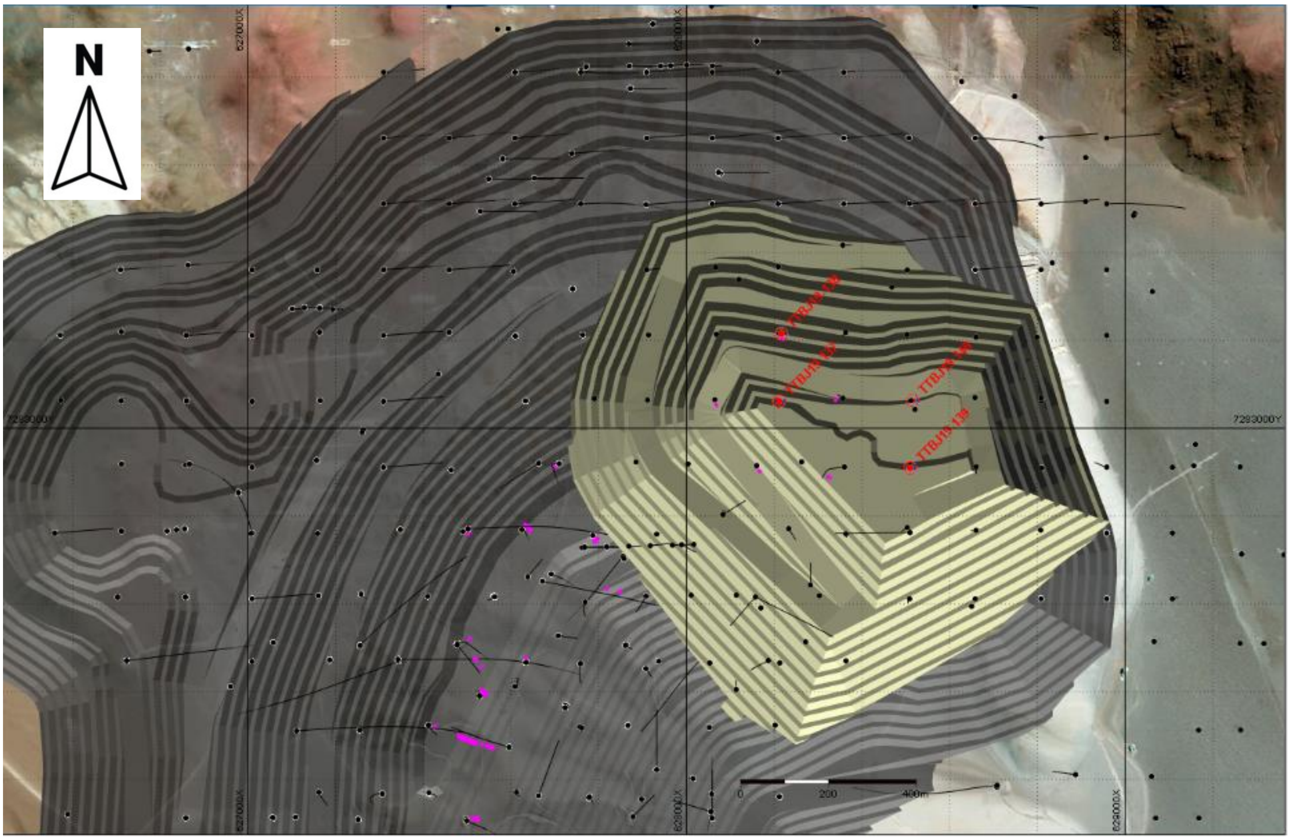


Figure 13-8 Drillhole TTBJ19-138 cross-section

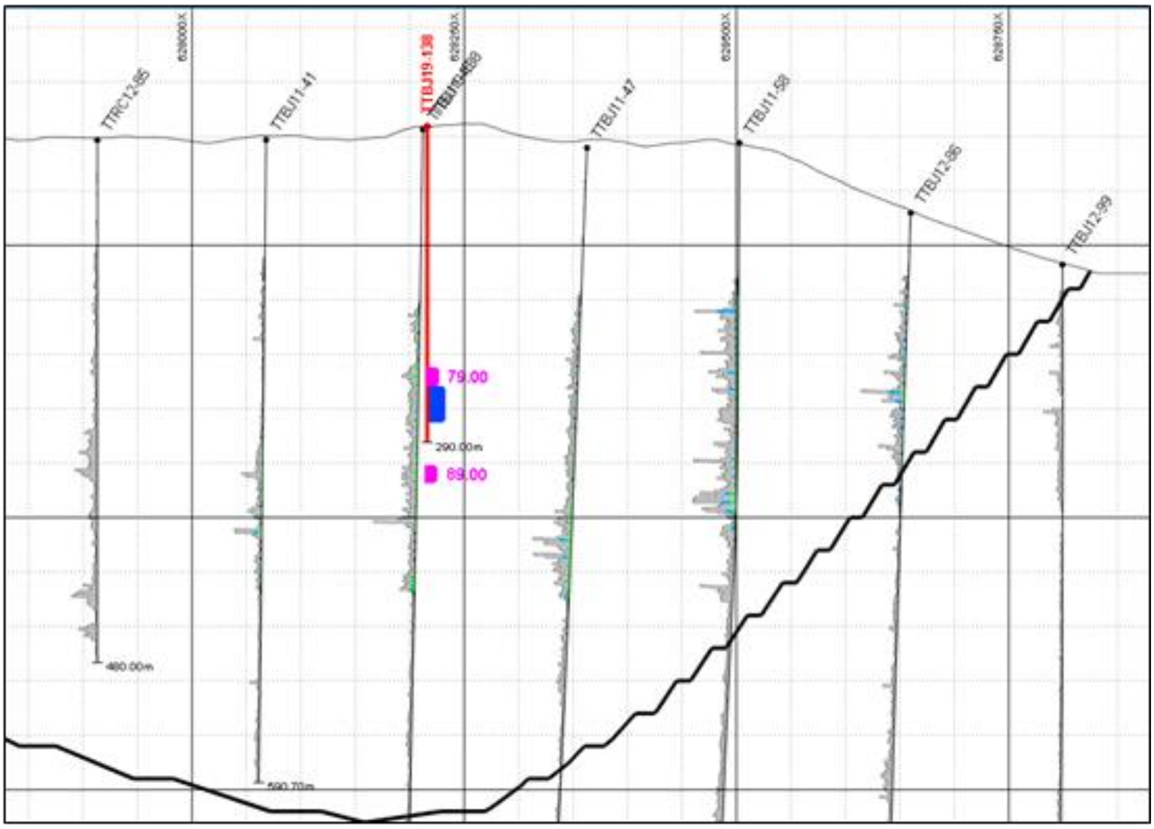


Figure 13-9 Drillholes TTBJ19-137 and 140 cross-section

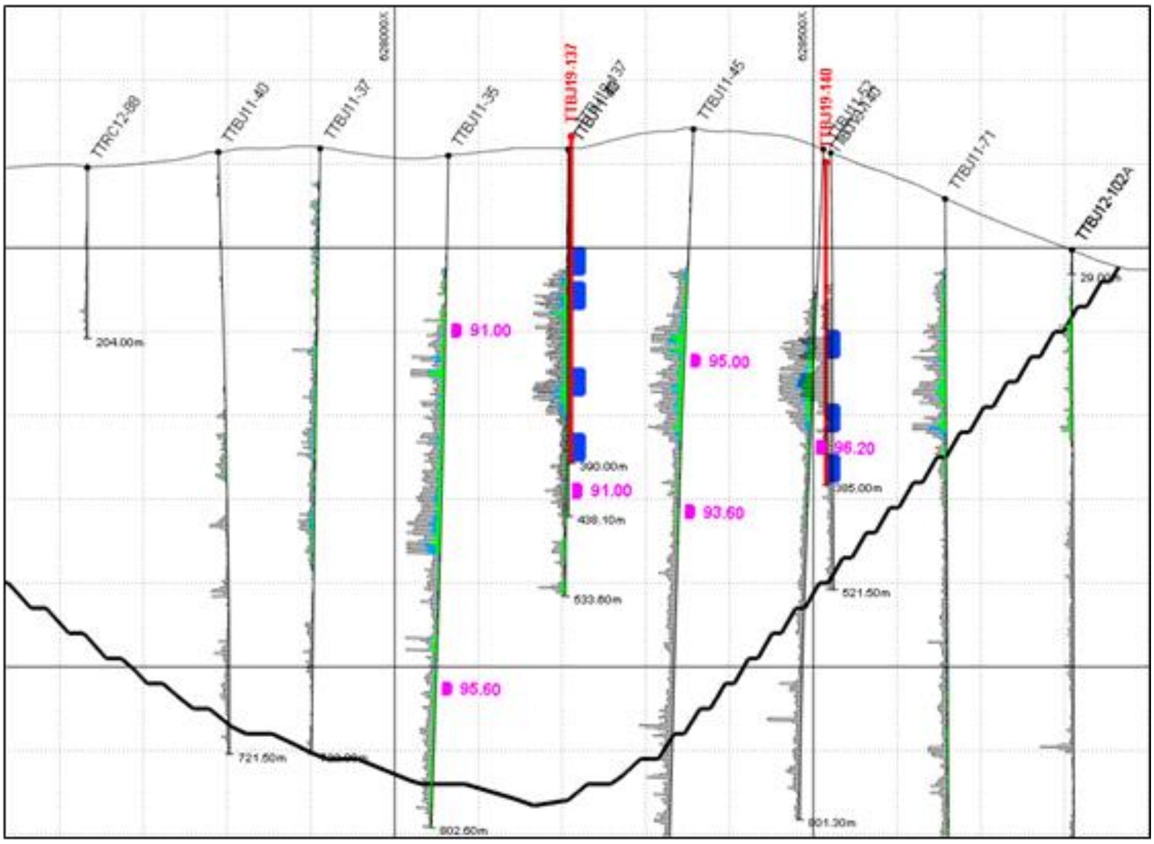


Figure 13-10 Drillhole TTBJ19-139 cross-section

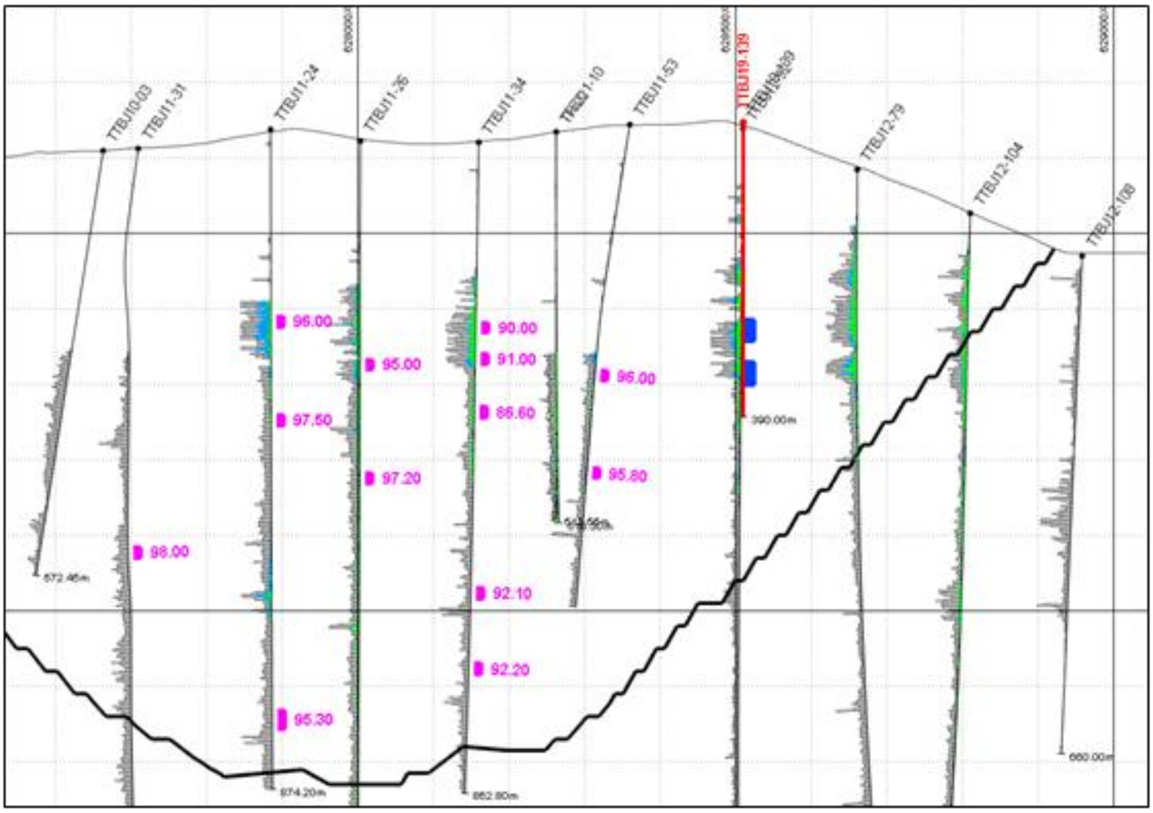


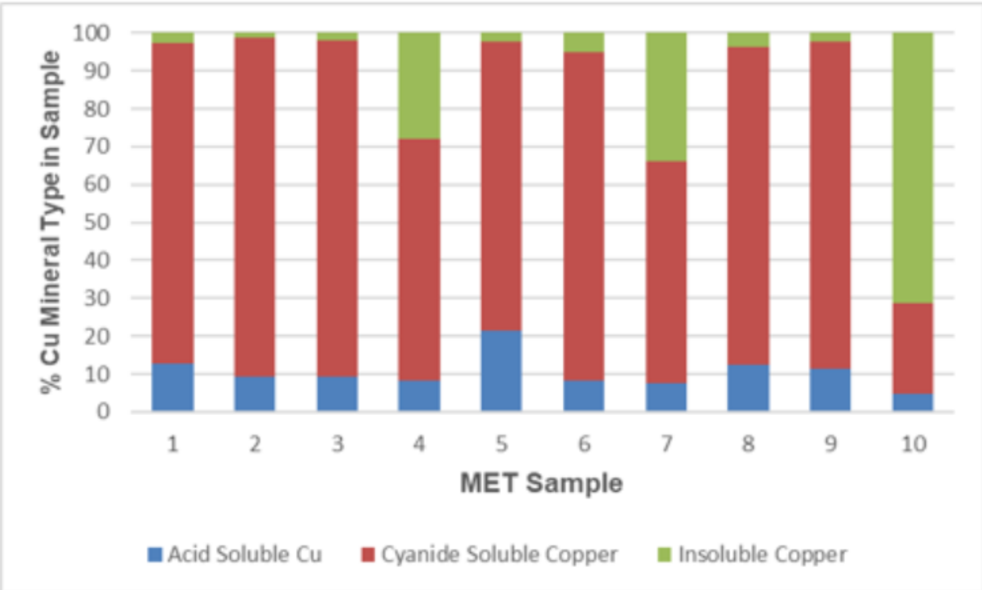
Table 13-20 Sample details for the ALS testwork programme

Sample	Intercept, m		Hole	Cu Domain	Pyrite	Lithology	Cu Assay, % Cu					Element Analyses				S/Cu
	Start	End					Assay	Total	CuOx	CuCN	CuRes	Fe, %	S, %	Mo, %	Au, ppm	Ratio
MET 001	134	164	137	CC > Sol + PL	Low	Granite	Ave	0.68	0.07	0.59	0.02	1.4	2.90	0.011	0.14	4.3
MET 002	175	205		CC > Sol	Low - Med	Granite	Ave	1.75	0.14	1.45	0.16	1.8	3.01	0.020	0.30	1.7
MET 003	278	308		CC > Sol	Med	Granite	Ave	1.64	0.15	1.34	0.15	1.5	2.63	0.021	0.20	1.6
MET 004	356	386		CC + FR	Med - High	Biotite - Granite	Ave	0.67	0.05	0.44	0.18	1.8	2.61	0.016	0.07	3.9
MET 005	240	270	138	CC > Sol	High	Granite	Ave	0.42	0.07	0.34	0.01	2.5	4.55	0.004	0.02	10.8
MET 006	261	290	139	CC > Sol	Low - Med	Granite	Ave	1.48	0.10	1.33	0.05	1.1	2.16	0.020	0.30	1.5
MET 007	317	348		CC + FR	Low - Med	Biotite - Granite	Ave	0.45	0.03	0.25	0.17	1.4	2.06	0.012	0.15	4.6
MET 008	202	232	140	CC > Sol + PL	Low - Med	Granite	Ave	0.89	0.09	0.76	0.04	1.9	3.67	0.012	0.11	4.1
MET 009	290	320		CC > Sol	Low - Med	Granite	Ave	0.96	0.10	0.86	0.00	2.0	2.66	0.016	0.09	2.8
MET 010	350	380		CC + FR	Low	Biotite - Granite	Ave	0.34	0.01	0.09	0.24	2.1	2.91	0.017	0.15	8.6
Arithmetic Averages of FQM & ALS Head Grades								0.93	0.08	0.75	0.10	1.75	2.92	0.015	0.153	

PL = Partial Leach - just below the leach cap, with visual iron staining - may have elevated levels of iron oxides
Sol = acid soluble Cu
CC = chalcocite
FR = Fresh sulphides - chalcopyrite

Note the high sulphur to copper ratio in sample MET-005. This sample proved problematic throughout the testwork programme, giving low copper recoveries, and poor concentrate grades under all conditions tested.

Figure 13-11 Proportion of copper species in the ten metallurgical samples from 2019



Each of these composite samples was subject to comminution testwork, plus rougher and cleaner flotation testing in tap water, to establish base line conditions. These tests were then followed by batch cleaner testwork using brine from site in the rougher flotation, dewatering of rougher concentrates, then regrind and re-dilution with brackish water from site prior to cleaner flotation.

The best conditions established from these tests were used for locked cycle testwork; final concentrates from these tests were analysed by ICP (inductively coupled plasma) to provide a full elemental breakdown for each composite.

Bulk flotation tests were also carried out to provide samples (rougher tailings, rougher concentrates, cleaner scavenger tailings, and final concentrates) for dewatering testwork conducted by Outotec (Metso).

13.3.2 Comminution testwork

The comminution testwork is reported in an SMC report 19017/P14, dated August 2019. Analysis of the results was carried out by Orway Mineral Consultants (OMC) and is reported in their report 8024-02 dated

October 2019. Sample data information and test results are listed in Table 13-21. Historic samples, tested in 2010 & 2012, are included in the table for comparison purposes.

Table 13-21 Comminution testwork data

Sample	Depth	SMC	RWi	BWi	Ai	SG
	m	Axb	kWh/t	kWh/t	g	
Historic Samples (2010 & 2012)						
Supergene	270-277		15.52	20.36	0.15	2.70
	300-306		15.34	18.63	0.18	2.72
	400-407		14.24	16.81	0.28	2.69
	430-438		12.81	15.08	0.19	2.69
Composite and Averages		**68.8**	**14.48**	**17.72**	**0.20**	**2.70**
Primary	316-322		12.83	16.29	0.22	2.79
	350-356		12.69	15.58	0.23	2.69
	430-436		13.91	16.60	0.19	2.71
	454-464		13.11	14.94	0.25	2.69
	472-482		14.53	16.08	0.23	2.69
	538-544		14.29	14.99	0.26	2.69
Composite and Averages		**60.9**	**13.56**	**15.75**	**0.23**	**2.71**
2019 Samples						
MET001	134-164	45.1	14.7	16.4	0.203	2.62
MET002	175-205	38.8	14.0	14.7	0.179	2.63
MET003	278-308	43.0	14.8	16.2	0.200	2.62
MET004	356-386	50.4	13.7	14.7	0.158	2.65
MET005	240-270	44.0	14.4	16.3	0.200	2.69
MET006	261-290	38.8	15.1	16.3	0.165	2.60
MET007	317-348	47.0	12.6	14.7	0.177	2.58
MET008	202-232	42.6	15.2	16.8	0.170	2.64
MET009	290-320	50.6	13.5	15.6	0.161	2.63
MET010	350-380	48.9	12.1	13.9	0.152	2.64
Average		**44.9**	**14.0**	**15.6**	**0.177**	**2.63**

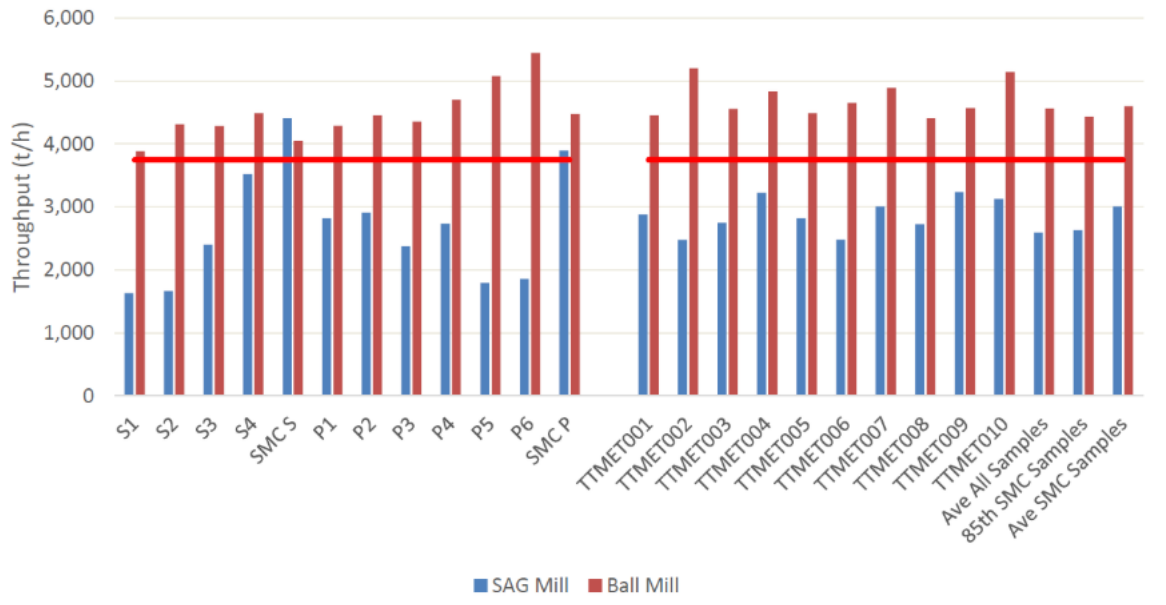
Table 13-21 shows that there is good agreement between the two sets of data for the rod and ball work indices (RWi & BWi), but the 2019 samples appear to have a lower abrasion index than the historic samples. The ore competency test results (SMC Axb) indicate the 2019 samples to be significantly tougher than the earlier samples. However, the earlier Axb results of more than 60 (indicating a soft ore) did not agree with the SAG design data, which indicate significant SAG mill power requirements.

The three samples highlighted in blue in Table 13-21 are biotite granite samples. Three out of the four highest Axb results, and three out of the four lowest RWi, BWi and Ai results are for this material, indicating it to be softer and less tough than the granites above it in the deposit.

An initial design of two milling trains, each comprising 1 x 28 MW SAG mill, and 2 x 22 MW ball mills was modelled by OMC, who indicated an average throughput of about 2,970 tph per train, or 47.6 Mtpa for two trains in an SABC configuration (i.e., with pebble crushing).

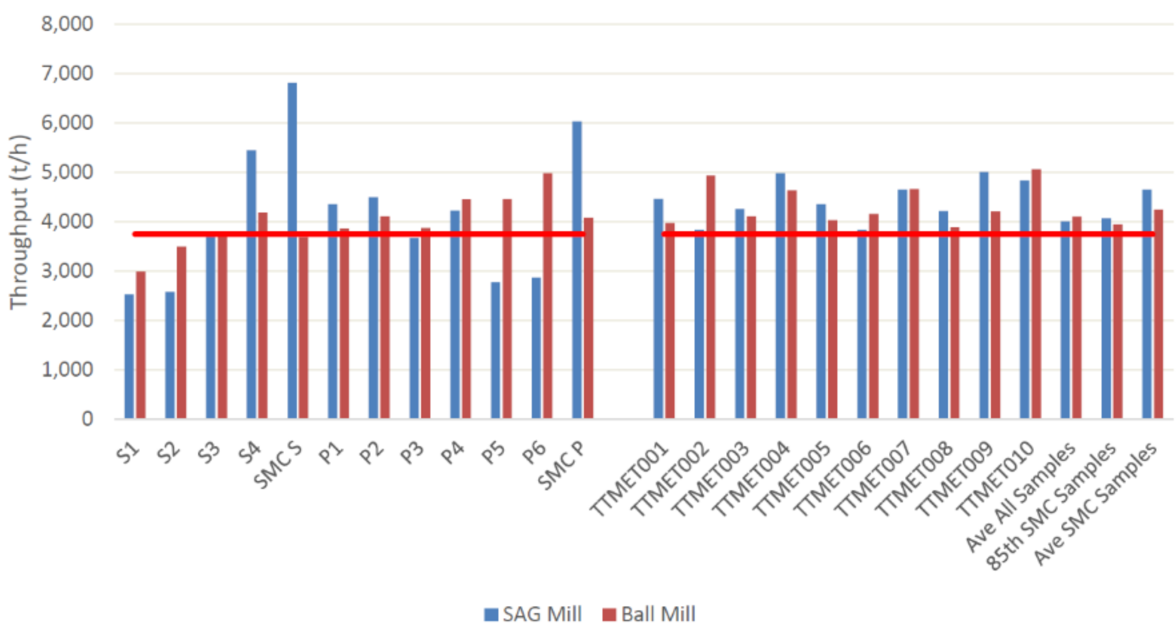
The throughput predictions from the individual samples, and the original target 3,750 tph (per milling train) are graphed in Figure 13-12.

Figure 13-12 Mill circuit throughput predictions for SABC circuit at 60 Mtpa



As can be seen from Figure 13-12, the proposed circuit is severely SAG mill limited, except in relation to the two historic SMC samples with high Axb values. A secondary crushing circuit was therefore examined to determine if the reduction in feed size would be sufficient to increase the SAG mill throughput and utilise the excess power in the ball mills. The secondary crush SABC throughput predictions for all samples are shown in Figure 13-13.

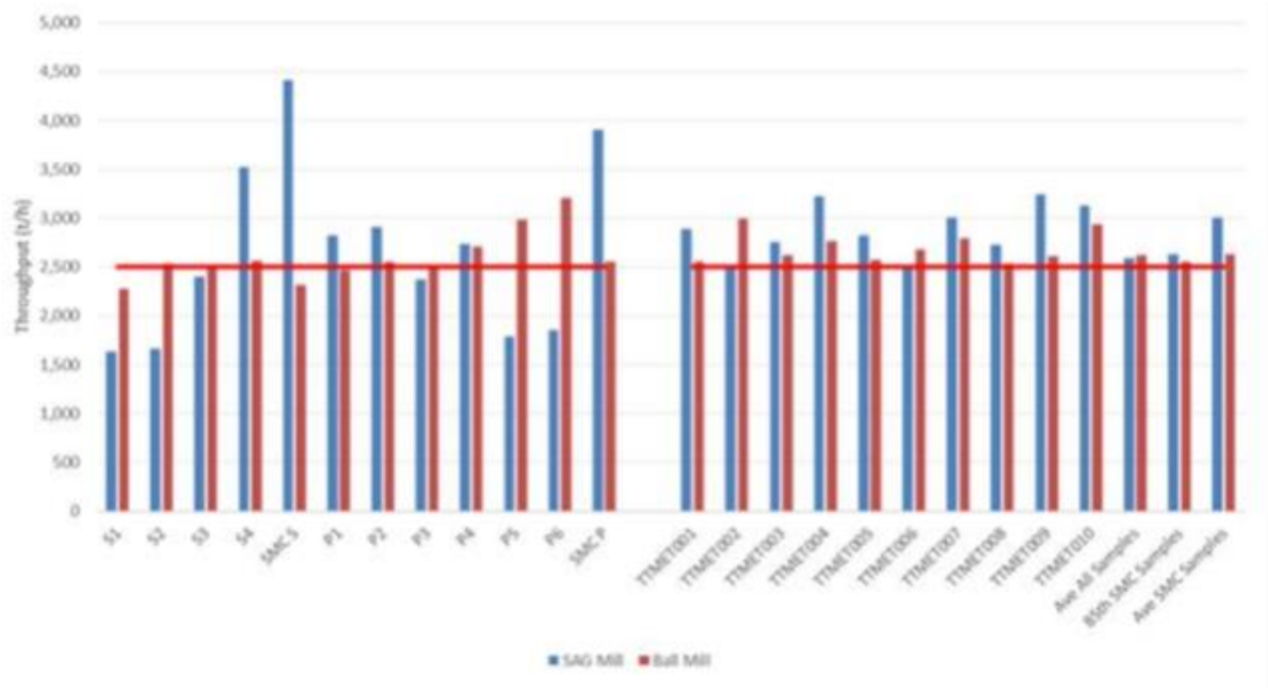
Figure 13-13 Mill circuit throughput predictions for a secondary crush SABC circuit



Using the 85th percentile on the SMC samples, the mill throughput is projected to be a maximum of about 8,000 tph through two milling trains, with 24 MW of pinion power required at the SAG mills and 18.5 MW at each of the ball mills. Achieving 60 Mtpa would require approximately 90% of the mill feed to be secondary crushed. For ease of layout and operability, the process design should allow for full secondary crushing of the mill feed.

A repeat of Figure 13-12 for a throughput of 40 Mtpa, or 2,500 tph per milling train, is presented in Figure 13-14.

Figure 13-14 Mill circuit throughput predictions for SABC circuit at a throughput of 40 Mtpa through two milling trains



A throughput of 2,500 tph is thus achievable through a SAG and ball mill combination, except for four historic samples. At a final target grind size of 80% passing 180 µm, a BWi of 19 (design number) and a P_{80} size of 1.0 mm, the ball mill power requirements would be 20.4 MW per milling train.

13.3.3 Primary and concentrate regrind size requirements

The 2013 Scoping Study (Ausenco, 2013) reported the optimum primary grind to be 80% passing 150 µm. This grind size was initially used in the testwork at ALS, but coarser grinds were subsequently tested during the initial rougher flotation tests in tap water, as undertaken to optimise flotation conditions. Rougher recoveries, concentrate grades and mass pulls appeared to be insensitive to the primary grind as it was increased to 180 µm and then to 212 µm. For sample MET 010, a primary grind of 80% passing 230 µm was found to be suitable.

The early work suggested an optimum regrind size of 80% passing 30 µm. It appeared that concentrates were very sensitive to the regrind size, e.g. at 45 µm, the copper concentrate grade was only 22.6% Cu. However, at finer regrind sizes, gold recovery drops off, but molybdenum recovery and grades in the bulk concentrate were improved.

The 2019 to 2020 testwork programme evaluated the regrind size required to give the best trade-off of concentrate grades and copper recovery. This work was performed in tap water initially, followed by batch cleaning tests using brine for the rougher flotation. The optimum conditions in terms of primary grind and concentrates regrind size were used in the locked cycle testwork and are shown in Table 13-22, tabulated against borehole and sample depth.

Table 13-22 Optimum primary grind and concentrate regrind sizes for each sample

Bore Hole	Sample	Material	Sample Depth, m		Grind Size, µm	
			Start	Finish	Primary	Regrind
137	MET 001	Partial Leach	134	164	192	23
	MET 002		175	205	216	24
	MET 003		278	308	182	25
	MET 004	Biotite Granite	356	386	186	20
138	MET 005		240	270	206	17
139	MET 006		261	290	222	30
	MET 007	Biotite Granite	317	348	216	19
140	MET 008	Partial Leach	202	232	202	18
	MET 009		290	320	213	18
	MET 010	Biotite Granite	350	380	231	17

The data does not suggest any trend regarding grind size requirements with material type or sample depth. On the basis of the above data, a primary grind of 80% passing 180 µm, and a regrind size of 80% passing 20 µm was selected for plant design.

13.3.4 High Pressure Grinding Rolls (HPGR) option review

In September 2024, OMC reviewed an HPGR-ball mill circuit option compared to the previous Primary Crush SABC circuit modelled in 2019. However, a revised total throughput rate of 40 Mtpa (total) was targeted at a float feed grind P_{80} size of 150 µm. Various staged HPGR-ball mill configurations were presented and compared to the revised Primary Crush SABC circuit consisting of two trains, each comprising a 28 MW SAG mill and a single 22 MW ball mill. The HPGR option model predicted a lower power consumption of ~16% and lower steel media consumption of 32% compared to the revised SABC option.

Detailed capital cost estimates for the two options were not conducted, although the HPGR option capital is expected to be 20% higher than for the SABC option. Given the ore competency is considered marginal for HPGR use, and the remote logistics to support HPGR roller maintenance, the SABC circuit was selected, representing a known technology within similar mills being employed within the FQM group.

13.3.5 Recommended comminution circuit

On the basis of the above testwork data and analysis, it is recommended that for a design throughput of 40 Mtpa (5,000 tph) for Stage 1, the milling circuit should comprise two trains each of a 28 MW SAG mill and a 22 MW ball mill. No secondary crushing will be installed.

For the throughput increase in Stage 2 to 60 Mtpa is contemplated, there are two options available to increase mill throughout;

- add a third milling train of one SAG and one ball mill, or
- add a secondary crushing circuit and a second ball mill per milling train.

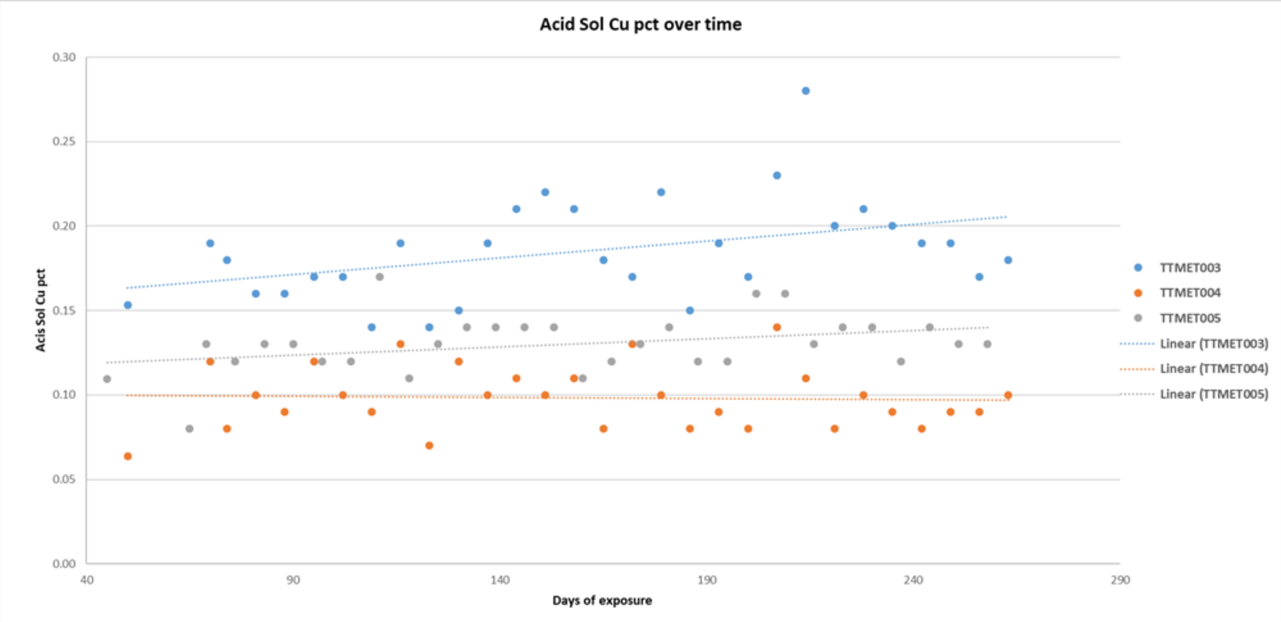
13.3.6 Oxidation testwork

An inspection of drill core in 2018 showed fine green rimming of chalcocite particles, suggesting rapid oxidation of the core. It was thought that this might explain the acid soluble copper assays in the samples sent for testwork, despite the absence of oxide minerals in the mineralogical investigations.

This observation was tested using the MET003 sample composite, which was crushed, spread out on a tray and left uncovered. A subsample was taken on a regular basis, and analysed for acid soluble copper, cyanide soluble Cu and total copper. Initially, samples were taken daily, then weekly and finally monthly. No trend towards increased acid soluble content and reduced cyanide content could be ascertained from this work.

A parallel programme at the Company's core storage facilities in Salta suggested a very weak trend towards increasing acid soluble and decreasing CN soluble copper (Figure 13-15). There was no greening (i.e., oxidation) of the samples as they were exposed to multiple rainfall events.

Figure 13-15 Oxidation testwork - increase in acid soluble copper with time



This testwork indicates that stockpiling of ore (e.g. for longer than six months) may lead to minor oxidation of the secondary sulphide minerals. However, the climatic differences between the Project site (i.e., drier) and Salta imply that the impact of oxidation should be negligible in normal plant operations through the crushing, crushed ore stockpiling and milling timeframe.

13.3.7 Flotation testwork at ALS

The testwork performed at ALS was designed to 'prove up' a process for the recovery of copper concentrates through flotation in brine solutions extracted from the salars in the vicinity of the Project. Since the completion of this work, a re-definition of the Project has led to a throughput reduction from 60 to 40 Mtpa and a process operating purely in fresh or brackish water supplied from borefields.

The emphasis of the testwork was therefore on defining conditions for flotation in brine, which are no longer entirely relevant to the Project. However, some baseline testwork was conducted in fresh or brackish water and these results are compared with the results achieved in brine and can be used as a basis for process design.

All composite samples, except sample MET003 were tested individually, initially using tap water in both rougher and cleaner flotation tests. Once baseline conditions had been defined, flotation work progressed using brine and process water for rougher flotation, followed by dewatering, regrind, and re-dilution with site brackish water for cleaner flotation.

The use of process water (water recycled from filtration of rougher tails from the previous test) was included in the testwork programme to simulate real plant conditions where rougher flotation tailings would be thickened prior to disposal, and with the thickener overflow water recycled to the milling circuit.

Numerous batch cleaner flotation tests were run to establish the best operating conditions, after which a single locked cycle test was run on each composite (including sample MET 003). A master composite sample comprising equal parts of all the ten individual composites was prepared and subject to a bulk flotation test (batch cleaning) to produce samples for dewatering testwork on rougher tailings, cleaner scavenger tailings, rougher concentrates and cleaner concentrates.

Water quality analyses

Analyses of the brine and brackish water used for the flotation testwork were undertaken. For the sake of brevity, the results shown in Table 13-23 exclude all of the dissolved metals for where the assay was below the detection limit.

Brine solutions were obtained from the Salar de Arizaro, about 3 km east of the Project site. The brackish water used in the testwork was a 1 to 1 blend of water from Valle de Arizaro and Valle de las Burras, in an attempt to replicate a blended supply to the plant.

The brine has a TDS of 324,000 ppm, which is mostly chlorides (177,000 ppm) and sodium (131,000 ppm). The rougher flotation tailings water was also analysed; most of this water would be returned to the plant as process water. This solution exhibited a much lower TDS of 182,000 ppm and the indications were that sodium and chloride were precipitated from solution in the flotation circuit.

An attempt was made to promote salt precipitation using lime addition. The brine TDS was reduced from 324,000 ppm to 288,000 ppm by the addition of 1 kg of lime per m^3 of solution. The lower TDS of 182,000 ppm measured in the rougher flotation tails water could not be replicated by the addition of lime, and thus no further work on ‘pre-treating’ the brine was undertaken.

Rougher flotation testwork in freshwater

The first phases of flotation work at ALS, conducted in tap water, were aimed at finding the optimum conditions for these samples.

A grind of about 80% passing 150 µm was targeted initially (based on the previous Lumina testwork), lime was added to bring the pH up to approximately 8.5 in the first test on each sample, followed by a pH of over 10 on the second set of tests. A low addition of PAX (potassium amyl xanthate, 3 to 14 g/t) was used, which together with the pH was aimed at minimising pyrite recovery to the rougher flotation concentrate.

Copper recoveries were high, averaging 94.2% over nine samples (sample MET 003 was excluded). Only two samples (out of nineteen tests) gave copper recoveries of less than 90%, and eleven results were over 95%. A repeat test on one of the poorer performing samples gave a recovery of 93%. Gold recoveries ranged from 67% to 88%, Mo recoveries from 53% to 94%, and mass recoveries from 6.3% to 11.4%. In four out of the nine samples sulphide sulphur recovery was lowered by operating the rougher flotation at the higher pH of over 10, indicating some depression of pyrite.

A third set of rougher tests was performed at coarser grind sizes, ranging from 80% passing 172 µm to 216 µm without any meaningful change in recoveries or mass pull. A coarser grind was used in all subsequent testwork, i.e. typically 186 µm to 216 µm, but as coarse as 231 µm for sample MET010.

The full set of results are presented in Figure 13-16.

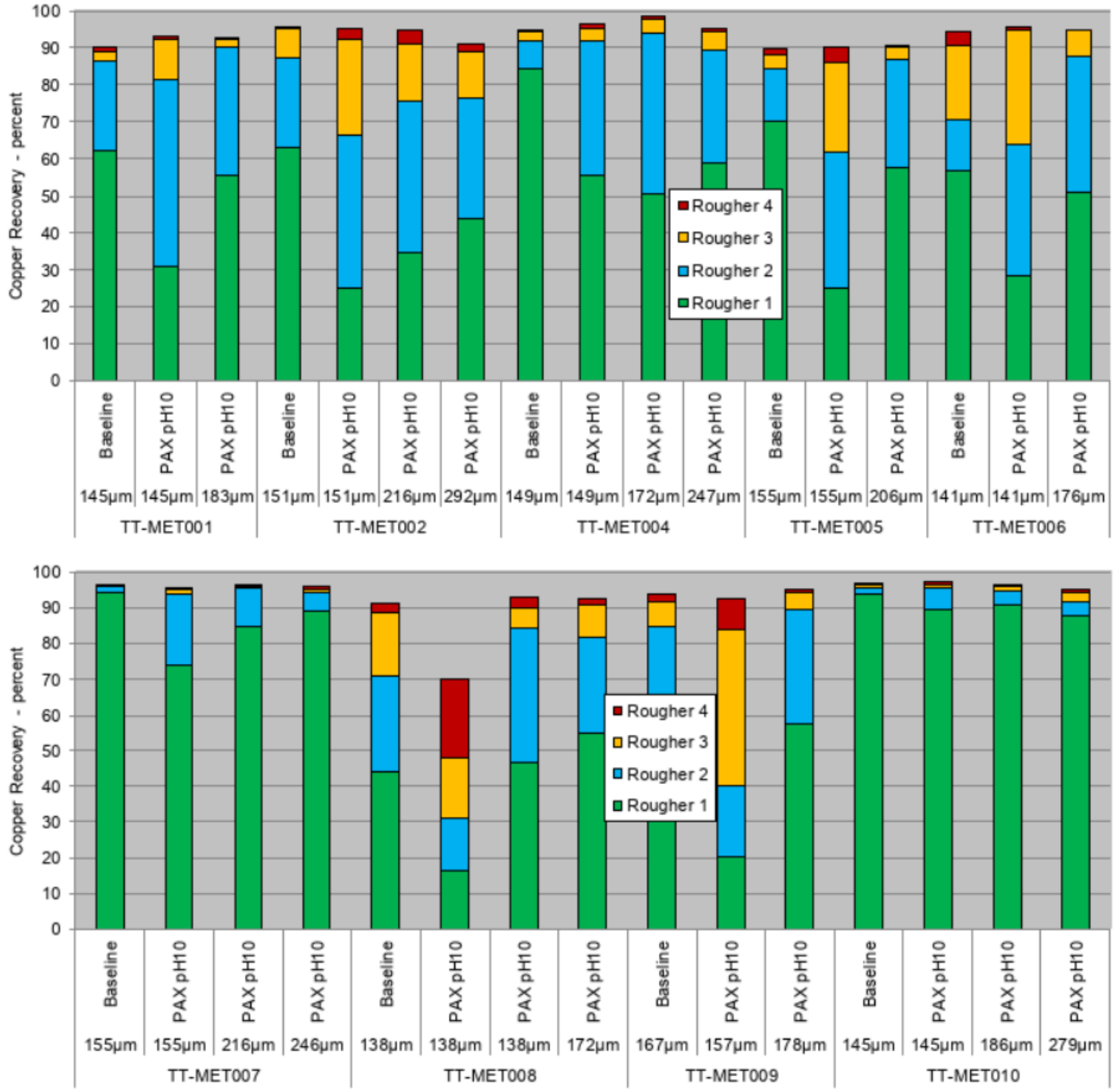
These results reconfirm the selection of 80% passing 180 µm for the primary grind size.

Similar charts for molybdenum and gold recovery at various grind size are provided in the ALS testwork report. The average recoveries from the total of 42 rougher tests in tap water were 93.7% for Cu, 77.4% for Mo and 78.8% for Au. The average mass pull was 9.2%.

Table 13-23 Analyses of water used in the testwork

Parameter	Lowest Detection Limit	Units	Valle de Arizaro	Vega de las Burras	Pit Water (Lumina)	Salar de Arizaro Brine	Rougher Flot Tailings Water	Sea Water (for Comparison)
Physical Tests								
Conductivity	2.0	µS/cm		1,600	241,700	216,000	196,000	50,000
Hardness (as $CaCO_3$)	63	mg/L	648	500		4540	3650	
pH	0.10	pH	7.8	7.1	7.05	7.55	7.55	7.5-8.4
Total Suspended Solids	3.0	mg/L				26.3	7.3	
Total Dissolved Solids	800	mg/L	2,412	1,040	317,596	324,000	182,000	35,000
SG						1.22		1.024
Anions and Nutrients								
Alkalinity, Bicarbonate (as $CaCO_3$)	1.0	mg/L		248		84.9	66.6	100
Alkalinity, Carbonate (as $CaCO_3$)	1.0	mg/L				<1.0	<1.0	
Alkalinity, Hydroxide (as $CaCO_3$)	1.0	mg/L				<1.0	<1.0	
Alkalinity, Total (as $CaCO_3$)	1.0	mg/L		129		84.9	66.6	
Chloride (Cl)	500	mg/L	1,010	168		177,000	122,000	19,900
Fluoride (F)	20	mg/L	0.6	<0.5		<20	<20	
Sulfate (SO_4)	300	mg/L	145	322		12,100	9,080	2,700
Nitrate (NO_3)		mg/L	14.8	5.2				
Organic / Inorganic Carbon								
Total Organic Carbon	0.50	mg/L				1.83	6.65	
Aggregate Organics								
COD	400	mg/L				15,300	8,950	
Dissolved Metals								
Barium (Ba)	0.050	mg/L				0.066	0.072	
Boron (B)	5.0	mg/L	1.7	1		38.7	26.6	
Cadmium (Cd)	0.0025	mg/L				<0.0025	0.0124	
Calcium (Ca)	25	mg/L		185		534	541	400
Copper (Cu)	0.10	mg/L				<0.10	0.31	
Iron (Fe)	5.0	mg/L		0.16		<5.0	<5.0	
Lead (Pb)	0.025	mg/L				<0.025	0.091	
Lithium (Li)	0.50	mg/L	0.17	0.13		76.0	54.4	
Magnesium (Mg)	2.5	mg/L		14.8		778	558	1,300
Manganese (Mn)	0.050	mg/L		0.09		1.07	1.19	
Molybdenum (Mo)	0.025	mg/L				0.806	0.532	
Potassium (K)	25	mg/L		9.1		3,480	2,540	400
Selenium (Se)	0.025	mg/L				0.096	0.048	
Silver (Ag)	0.0050	mg/L				<0.0050	0.0106	
Sodium (Na)	25	mg/L		119		131,000	86,100	11,000
Strontium (Sr)	0.10	mg/L		2.7		13.6	9.52	
Sulfur (S)	250	mg/L				4,620	3,690	
Thallium (Tl)	0.0050	mg/L				0.0202	<0.0050	
Zinc (Zn)		mg/L	0.35	4.02				

Figure 13-16 Rougher flotation recoveries in fresh water vs grind size (source ALS Testwork Report)



Rougher flotation testwork in brine

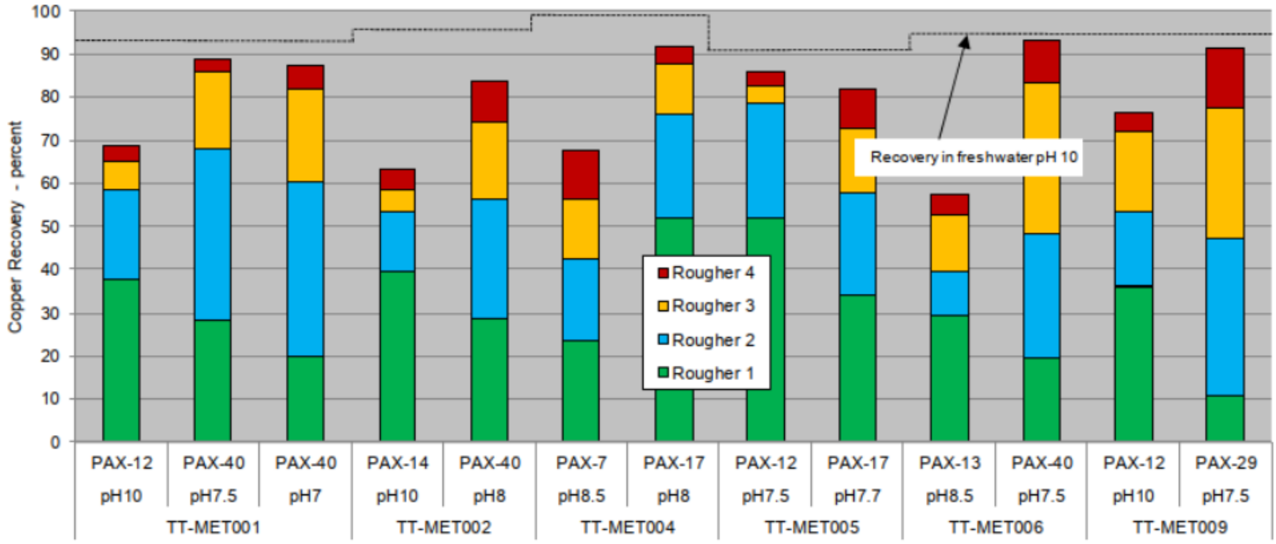
The first rougher tests using brine solutions from site required the addition of over 3 kg/t of lime to obtain a pH of 10. In all cases mass pull increased, and in seven of the samples, copper recoveries dropped significantly to between 76% and 50%. Three samples appeared to be less affected by brine flotation, i.e. sample MET004 (85.4% recovery), sample MET007 (94.8%) and sample MET010 (95%). All of these recoveries, however, were lower than the recoveries achieved in tap water. The three 'best performing' samples were the biotite - granite samples. Lower rougher recoveries to bulk concentrates were associated with composites that had higher sulphate mineral contents.

Subsequent tests using between 200 and 500 g/t lime addition achieved a pH in the roughers of between 7.5 and 8.0 and improved recoveries. Average recoveries across the ten samples were 87.2 % for Cu, 70.8 % for Mo and 80.6 % for Au with a mass pull of 11.2%.

Further testwork at natural pH (no lime addition in the roughers) improved rougher recoveries slightly and had a positive effect on cleaner flotation results. The average recoveries from 41 rougher tests in brine, with no lime addition, were 89.1% for Cu, 72.7% for Mo and 75.2% for Au. However, the mass pull increased to 16.1%.

Previous testwork had been conducted using low levels of collector (3 to 14g/t) in an attempt to reduce pyrite recovery to the bulk concentrate. Repeat tests (in brine) were conducted at PAX dosages of 17 to 40 g/t, which resulted in higher copper and gold recoveries in most cases. Sulphide sulphur recoveries to the rougher concentrates also increased. The results for copper recovery are presented in Figure 13-17.

Figure 13-17 Effect of collector dosage on copper recoveries for rougher flotation in brine (source: ALS testwork report)



The chart also shows the copper recoveries achieved in flotation in freshwater at a pH of 10. In all cases, flotation in freshwater gave higher recoveries.

Additional rougher flotation testing in brine was conducted to determine the effect of 3418A collector and polyglycol frothers on recoveries and mass pulls. Typically, recoveries dropped when using 3418A, even at high dosages of collector. Polyglycol frothers (W31 and H57) gave similar copper recoveries to MIBC, but at much higher mass pulls.

Cleaner flotation testwork in tap water

A single batch cleaner test was performed on each composite sample, using tap water in both the rougher and the cleaner flotation stages. The test conditions developed during the rougher flotation testwork were used, with a nominal primary grind size of 80% passing 200 µm, and a regrind size of 35µm. The nine samples (excluding MET 003) gave an average recovery of 85.5% Cu at a concentrate grade of 33.7% Cu. Sample MET005 (0.43% Cu with high pyrite) proved to be a difficult sample to deal with, and flotation tests under a variety of conditions consistently produced disappointing results from a high of 67.0% Cu recovery to a low of 23.4%. Figure 13-18 summarises the results.

Cleaner flotation testwork using brine in the rougher flotation, and in tap water in cleaner flotation

The first batch cleaner tests using brine in the rougher float (followed by freshwater in the cleaner float) were run with a lime addition of 200 g/t to achieve a pH of greater than 10 in the roughers. This led to high mass pull, and reduced recoveries to a rougher concentrate. Cleaner testwork on these rougher concentrates gave average recoveries of only 67% Cu (47% Mo & 55% Au) with a concentrate grade of 34.7% Cu.

Operating the rougher circuit without lime addition, but with high lime addition in the cleaner flotation tests to achieve a pH of greater than 10 for pyrite depression, gave average recoveries to a third cleaner concentrate of 77.9% Cu at a grade of 36.6% Cu. Relaxing the concentrate grade by running only a two stage cleaning circuit could increase recoveries to 82% at a grade of 29.9% Cu. Sample MET005 is included in these results; excluding the results from this sample would increase the average recoveries by about 2%.

Figure 13-18 Copper recoveries and concentrate grades for rougher and batch cleaner flotation in fresh water (Source: ALS testwork report)

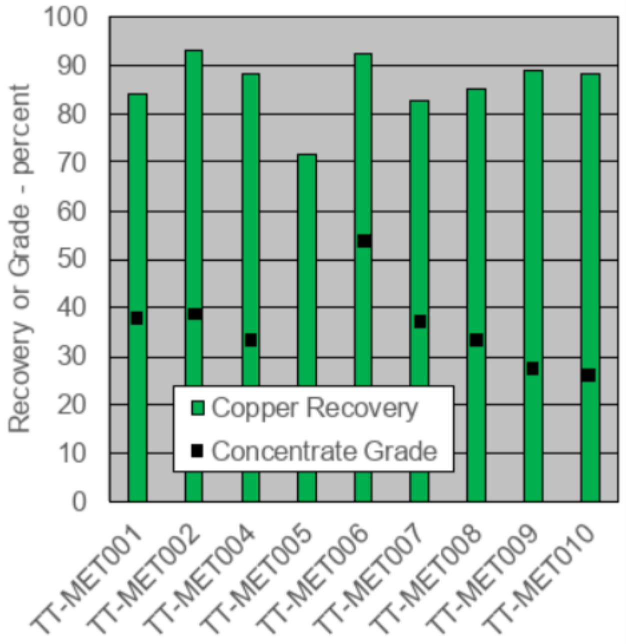
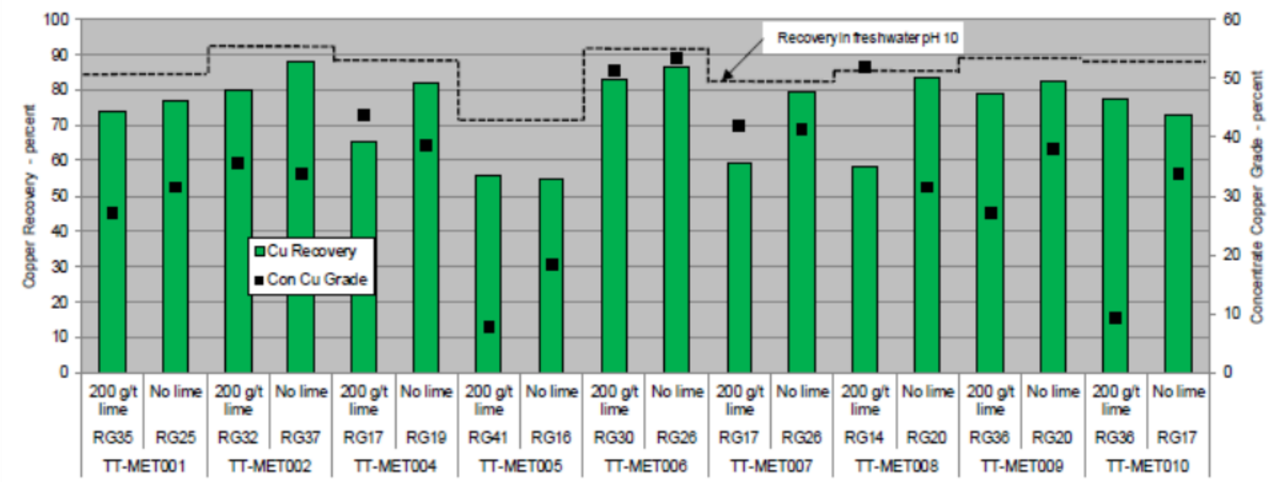


Figure 13-19 presents the results of these tests. The recoveries for rougher and cleaner flotation conducted in freshwater are also shown for comparison.

Figure 13-19 Copper recoveries and concentrate grades for rougher flotation in brine and cleaner flotation in freshwater (source: ALS testwork report)



Locked cycle testwork

A single locked-cycle test was completed for each of the composites using a mixture of process water (recycled water from the dewatering of rougher concentrates) and brine in the roughers, and brackish water in the cleaners.

The testwork protocols used previously involved dewatering the rougher flotation concentrate by filtration, achieving a filter cake of about 15% moisture content. This was diluted with brackish water from site for regrind and cleaner flotation at about 25% solids. This resulted in a slurry feed to cleaner flotation with the water being a combination of only about 6% brine and 94% brackish water.

At a plant scale, the rougher flotation concentrate would be thickened to 60% solids, prior to being re-diluted with brackish water. This would result in a water composition in the cleaner circuit of about 22% brine and 78% brackish water.

The plant water circuit was replicated in the locked cycle testwork by adding some of the filtrate from the rougher concentrate filtration back into the reground concentrates, together with brackish water as dilution for cleaner flotation in order to simulate a 55% thickener underflow slurry.

Locked cycle tests were run for five cycles, with the average results for cycles 4 and 5 being reported.

Average recovery results from the ten composites were 80.3% Cu, 50.6% Mo, and 58% Au at a concentrate grade of 36.7% Cu. Excluding sample MET005, average Cu recoveries were 82.1% at 39.7% grade. Recoveries could be increased to about 85% at a 30% Cu grade with less intensive cleaning.

The locked cycle testing results are presented in Table 13-24. The samples have been re-ordered such that samples with similar copper domains and lithologies are presented together:

- Samples MET 001 and 008 are classified as chalcocite dominant, with partially leached material being present.
- Samples MET 002, 006, 009 are chalcocite dominant with low to medium pyrite content.
- Sample MET 003 is from the same domain, but with medium pyrite content, whilst sample MET 005 has a high pyrite content (i.e., head assays of 2.95% Fe and 4.75% S).
- Samples MET 004, 007, and 010 are classed as chalcocite with higher levels of chalcopyrite present. The lithology of these samples is biotite – granite.

Table 13-24 Locked cycle test results

Sample	Grind, P_{80} µm		Weight	Assay - percent or g/tonne					Distribution - percent				
	Primary	Regrind	%	Cu	Mo	Fe	S	Au	Cu	Mo	Fe	S	Au
MET 001	192	23	1.4	**35.1**	0.35	21.9	32.6	4.37	73.8	46.8	20.9	17.6	51.1
MET 008	202	18	1.9	**37.3**	0.29	21.7	35.5	2.82	80.3	52.4	21.6	19.6	59.0
MET 002	216	24	3.0	**44.1**	0.45	16.3	30.6	4.73	79.8	68.1	27.9	33.4	52.3
MET 006	222	30	2.7	**43.2**	0.319	12.7	23.5	6.20	88.3	42.0	27.1	30.6	46.7
MET 009	213	18	2.0	**39.7**	0.36	19.6	36.0	3.06	85.2	53.5	20.0	28.7	56.3
MET 003	182	25	3.2	**45.6**	0.51	15.1	29.0	4.45	88.7	81.1	32.0	38.5	74.8
MET 005	206	17	1.7	14.7	0.106	35.3	44.9	0.62	64.4	52.6	22.0	17.7	32.0
MET 004	186	20	1.8	**32.1**	0.546	23.1	32.4	2.85	90.5	63.4	20.2	25.1	78.0
MET 007	216	19	0.8	**40.0**	0.62	22.0	33.2	10.03	69.3	39.0	11.0	13.1	62.5
MET 010	231	17	1.0	**35.2**	0.15	26.2	34.0	9.39	82.7	7.2	10.1	10.3	67.6
	Arithmetic Averages		1.9	**36.7**	0.37	21.4	33.2	4.85	80.3	50.6	21.3	23.5	58.0
	Composite (Bulk Flot)		1.8	**43.0**	0.42	15.1	29.5	5.25	83.2	51.7	14.3	19.5	62.9

Disappointing copper recoveries were obtained from samples MET001, MET005, and MET007. MET001 was a partially leached sample, whilst sample MET005 proved to be a difficult sample to treat throughout the testwork owing to the high pyrite content. The results from sample MET007 were low when compared with all other results obtained from this sample, and a recovery of about 76% Cu was expected.

Molybdenum recoveries averaged 50.6% despite a poor (and unexplained) 7.2% for sample MET010. Ignoring this result, the range of recoveries to final concentrate was 39% to 81% at concentrate grades of 0.29% to 0.62% Mo. Molybdenum recovery in the bulk float was 51.7%. Gold recoveries averaged 58%, with one poor result of 32%. The bulk float achieved a gold recovery of 62.9%. Concentrate grades ranged from 0.62 g/t (for the sample achieving 32% recovery) to 10 g/t Au.

Results from a bulk flotation test are reported in Table 13-24 for comparison. A composite sample comprising 15 kg of each of the ten samples was subject to a bulk flotation test to provide samples for thickening testwork. This was a batch cleaner test and so would be expected to give lower recoveries and higher concentrate grades than the average results from the locked cycle testwork. A recovery of 83% Cu at a concentrate grade of 43% Cu, compared with 80.3% Cu recovery at 36.7% Cu from the locked cycle work is encouraging.

A summary of ‘best’ batch cleaner results in tap water and in process/brine/brackish waters are summarised and compared to lock cycle test results in process/brine/ brackish waters in Table 13-25. Results for the MET005 composite were removed from the averages and summarised separately at the bottom of the table, since its performance was significantly worse than the other composites.

It is immediately clear that copper, molybdenum and gold recoveries were higher when tap water was used in the roughers than when brine and/or process water was used. The open circuit cleaner recoveries would be expected to improve slightly in closed-circuit testing but concentrate grades may decrease.

Locked-cycle testing with brine and process water did not, on average, improve metallurgical performance over batch cleaner test results almost entirely because of the (unexpected) poor performance of MET-007 in locked-cycle testing.

MET-005 recorded poor performance regardless of the water type used.

Copper recovery variations with depth are listed in Table 13-26.

Table 13-25 Comparison of batch cleaner and locked cycle flotation test results

Sample	Test	Regrind	Rougher	Rougher PAX	Concentrate Assay, % or g/t			Distribution - %		
	#	Sizing - µm	pH	Dosage - g/t	Cu	Mo	Au	Cu	Mo	Au
Batch Cleaner Tests - Tap Water										
MET001	33	35	10	12	37.4	0.3	4.58	84.1	48.3	52
MET002	34	34	10	14	38.5	0.3	4.53	92.9	70.3	72.7
MET003B	No Test									
MET004	35	32	10	7	33	0.44	3.47	88.2	53	73.1
MET006	37	31	10	13	53.5	0.33	7.39	92.1	45.1	60.8
MET007	38	33	10	8	36.7	0.91	8.37	82.6	76.7	63.8
MET008	39	35	10	11	33	0.25	2.33	85.1	54.2	54.7
MET009	40	41	10	12	27.2	0.38	2.11	89	78	77.1
MET010	41	38	10	5	25.8	0.76	6.55	88.1	56.3	61.3
Average					**35.6**	**0.46**	**4.92**	**87.8**	**60.2**	**64.4**
Batch Cleaner Tests - process and/or brine water in roughers, brackish water in cleaners										
MET001	80	25	Natural	29	31.3	0.28	3.57	77	42.2	40.1
MET002	94	22	Natural	40	44.8	0.39	5.27	70.6	59.1	51.4
MET003B	98	23	Natural	40	41.9	0.46	3.63	86.5	76.5	62.1
MET004	81	19	Natural	25	38.5	0.53	3.25	81.9	43.7	48.1
MET006	83	26	Natural	40	53.4	0.31	5.83	86.6	38	47.9
MET007	84	26	Natural	25	41.2	1	7.44	79.2	67.8	45.7
MET008	85	20	Natural	40	31.3	0.25	3.75	83.7	54.6	71.5
MET009	86	20	Natural	40	37.8	0.46	2.65	82.2	68.6	71.1
MET010	93	18	Natural	20	35.7	0.59	7.26	83.4	27.7	49.7
Average					**39.5**	**0.47**	**4.74**	**81.2**	**53.1**	**54.2**
Locked Cycle Tests - process water and brine in roughers, brackish water in cleaners										
MET001	99	23	Natural	26	35.1	0.35	4.37	73.8	46.8	51.1
MET002	100	24	Natural	40	44.1	0.45	4.73	79.8	68.1	52.3
MET003B	101	25	Natural	40	45.6	0.51	4.45	88.7	81.1	74.8
MET004	102	20	Natural	25	32.1	0.55	2.85	90.5	63.4	78
MET006	104	30	Natural	40	43.2	0.32	6.2	88.3	42	46.7
MET007	108	19	Natural	25	40	0.62	10	69.3	39	62.5
MET008	109	18	Natural	40	37.3	0.29	2.82	80.3	52.4	59
MET009	110	18	Natural	40	39.7	0.36	3.06	85.2	53.5	56.3
MET010	111	17	Natural	20	35.2	0.15	9.39	82.7	7.2	67.6
Average					**39.1**	**0.4**	**5.32**	**82.1**	**50.4**	**60.9**
MET005 Results - Batch Cleaning in Tap Water, Brine in Roughers with Brackish water in Cleaners, and Locked Cycle										
Tap Water	42	18	10	12	27.1	0.25	1.32	62.5	60.2	46.2
Brine	82	16	Natural	12	18.3	0.15	0.90	55.0	48.1	31.0
Locked	103	17	Natural	12	14.7	0.11	0.62	64.4	52.6	32.0

Table 13-26 Copper recovery variation with depth

Sample	Material	Sample Depth, m		Cu Recov
		Start	Finish	%
MET 001	Partial Leach	134	164	73.8
MET 002		175	205	79.8
MET 008	Partial Leach	202	232	80.3
MET 005		240	270	64.4
MET 006		261	290	88.3
MET 003		278	308	88.7
MET 009		290	320	85.2
MET 007	Biotite Granite	317	348	69.3
MET 010	Biotite Granite	350	380	82.7
MET 004	Biotite Granite	356	386	90.5

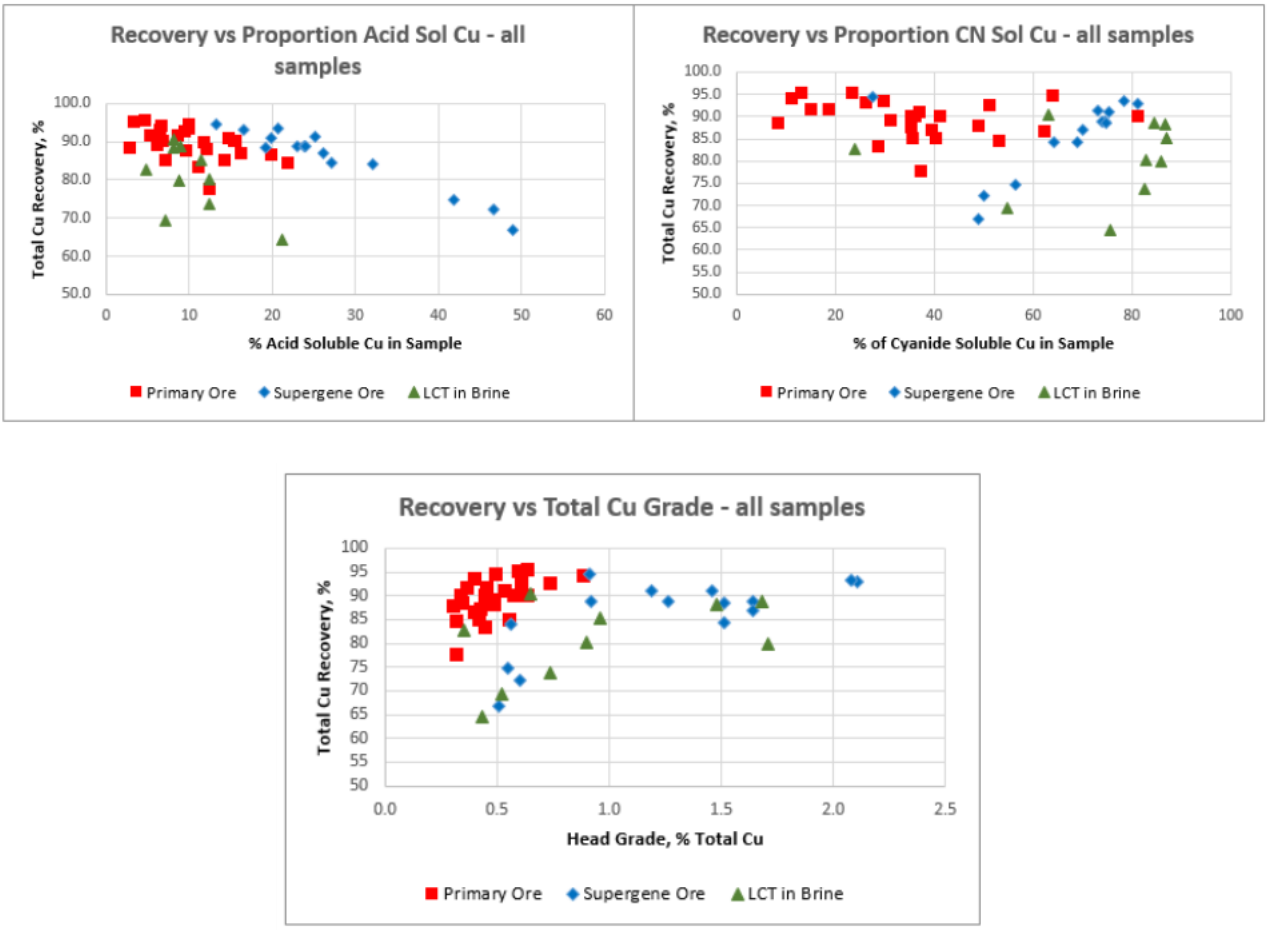
Bore Hole	Sample	Material	Sample Depth, m		Cu Recov
			Start	Finish	%
137	MET 001	Partial Leach	134	164	73.8
	MET 002		175	205	79.8
	MET 003		278	308	88.7
	MET 004	Biotite Granite	356	386	90.5
138	MET 005		240	270	64.4
139	MET 006		261	290	88.3
	MET 007	Biotite Granite	317	348	69.3
140	MET 008	Partial Leach	202	232	80.3
	MET 009		290	320	85.2
	MET 010	Biotite Granite	350	380	82.7

No distinct correlation of recovery with depth is evident from the information in Table 13-26, although with some exceptions there is a general trend of increased recovery with depth as the material type changes from partial leach material through the chalcocite dominant granites and into the biotite- granites. This is most evident in the recovery figures for borehole 137 samples. Unfortunately, there is insufficient data to confirm these trends.

No correlation appears to exist between concentrate grades and sample depth.

The results from the locked cycle tests were plotted on the same graphs presented earlier in Figure 13-20 and Figure 13-21, showing copper recovery as a function of acid soluble copper, cyanide soluble copper, and total copper in the feed.

Figure 13-20 Copper recovery from variability samples and recent locked cycle tests vs copper speciation



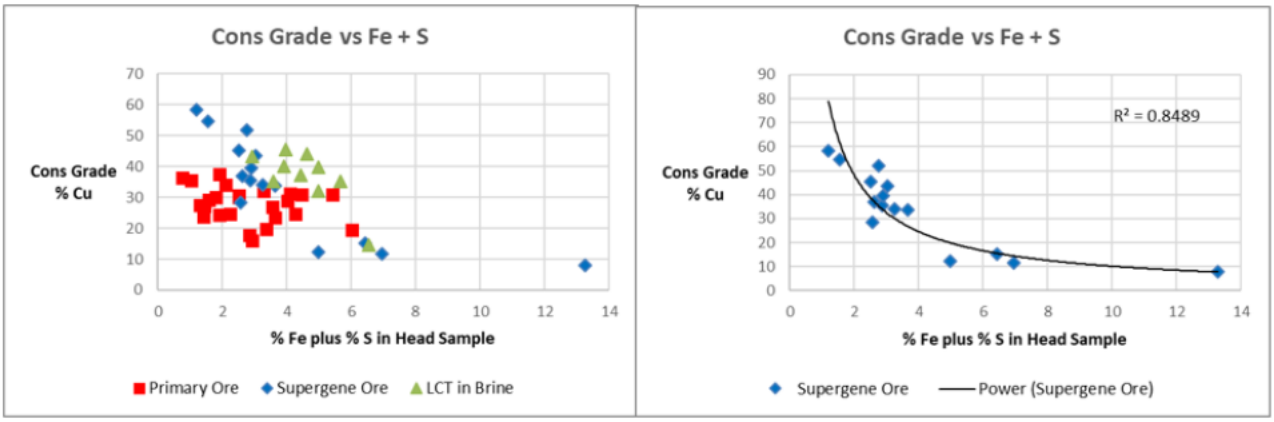
The graphs in Figure 13-20 illustrate the lower recoveries from the recent ten metallurgical samples compared with the variability samples tested in 2011. The variability sample results were obtained from batch tri-cleaner tests performed in tap water, whilst the results from the ten samples tested in 2019 to 2020 are from locked cycle tests using brine for rougher flotation, and site brackish water for cleaner flotation.

The graphs also show the much higher proportion of cyanide soluble copper in the latest samples when compared with the previous samples. In comparing the supergene ore samples tested previously (blue points) against the recent set of results (green points), it would appear that there are two different recovery curves. There is one for samples with cyanide soluble copper content ranging from about 50% to 80% of the total copper content, and one for material containing over 80% cyanide soluble copper.

The effect of pyrite content in the sample on concentrate grade is illustrated in Figure 13-21 (using a proxy of Fe plus S for pyrite). There appears to be little correlation between the two parameters for the primary ore samples from the variability testwork, or for the recent locked cycle testwork.

For the variability testwork on supergene samples, a strong relationship between concentrate grade and pyrite content in the head sample does appear to exist, with concentrate grades being decreased by high pyrite levels in the feed. No such trends occur for copper recovery versus Fe and S levels in the feed.

Figure 13-21 Copper concentrate grade variability samples and recent locked cycle tests vs pyrite content



13.3.8 Mineral liberation analyses

The rougher tailings and cleaner scavenger tails from each of the locked cycle tests were submitted for mineral liberation determination. Table 13-27 lists the tails assay grades for each sample and the percentage of total copper losses[94].

Copper sulphide liberation averaged about 54% which would be adequate for good rougher recovery in the roughers. Sample MET-005 had the lowest liberation of copper sulphides at 25%, with about half of the copper minerals in binary form with pyrite observed as rimming.

Table 13-27 Copper distribution in rougher tails by mineral species

Sample	Tail Assay	% of Total Copper Losses				
	% Cu	Cp	Bn	Ch/Cv	Ten/En	CuOx
MET001	0.10	9.3	0.6	90.0	0.1	
MET002	0.19	2.0	0.7	90.8	0.1	6.5
MET003	0.12	4.7	0.5	92.9	0.2	1.7
MET004	0.04	32.8	2.2	61.2	0.3	3.5
MET005	0.09	13.8	4.4	81.8	0.0	
MET006	0.13	8.7	0.8	90.6	0.0	
MET007	0.11	25.7	3.9	64.6	0.8	5.2
MET008	0.11	2.1	2.5	94.4	0.9	
MET009	0.10	2.6	3.0	94.4	0.0	
MET010	0.04	55.8	3.1	40.9	0.2	

Most copper losses in the rougher tails are carried by chalcocite; the only samples where chalcopyrite carries a high proportion of copper losses are in samples MET004, 007 and 010. These samples were classified as the biotite-granite material type.

[9] In Table 13-28 Cp = chalcopyrite, Bn = bornite, Ch/Cv is chalcocite and covellite, Tn/En is tennantite and enargite, and Cu Ox = oxide copper minerals.

Samples MET002 and 003 were high grade samples containing 0.15% acid soluble copper and this might explain the losses in oxide copper minerals in these particular samples. Samples MET004 and 007 contained low levels of acid soluble copper in the feed, so the oxide copper losses in this instance are unexplained.

Table 13-28 shows copper losses by size distribution; for all samples, approximately 30% of copper losses occur in the + 150 µm particle size range, whilst 40 to 50% occur in the -14 µm fines. These bimodal losses are typical of a rougher tail. Losses in the coarse size fractions were as binary particles associated with gangue minerals, and the greater proportion of losses in the -14 µm fraction were as liberated particles. However, there were instances of locking of chalcopyrite, and particularly for bornite with chalcocite in the finer size

Table 13-28 Copper distribution in rougher tails by size distribution

Sample	Cu Losses by Size Range				
	>150 µm	<150>75 µm	<75>25 µm	<25>14 µm	<14 µm
MET001	31.8	10.3	7.8	5.7	44.4
MET002	45.7	9.4	5.2	3.1	36.6
MET003	20.7	16.4	7.4	3.8	51.7
MET004	22.3	9.7	7.0	3.8	57.2
MET005	37.4	12.1	6.2	3.0	41.3
MET006	34.0	9.4	6.7	3.9	46.1
MET007	33.3	15.0	13.7	3.5	34.5
MET008	32.7	12.0	6.5	3.5	45.2
MET009	30.9	9.5	5.9	3.5	50.1
MET010	36.6	9.1	5.7	2.2	46.4

Table 13-29 Copper distribution in cleaner tails by mineral species

Sample	Tail Assay	% of Total Copper Losses			
	% Cu	Cp	Bn	Ch/Cv	Ten/En
MET001	0.70	1.8	0.5	97.7	0.0
MET002	1.14	1.5	2.1	96.4	0.0
MET003	0.67	4.5	1.6	93.8	0.1
MET004	0.19	24.0	2.1	73.9	0.0
MET005	0.43	7.1	4.3	88.6	0.0
MET006	0.35	2.5	1.5	95.1	0.9
MET007	0.39	35.7	1.1	62.6	0.7
MET008	0.49	1.5	1.7	96.7	0.1
MET009	0.38	0.9	0.7	98.3	0.0
MET010	0.27	53.9	6.9	38.7	0.4

As with the rougher tails, most copper losses are carried by chalcocite. The proportion of losses to chalcopyrite was lower than for the rougher tails, and the losses to bornite slightly higher. The only samples where chalcopyrite carried a high proportion of the copper losses were in samples MET004, 007 and 010; these samples were classified as the biotite granite material type.

The solids size distribution is much finer than in the rougher tails because of the regrind of rougher concentrate, hence only three size fractions were assessed. Most losses occurred as liberated particles in the <14 µm fraction (Table 13-30), indicating a need for improved flotation of ultra-fines. Some copper losses in the fines appear to be in binary particles with gangue minerals.

Table 13-30 Copper distribution in cleaner tails by size distribution

Sample	Cu Losses by Size Range				
	>150 µm	<150>75 µm	<75>25 µm	<25>14 µm	<14 µm
MET001	0.0	0.0	31.2	19.9	48.8
MET002	0.0	0.0	43.3	19.0	37.8
MET003	0.0	0.0	36.6	13.0	50.3
MET004	0.0	0.0	7.3	7.3	85.4
MET005	0.0	0.0	9.4	9.2	81.4
MET006	0.0	0.0	20.6	12.3	67.1
MET007	0.0	0.0	2.6	4.6	92.8
MET008	0.0	0.0	11.1	9.3	79.6
MET009	0.0	0.0	13.4	9.0	77.6
MET010	0.0	0.0	5.5	6.2	88.3

Losses of copper in the 25 µm to 75 µm range were mainly as chalcopyrite, bornite and chalcocite locked with pyrite, with some minor locking with gangue minerals. However, there was also some liberated copper sulphide particles lost in this size range, noticeably for sample MET002 which had a high tailings assay.

13.3.9 Concentrate analyses

Multi element scans of concentrates produced from the locked cycle testwork on supergene and primary composite samples and from the 2020 testwork at ALS on the ten metallurgical samples produced the results listed in Table 13-31.

The original testwork results were analysed by Plenge and in some cases the analytical detection limits for some elements differs to those reported by ALS. Nevertheless, the analytical data indicates that the concentrates contain low levels of penalty elements such as arsenic, bismuth, cadmium, antimony or mercury.

Chloride levels are elevated in the concentrates from the recent ten metallurgical samples, and this is probably because of residual brine solutions used in flotation. Washing of concentrates in low chloride water may be required to maintain low residual chlorides (<100 ppm) in the final product, prior to shipment.

The Mo content of the concentrate measured between 0.1 and 0.6%. Gold assayed between 3 and 10 g/t (except for MET-005), which exceeds the payable limit for smelters; silver levels ranged from 3 to 26 g/t and are likely to be below payable levels.

13.3.10 Sedimentation and filtration testwork

Samples of rougher tailings, cleaner scavenger tailings, rougher concentrate and cleaner concentrate (from a three stage batch cleaning test) were sent by ALS to Outotec for sedimentation testwork. These samples had been produced by a bulk flotation of a composite sample comprising 35 kg of each of the ten metallurgical samples tested by ALS in 2019 to 2020. The results were reported in:

1. Thickening test report Part A and Part B - Outotec, July 2020, and
2. Copper Con Filtration test Report - Outotec, July 2020.

The flotation testwork had been carried out in brine for rougher flotation, followed by brackish water for the cleaners. The primary grind was 80% passing 180 µm, and the regrind size for the copper concentrate was 80% passing 24 µm.

Table 13-31 Concentrate analyses

Element	Units	Original Testwork		2020 Metallurgical Testwork Samples									
		Supergene	Primary	MET 001	MET 002	MET 003	MET 004	MET 005	MET 006	MET 007	MET 008	MET 009	MET 010
Cl	ppm	<10	<10	540	330	310	<50	<50	450	320	350	470	670
F	ppm	<10	<10	180	180	170	210	140	310	<20	130	130	<20
Hg	ppm	0.1	0.1	<1	1	1	2	1	1	1	1	<1	1
Fe	%	24.4	25.2	21.9	16.3	15.1	23.1	35.3	12.7	22	21.7	19.6	26.2
S total	%	35.5	33.9	32.6	30.6	29.0	32.4	44.9	23.5	33.2	35.5	36	34
Au	ppm	1.90	5.40	4.37	4.73	4.45	2.85	0.62	6.20	10.00	2.82	3.06	9.39
Ag	ppm	11.17	17.83	22.0	20.1	14.6	15.3	3.1	16.7	17.6	14.1	5.8	25.7
Al	%	<0.01	<0.01	0.42	0.59	0.4	0.61	0.27	0.86	0.11	0.26	0.26	0.14
As	ppm	198	17	51	41	57	172	12	13	18	74	51	19
Ba	ppm	1	3	<50	<50	<50	<50	<50	60	<50	<50	<50	<50
Be	ppm	5	5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Bi	ppm	5	5	3.1	2.8	1.7	5.8	1.5	1.1	4.6	1.7	1.2	5.2
Ca	%	0.13	0.19	0.47	0.29	0.19	0.32	0.26	0.27	0.30	0.13	0.09	0.35
Cd	ppm	6	10	1.2	1.7	0.8	3.8	0.4	0.5	1.1	0.6	2	3.6
Ce	ppm			9.5	9.1	4.4	8.1	6.6	9.2	3.3	6.5	4.1	3.6
Co	ppm	56	56	174	139	108	118	160	75	32	136	134	14
Cr	ppm	311	640	240	280	100	170	140	190	50	70	80	30
Cs	ppm			<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Ga	ppm			1.9	1.6	1.3	2.5	0.6	5.7	0.7	0.8	0.8	0.8
Ge	ppm			0.8	0.8	0.8	1	<0.5	0.8	0.9	0.6	0.7	0.8
Hf	ppm			<1	<1	<1	1	1	<1	1	1	1	1
In	ppm			0.11	0.19	0.1	0.77	<0.05	0.07	1.75	0.08	0.06	2.37
K	%	0.04	0.05	0.29	0.31	0.25	0.49	0.12	1.05	0.05	0.1	0.11	0.08
La	ppm			<5	<5	<5	<5	<5	<5	<5	<5	<5	<5
Li	ppm			<2	<2	<2	2	<2	3	<2	<2	<2	<2
Mg	%	0	0	0.02	0.03	0.02	0.04	0.02	0.04	0.03	0.02	0.02	0.04
Mn	ppm	0	0	250	130	90	120	50	170	50	80	70	20
Mo	ppm			3,700	4,590	4,550	5,580	1,130	3,330	7,000	3,120	3,680	1,565
Na	%	0	0	0.05	0.03	0.03	0.09	<0.02	0.09	<0.02	0.03	0.03	<0.02
Nb	ppm			1	1	<1	1	<1	1	<1	<1	<1	<1
Ni	ppm	180	339	177	232	92	117	118	138	38	116	105	22
P	ppm	4	3	100	100	<100	100	<100	100	<100	100	<100	<100
Pb	ppm	100	100	37	12	8	22	9	10	30	9	32	69
Rb	ppm			4	5	5	10	3	17	1	2	2	2
Re	ppm			7	11.2	8.4	9.1	1.6	5.5	10.9	4.9	5.8	2.5
Sb	ppm	25	8	6	3.1	6.7	7.1	6.3	2.1	7.4	9	2.2	4.2
Sc	ppm	3	2	<1	1	<1	1	<1	4	<1	<1	<1	<1
Se	ppm			250	280	200	210	100	250	340	200	190	330
Sn	ppm	16	31	6	5	5	7	<2	7	5	2	2	5
Sr	ppm	20	32	45	65	23	25	18	97	13	60	14	4
Ta	ppm			<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Te	ppm			2.8	2.7	1.3	2.1	2.4	1.1	0.7	2.1	1.5	0.7
Th	ppm			4	3	2	4	3	5	2	2	2	<2
Ti	%	0	0	0.01	0.02	0.01	0.02	0.01	0.03	0.01	0.03	0.02	0.01
Tl	ppm			1.7	1.9	1.4	1.5	0.9	1.3	2.1	0.9	2.8	1.4
U	ppm			1	1	<1	1	<1	1	1	<1	1	<1
V	ppm	3	3	<5	6	<5	17	<5	17	<5	<5	<5	<5
W	ppm	24	37	3	3	1	4	1	5	1	1	1	1
Y	ppm	1	2	1	1	1	2	1	1	1	1	1	1
Zn	ppm	300	400	90	30	30	130	20	20	90	40	50	180
Zr	ppm	1	1	<5	<5	16	39	43	<5	43	25	35	47

Unfortunately, there was insufficient brine available to enable the rougher tails and concentrate samples to be tested at the correct thickener feed densities in brine. A synthetic solution was thus made up using NaCl, which is the main constituent of the locally available brine.

SNF flocculants were screened in static settling tests prior to the dynamic thickening tests. SNF 920 VHM, an anionic polyacrylamide, was found to be best for the scavenger tailings and final concentrate samples, whilst SNF 125 SH was preferred for use in highly saline environments.

The thickener sizings for a throughput of 40 Mtpa are listed in Table 13-32. Note that the bulk concentrate thickener sizing is based on the peak production years and not the LOM average.

Table 13-32 Thickener size recommendations (at 40 Mtpa)

Process Stream	Concentrate	Rougher Cons	Rougher Tails
Feed Density, % solids	15 - 25	25 - 35	25 - 35
Diluted Feedwell % solids	15	13	20
Slurry pH	11	7	7
Floc Dosage, g/t	10	80	30
Underflow density, % solids	68.8	47.6	67.2
Overflow clarity, mg/L	<150	415	<150
Solids loading, t/m^2/h	0.2	0.5	1.4
Feed rate, tph	150	750	5000
Required Thickener Area, m^2	750	1500	3571
Single Thickener Diam, m	31	44	67

These tests were repeated in 2021 for final tailings, using samples prepared in fresh water, with the following reports produced:

1. Thickening test report Part A and Part B – Metso:Outotec, September 2021
2. Tailings filtration test report – Metso:Outotec, September 2021

The results from the thickening tests are presented in Table 13-33.

Table 13-33 Thickening results from 2021 testwork

	Flocculant Type	Flocculant Dose	Flux Range	Underflow Density	Overflow Solids
		(g/t)	(t/m2h)	(% w/w)	(mg/L)
Final Tailings (HRT)	934 SH	10 - 30	0.6 - 1.00	64.0 - 65.7	44 - 86
Final Tailings (HCT)	934 SH	10 - 30	0.8	75.2 - 75.3	-

The testwork demonstrated that the final tailings could be successfully thickened in a high rate thickener to 60 to 65% solids at an average flux density of 0.8 t/m^2/h. In practice, for a throughput of 40 Mtpa, this would require two tailings thickeners at 63 m in diameter, or three at 50 m.

The results also indicated that increased tailings densities up to 75% solids could be achieved using high compression thickening at around the same flux rate. This would require a minimum of 4 x HCT's, as their maximum diameter is limited to 45 m. However, it was noted in the testwork that the underflow from the HCT tests could not be pumped, and the density reported is indicative only.

Filtration tests on the final tailings indicated that they could dewatered to less than 10% moisture, at a filtration rate of about 250 kg/m^2/h. This could be increased to about 380 kg/m^2/h, by reducing the time of the air drying stage – but at the expense of increased moisture content (to 12.2% moisture). At 380 kg/m^2/h, the filtration requirements for a throughput of 40 Mtpa would be approximately 13,200 m^2.

The rougher concentrate sample also proved difficult to thicken, with low settling rates of 0.5 t/m^2/h, and relatively low thickener underflow densities. However, the results for the 2020 and 2021 testwork are consistent.

The 2012 testwork gave thickener requirements for both copper concentrates and flotation tails of 0.07 m^2/(t/d), which equates to 0.6 t/m^2/h. The recent testwork gives a lower settling rate for the concentrate of 0.2 t/m^2/h , but this agrees with the data from the 2010 testwork of 0.16 to 0.23 t/m^2/h.

Outotec noted that a solids loading for a concentrate thickener of over 0.25 t/m^2/h was not recommended. The rougher concentrate thickener area requirements would therefore double to 6,000 m^2, giving a single thickener diameter of 87 m. Filtration tests on the final concentrate (2020 Testwork) were performed with a fixed feed density of 60% solids. The conclusions from this testwork were:

- cake release from cloth was good and the cloth was easy to wash
- cake moisture after washing was higher than during non-washing pressure filtration
- non-washing cake moistures ranged from 10.3% to 10.8% wt
- peak wash ratio was at 0.5 m^3/t and had an 84% efficiency for the removal of chloride
- cake moistures after washing ranged from 11.3% to 11.9% wt
- lower moisture can be achieved with higher drying pressures; further testing would be required using a larger test unit provided that enough sample quantity is available

With the wash stage, the average filtration rate over four tests was 424 $kg/m^2/h$; this is 2.35 $m^2/(t/h)$ and is considerably lower than the average of 1.14 $m^2/(t/h)$ reported in the earlier testwork. The reasons for this may be the finer regrind size used and the inclusion of a wash stage in the filtration test.

13.3.11 Rheology testwork

The sedimentation testwork described above noted that flotation tailings could be thickened to high densities, but the material was difficult to pump. Rheology work was therefore carried out in 2025 to examine the yield stress of the flotation tailings at various slurry densities and to determine the optimum densities for pumping of this material to the TSF.

The work was undertaken by Tailpro Consulting, Santiago, and is reported in Tailings Thickening and Deposition dated May 2025.

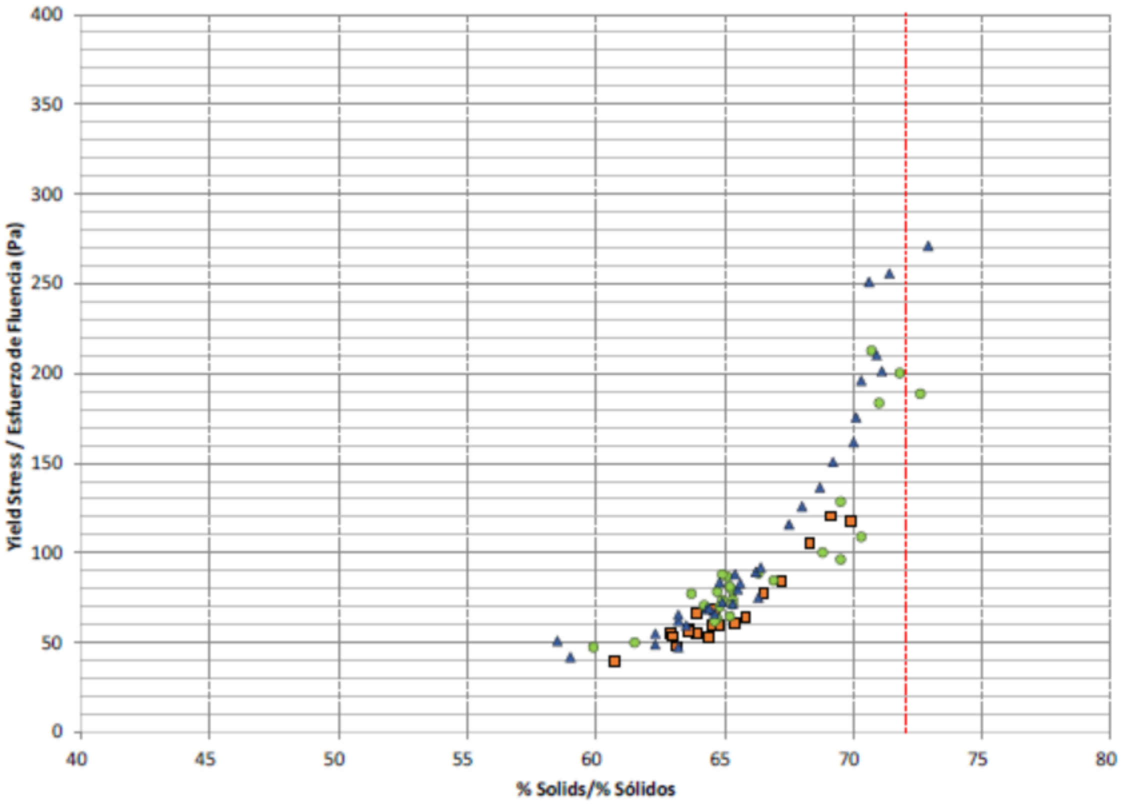
Tailpro concluded that the sample thickened very well, achieving their target underflow densities during continuous operational mode (65-72% solids) with relatively low bed heights and high flux rates in the semi-pilot dynamic unit. The overflow water clarity was less than 50 NTU in the semi-pilot unit tests that are expected to be improved on an industrial scale due to the feedwell to outer wall ratio variation.

The semi-pilot test work showed that the optimum solids flux rate for the material tested, was in the order of 0.7 to 0.8 $t/hr/m^2$. As a result, 4 x 45 m diameter high compression thickeners would be required to treat the expected process solids tonnage of 4,886 t/h at a flux rate of 0.77 $t/hr/m^2$.

Tailpro commented that even though underflow solids concentrations of up to 72% might be achieved periodically, based on the thickener sizing exercise, 70% solids would be more achievable on a continuous basis.

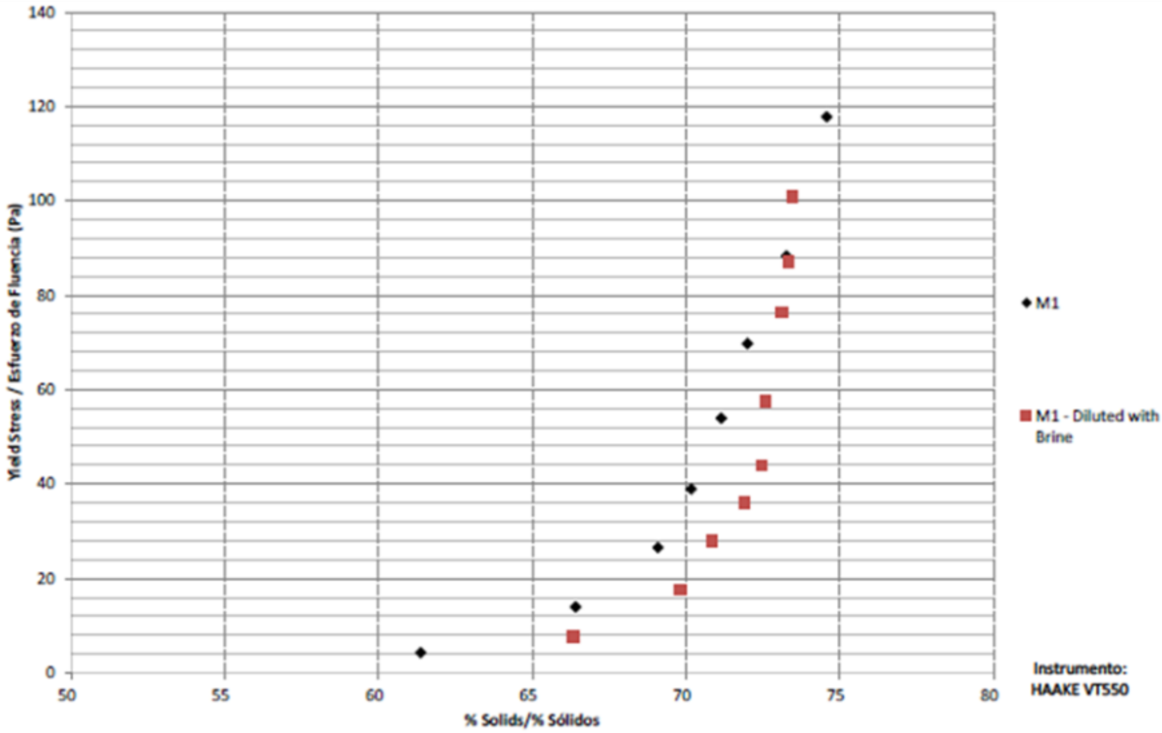
Figure 13-22 shows the unsheared yield stress for the sample tested, directly from the discharge of the semi-pilot dynamic thickener. This shows a yield stress of around 200 to 250 Pa for tailings with a slurry density of 72% solids.

Figure 13-22 Unsheared yield stress vs % solids of tailings sample



This plot is repeated for sheared tailings and presented in Figure 13-23 for thickened tailings and for tailings re-diluted with brine. This illustrates the rapid increase in yield stress above about 70% solids.

Figure 13-23 Yield stress vs % solids of tailings samples



The graph shows the rapid increase in yield stress above about 70% solids and also shows a reduction in yield stress for 70% solids from between 150 and 200 Pa for unsheared material to 20 to 40Pa for sheared tailings.

Re-diluting tailings to 62 to 65% solids using brine reduces the yield stress to below 10 Pa, which suggests this density would be suitable for distribution around the TSF through a spigot system.

Recommendations from this testwork are:

- Four HCT thickeners would be required for the design throughput of 40 Mtpa (confirming the previous work).
- The thickener underflow slurry density should be designed at 70% solids.
- Thickened slurries require shear thinning to reduce the yield stress to enable the use of centrifugal pumps.

The report by Tailpro includes estimates of the TSF beach angle for various re-diluted slurry densities, prepared by two independent consultants.

Dr McPhail of Water, Waste and Land Australia suggested that at 65% solids, the tailings would be sufficiently fluid that an average beach slope of only 0.15% would be achieved.

Prof. Andy Fourie of the University of Western Australia reported a beach slope of 0.97 to 1.88% depending on the number of discharge spigots used.

Both consultants confirmed that the tailings slurry exhibits a low yield stress at the range of slurry densities being considered (62 to 68%) and would be suitable for deposition on the TSF. Spigot spacing would be an important design aspect of the TSF, and for a beach length of 1,000 m, should not be less than 30m.

13.4 Gold leach testwork on near surface material

Three samples of near-surface oxidised material containing gold were tested by Plenge in March 2011 to determine the potential for gold recovery from the perched gold domain during near-surface mining of successive pit phases.

Core samples were selected from five different holes, at depths varying from 22 m to 300 m, and composited into high grade (0.48 g/t Au), low grade (0.16 g/t Au), and average grade (0.34 g/t Au) samples. The tests were run as bottle roll tests in tap water and in brine solutions from site. The tests were run for 96 hours at 40% solids with 0.10% cyanide (NaCN), and a pH of approximately 10.5.

The results from the bottle roll tests in tap water (Table 13-34) indicated an increase in recoveries with decreasing particle size, although accompanied by increasing cyanide and lime consumption. At a 19 mm crush size, gold recoveries were below 50%, suggesting that a heap leach operation may not be viable. A particle size distribution of 100% minus 10 mesh (1.7 mm) would be required to achieve recoveries of over 60%.

Table 13-34 Gold leach recoveries at different grind sizes

Size	Head Grade	Recovery	Consumption, kg/t	
	Au, ppm	% Au	NaCN	CaO
Low Grade Material				
100% - 3/4"	0.21	46.2	1.7	0.5
100% - 10#	0.21	62.9	2.8	2.3
95% - 200 #	0.19	89.3	5.1	5.0
Medium Grade Material				
100% - 3/4"	0.41	46.5	1.8	0.6
100% - 10#	0.29	67.5	2.7	2.3
95% - 200 #	0.30	72.4	3.3	6.7
High Grade Material				
100% - 3/4"	0.45	45.3	1.6	0.6
100% - 10#	0.49	71.9	2.4	2.0
95% - 200 #	0.47	85.5	2.6	4.2

Leaching in brine was limited to a pH of 10.5 due to the buffering effects of the brine. Lime consumptions of about 7 kg/t were recorded, and gold recoveries dropped by between 2% and 5%. In both tap water and brine bottle rolls, the presence of cyanide soluble copper caused high cyanide consumptions at finer crush sizes (320 ppm Cu measured in the pregnant leach solution).

Further work was carried out and reported by Plenge in 2012. Again, three samples at varying gold grades were bottle rolled in tap water for 96 hours, producing the results listed in Table 13-35.

Table 13-35 Gold leach recoveries vs grind size and feed grades

Sample	Size	Head Grade, g/t		Leach Recovery, %		Consumption, kg/t	
		Au	Ag	Au	Ag	NaCN	CaO
Low Grade	-1/2 "	0.29	0.8	59.5	22.4	0.33	0.58
Medium Grade	(-12.5 mm)	0.37	0.7	53.0	12.2	0.36	0.49
High Grade		0.53	1.2	53.8	46.5	0.37	0.50
Average		0.40	0.9	55.0	33.3	0.35	0.52
Low Grade	-1/4"	0.29	0.7	73.9	57.2	1.41	0.73
Medium Grade	(-6 mm)	0.37	0.6	63.4	35.1	0.74	0.71
High Grade		0.59	1.3	70.6	77.3	1.82	0.66
Average		0.42	0.87	69.3	56.5	1.32	0.70
Low Grade	100% - 10#	0.29	0.9	81.1	31.9	1.5	1.6
Medium Grade	(1.7mm)	0.35	0.8	80.1	20.8	1.1	1.3
High Grade		0.56	0.7	73.3	64.0	1.3	1.9
Average		0.40	0.8	78.2	38.9	1.3	1.6
Low Grade	80% - 200#	0.28	1.3	91.0	76.1	0.8	1.4
Medium Grade	(74 µm)	0.38	0.9	92.4	32.3	0.7	0.9
High Grade		0.52	1.5	91.9	80.4	0.8	0.9
Average		0.39	1.2	91.8	62.9	0.8	1.1

These results are an improvement of those presented in Table 13-34 but nevertheless a crush size of between 6 mm (1/4 inch) and 1.7 mm (10#) is still indicated as being necessary for reasonable gold recoveries. Silver recoveries follow a general trend of increasing recovery with finer crush size, but many of the results, particularly for the medium grade sample at all crush sizes appear anomalous. A comparison of results using tap water and brine is listed in Table 13-36 for the samples crushed to -1/2 inch.

Table 13-36 Effect of water quality on gold recoveries

Sample	Liquid	Head Grade, g/t		Leach Recovery, %		Consumption, kg/t	
		Au	Ag	Au	Ag	NaCN	CaO
Low Grade	Tap	0.29	0.8	59.5	22.4	0.33	0.58
Medium Grade		0.37	0.7	53.0	12.2	0.36	0.49
High Grade		0.53	1.2	53.8	46.5	0.37	0.50
Average		0.40	0.9	55.0	33.3	0.35	0.52
Low Grade	Brine	0.32	0.9	45.3	33.6	0.03	52.0
Medium Grade		0.35	0.8	36.5	55.2	0.05	57.3
High Grade		0.55	1.2	41.0	49.4	0.05	49.6
Average		0.41	0.97	40.6	44.8	0.05	53.0

Table 13-36 indicates that silver recovery improved using brine, whilst gold recovery dropped by approximately 15%. Cyanide consumption appears to drop to very low levels in the brine tests, but this is thought to be due to chloride interference in the free cyanide determination. Lime consumptions are moderate in tap water, but extremely high in brine due to the buffering effects of this solution. It appears very hard to reach the pH required to achieve protective alkalinity in saline solutions.

13.4.1 Gold recovery testwork, 2017

Ore samples containing relatively high gold grades were submitted to Plenge for gold leaching testwork in early 2017.

Bottle roll tests were conducted on six ground samples at a grind size of 80% passing 74 µm. The test conditions were: 72-hour bottle rolls in 0.1% NaCN (1 g/L) at 40% solids, and at a pH of between 10.6 and 11.2. The results are presented in Table 13-37.

Table 13-37 Gold recovery from bottle roll tests at 80% passing 75 µm (200#)

Sample	Head Grade, g/t		Leach Recovery, %		Consumption, kg/t	
	Au	Ag	Au	Ag	NaCN	CaO
TTBJ 12-90	0.540	0.3	95.5	35.9	0.4	2.2
TTBJ 12-105	0.747	1.5	97.9	79.7	0.3	2.0
TTBJ 12-103	0.367	0.3	95.1	38.8	0.3	4.9
TTBJ 11-07	0.651	1.2	93.4	74.4	0.5	2.0
TTBJ 11-12	0.470	1.4	92.2	77.8	0.6	2.2
TTBJ 11-22	0.788	0.3	95.6	28.6	0.7	1.9

Head grades varied between 0.37 g/t and 0.79 g/t Au. Gold recoveries on all samples were greater than 90% after 24 hours and ultimately reached between 92% and 97%. Cyanide consumptions were modest at 0.3 kg/t to 0.7 kg/t NaCN, whilst lime consumptions were relatively high at 2 kg/t to 5 kg/t CaO.

Bottle roll tests and column leach testwork were subsequently performed on three samples of coarser material (Table 13-38), i.e. the bottle rolls on minus 10 mesh (1.7 mm) material, and the columns on minus 19 mm (3/4 inch) samples. The column tests were run at a cyanide concentration of 500 ppm NaCN (0.5 g/L) and a solution application rate equivalent to 10 L/h/m^2 for 40 days. Cyanide consumptions were between 0.8 kg/t and 1.25 kg/t, and lime consumptions 4.9 kg/t to 5.6 kg/t.

Table 13-38 Gold recovery at different grind sizes

Sample	Au	Particle Size		
	Head Grade	80% - 74 µm	100% - 10 Mesh	100% - 19mm
	g/t	(bottle roll)	(bottle roll)	(column leach)
TTBJ 11-07	0.65	93.4	74.4	62.3
TTBJ 11-12	0.47	92.2	65.6	45.3
TTBJ 11-22	0.79	95.6	70.8	40.6

These results emphasise the effect of particle size on gold recovery, as noted previously. The different leach responses between the three samples at the coarser size fractions (but not at the finer sizes) is unexplained.

The leach kinetics for gold and silver were based on the volume of solution passed through the columns. After 40 days, the leach curves had flattened, with an increase in recovery of only about 2% between days 30 and 40.

It should be noted that the minimum particle size that is practical for a heap leach operation is about 6 mm (1/4 inch). 10 mesh material (1.7 mm) may be treatable in a vat leach but would require 'de-sliming' with the fines treated in an agitated tank leach. A 6 mm heap leach operation would require a three stage crushing circuit for feed preparation.

13.4.2 Viability of gold recovery for the near surface domain

The viability of recovering gold from the near surface leached cap has been evaluated by FQM at a conceptual level, assuming a mined resource estimated at up to 127 Mt and at an average grade of 0.34 g/t Au. However, this drops off to only 26 Mt at a grade of about 0.5g/t Au.

Table 13-38 and Table 13-39 indicate high gold recoveries for samples with grades similar to the average Mineral Resource grades:

- 89% recovery from a head of 0.19 g/t at a grind of 95% passing 74 µm
- 63% recovery from a head of 0.21 g/t at a crush size of 100% passing 2 mm
- 46% recovery from a head of 0.21 g/t at a crush size of 100% passing 19 mm

These results compare favourably with those obtained from higher grade samples, and reported in Table 13-38.

A scoping level economic analysis conducted in 2020 using a gold price of $1,200/oz suggested that treating this material either through heap leaching or through CIL, would be only marginally economic, because of the low feed grade. However, at the current gold price of $3,000/oz, the treatment of this material is much more attractive.

A treatment rate of 10 Mtpa using heap leaching at a crush size of minus 12 mm appears to be the best option, albeit that this depends largely on the operating costs which are ill-defined at present. Additional engineering work is required to better define a circuit design, followed by estimation of operating and capital costs.

Additional studies on the effect of gold recovery on the overall site water balance will be required. The scheduling of the gold recovery circuit and its effect on the design, construction and operation of the Taca Taca copper circuit will also need to be evaluated.

This near-surface auriferous material should be set aside on a separate stockpile for future testwork and evaluation.

13.4.3 Gravity gold recovery

Flotation testwork results indicate that approximately 60% of gold present in the feed will be recovered into the flotation concentrates.

Gravity testwork was undertaken by ALS on the ten composite samples tested in 2020 to identify if any gold was recoverable into a gravity concentrate that could then be treated to produce doré.

The testwork procedure was the standard Knelson and panning test using a 100 g cone. The tests were performed on each individual metallurgical composite and did not replicate a typical installation in a milling circuit, where the centrifugal concentrator would be installed on the cyclone underflow stream (the circulating load). Nevertheless, the results were disappointing as shown in Table 13-39.

Table 13-39 Gravity gold recovery

Sample	Gold Grades, g/t		% Recoveries	
	Feed	Cons	Au	Mass
MET001	0.17	2.90	20.1	1.2
MET002	0.26	2.97	4.8	0.4
MET003	0.25	2.64	7.7	0.7
MET004	0.07	0.76	6.8	0.7
MET005	0.02	0.27	7.0	0.6
MET006	0.36	6.51	10.0	0.6
MET007	0.13	1.86	9.4	0.6
MET008	0.08	0.80	6.2	0.6
MET009	0.11	0.87	4.0	0.5
MET010	0.16	2.32	8.9	0.6
Average	0.16	2.19	8.5	0.65
Rough Cons	0.77	26	3.8	0.1

Average gold recoveries to concentrate were 8.5%, giving a concentrate grade of only 2.2 g/t Au. The best recovery of 20% Au from sample MET001 still resulted in a concentrate grade of only 2.9 g/t Au.

The test was repeated using a larger sample of rougher concentrate through a mini pilot plant. This sample had a grade that was considered to be more representative of a cyclone underflow stream. A concentrate grade of 26 g/t Au was achieved, but gold recovery to concentrate was only 3.8%.

These results indicate that there are low gravity recoverable gold values at Taca Taca, and a gravity gold recovery circuit will not be included in the design of the comminution circuit.

Not all ore types have been tested for gravity recovery of gold, and it is felt that this single testwork campaign should be repeated before the gravity circuit is entirely eliminated from the process flowsheet.

13.5 Geochemical characteristics of tailings

The tailings produced by the proposed processing plant would be typical of copper flotation tailings, worldwide. The milling process would result in a particle size distribution with a top size of 180 µm. The ore would be mostly competent and hence a fine clay-like component is unlikely to be present. The tailings should be relatively free draining and following settlement, should consolidate to a dry density of around 1.4 t/m^3.

Testwork on tailings samples was evaluated in the context of their potential for acid rock drainage (ARD) generation. ARD is a product of the oxidation of sulphurous material resulting from exposure to oxygen and water. The consequences of its production are an increase in pH, high sulphate contents and the movement of metals in mining/processing effluents.

Acid Base Accounting (ABA) testing determines the potential for acid production (AP) from sulphur bearing materials, the potential for acid neutralisation (NP) and the paste pH of the materials. It is considered to be a "static" means of testing because it reflects the results only at the time of testing, rather than over a prolonged period. Due to the complex mechanisms developed during the oxidisation of sulphides, over a prolonged period, "kinetic" testing is carried out by means of geochemical analyses on samples stored in the open over several years.

The neutralizing potential (NP) is measured by leaching the sample in hydrochloric acid, and then back-titrating the solution with sodium hydroxide.

The acid generating potential (AP) is calculated from the total sulphur content of the sample multiplied by 31.25, and is the amount of calcium carbonate required to neutralise the acid generated if all the sulphur oxidised to sulphuric acid, as follows:

The net neutralization potential is designated NNP $NNP = Np - AP$

The Neutralization Potential Ratio is NPR $NPR = NP/AP$

This could produce erroneous results if some of the sulphur is present as sulphate minerals that do not oxidise. In the absence of kinetic testing, the ABA static testing procedure is used to determine the potential for acid production using the grading criteria listed in Table 13-40.

Table 13-40 ABA criteria for determining the potential for acid production

ARD Criteria		NP/AP	NNP
Does not present potential for acid production	Non-PAF	>2	>20
Potential for acid production is uncertain	U-PAF	1<NP/AP<2	-20<NNP<20
Presents potential for acid production	PAF	<1	<-20

13.5.1 ABA testwork 2010

Geochemical analyses of testwork tailings were undertaken in 2010 on two samples identified as M1 7758-62 (primary mineralisation) and M1 7763-67 (supergene mineralisation). The results were reported in the Plenge report dated September 2010. The testwork included:

- ABA testwork in order to determine the potential for acid production
- compositional testing (total metals) in order to determine the presence of elements at levels anomalous to naturally occurring concentrations
- short term leach testing in order to identify easily soluble elements in the tailings

The results obtained from this work are presented in Table 13-41, with each result being the average of two tests. The rougher tails were low in sulphur content and consequently showed low acid generating potential, but the cleaner scavenger tails (particularly the supergene cleaner scavenger tails) exhibited higher acid generating potential due to the residual sulphides present.

Table 13-41 ABA testwork results

Sample	NP $CaCO_3$/kt	AP t $CaCO_3$/kt	NNP t $CaCO_3$/kt	NP/AP
Rougher Tail				
Supergene	0.12	9.55	-9.43	0.013
Primary	0.27	5.35	-5.08	0.050
Cleaner Scavenger Tail				
Supergene	1.39	139.9	-138.51	0.010
Primary	1.78	14.50	-12.72	0.123

NNP results of less than minus 20 and NP/AP results of less than 1 indicate the samples to be potentially acid generating. The above results place both rougher tails samples, and the cleaner scavenger tails from the primary ore, in an area of uncertainty with the NNP numbers lying between minus 20 and plus 20. However, all the NP/AP results were less than 1.

In a memo dated May 2011, reported on results obtained from the same samples as tested in the ALS laboratory in Peru (Table 13-42).

Ausenco also performed a Synthetic Precipitation Leach (SPLP), which is designed to simulate leaching of the sample contents in the environment. The sample was extracted at a 20:1 liquid to solids ratio for 16 to 20 hours using water adjusted to pH 5.0 or pH 4.20 by adding drops of 60/40 weight percent mixture of sulphuric and nitric acids. The extract was then filtered through a 0.6 to 0.8 µm glass fibre filter and analysed using ICP.

Table 13-42 ABA testwork results by Ausenco

Sample	Paste pH	Sulphur %	NP t $CaCO_3$/kt	AP t $CaCO_3$/kt	NNP t $CaCO_3$/kt	NP/AP
Primary	7.7	0.05	2	1.56	0.44	1.3
Supergene	6.9	0.09	1	2.81	-1.81	0.4

Table 13-43 lists the results of the compositional and leaching (SPLP) analyses. The results indicate that:

- the two samples contained high levels of Cu, Mo; Ni and Se are considered as anomalous elements (i.e., in higher concentrations than in the earth's crust)
- no significant concentration of the elements from the leaching (SPLP) extract were found

The conclusions drawn from this limited geochemical testing were:

- acid generation or significant concentration of metals from the two tailings samples is not evident
- however, there is uncertainty that warrants further testwork[10]
- there is a need for monitoring during the operations phase

[10] This would be despite the climatic and depositional conditions which may physically limit acid generation

Table 13-43 SPLP leaching result

Sample	Anomalous elements		pH final
	Element	ppm	
Primary	Cu	179	8.24
	Mo	34	
	Ni	176	
	Se	0.7	
Supergene	Cu	459	8.22
	Mo	45	
	Ni	167	
	Se	0.8	

13.5.2 ABA testwork 2020

The tailings geochemistry work was repeated for the ten composite metallurgical samples selected in 2019. Bulk final tails were generated from each sample in 2 kg batch cleaner tests by combining the rougher tails and first cleaner tails. Results of this work are presented in Table 13-44.

All NNP results were less than minus 20, and all NP/AP results were less than 1, indicating that all the samples are potentially acid generating.

Table 13-44 ABA testwork results, 2020

Sample	Paste	Sulphur%		NP	AP	NNP	NP/AP
	pH	Total	Sulphide	$tCaC0_3/kt$	$tCaC0_3/kt$	$tCaC0_3/kt$	
MET001	8.0	1.05	0.29	1	32.8	-32	0.030
MET002	8.0	1.93	0.35	0	60.3	-60	0.000
MET003	8.4	1.64	0.71	1	51.3	-50	0.019
MET004	7.8	1.53	0.95	0	47.8	-48	0.000
MET005	6.2	3.12	1.94	0	97.5	-98	0.000
MET006	8.4	1.21	<0.01	0	37.8	-38	0.000
MET007	8.3	1.19	0.43	1	37.2	-36	0.027
MET008	7.6	2.36	0.64	0	73.8	-74	0.000
MET009	8.1	1.30	0.96	0	40.6	-41	0.000
MET010	7.2	2.19	1.39	2	68.4	-66	0.029

As noted above, the AP is calculated using the total sulphur in the sample, not the sulphide sulphur, and thus the acid generating potential could be overstated if significant levels of insoluble sulphates are present in the material. Re-running these calculations using sulphide sulphur does not change the overall conclusions, however. All NP/AP results remain at less than 1, although four out of the ten samples gave NPP results of between 0 and minus 20, with the highest result for sample MET006 being minus 0.3.

SPLP tests were performed on the ten composite samples. The results indicated no concentration of any of the anomalous elements into the leachate. The final leachate pH varied from 8.36 to 6.21, confirming that there is not acid generation.

13.6 Further testwork requirements

Several aspects of the proposed metallurgical process have not been adequately covered by the testwork completed to date. However, it is felt that these specific items are not critical at this stage of the engineering phase and have been accounted for in the plant design so as to provide potential upside to metallurgical recoveries.

13.6.1 Copper-molybdenum separation

Cu-Mo separation testwork was performed by Lumina on bulk samples derived from locked cycle testwork on 100 kg composite samples of supergene and primary ores representing years 0 to 5 and 6 to 10 of the Project, as then planned. This testwork was carried out using tap water. All four composites achieved a molybdenum concentrate containing greater than 40% Mo at between 47% and 73% Mo recovery. Average results were 57% recovery to a concentrate of 47% Mo.

No Cu-Mo separation testwork was performed on the more recent metallurgical samples tested from 2019 to 2020.

It is accepted that the molybdenum circuit design cannot be predicted by testwork due to limitations in sample size. Plant design will incorporate significant flexibility in operation to allow for variations in mineralogy and flotation response. Typically, the implementation of the molybdenum flotation circuit is delayed, and this will allow testwork to be conducted on actual concentrate.

13.6.2 Sodium hydrosulphide addition to the rougher and cleaner flotation circuits

NaHS is proposed to be used in the Cu-Mo separation circuit, but its potential for modifying the redox potential in the rougher flotation circuit to assist in froth stabilisation, or as a sulphidising agent has not been explored.

The Company's Sentinel operations find NaHS addition to be beneficial when treating some partially oxidised ores or tarnished sulphides, which can otherwise cause froth collapse. Its use at Sentinel is intermittent, and the average consumption is about 11 g/t treated. At the Company's Kansanshi (KMP) operations a range of cupriferous ores is treated including mixed ores containing chalcocite. KMP uses NaHS at a rate of approximately 1 kg/t of ore in a controlled sulphidisation circuit for treating these ores and recently have started adding NaHS to the sulphide float. The ore feed to the sulphide float at KMP is not 100% chalcopyrite, and the addition of NaHS assists in recovery of the secondary sulphides present. The S3 Sulphide circuit at Kansanshi will be designed for NaHS addition of 100 g/t.

Considering that the Taca Taca deposit has no pure primary mineralisation, and the feed to the flotation circuit may contain some partially leached material and secondary sulphides, the circuit chemistry may resemble that at KMP, and thus future testwork should be undertaken to examine the benefit to recoveries of NaHS addition to the rougher flotation circuit.

13.6.3 Flotation reagents

The several testwork campaigns have defined the reagent additions in the flotation circuit, with the exception of frother, and molybdenum promoter and collector.

The consumption of most of the reagents used in the testwork were at levels typical for the industry, but consumption of the frother (MIBC) averaged 120 g/t in the rougher circuit and 150 g/t in the cleaner flotation circuit. These numbers compare with consumption rates for other copper operations which are typically around 50 g/t for the entire circuit.

It is not usual to screen different types of frother during testwork, although several tests with a higher molecular weight frother (designated W31) indicated a consumption of about 30 g/t in the rougher circuit. Higher molecular weight frothers should be evaluated in any future testwork programmes.

The latest testwork in brine did not optimise gold or molybdenum recoveries. No promoter was employed to try and increase gold recovery (although potassium amyl xanthate (PAX) rather than a weaker collector such as sodium ethyl xanthate (SEX) was used). Likewise, the addition of pine oil as used for molybdenum recovery, was stopped part way through the programme.

Despite not optimising for Mo and Au recovery, the ten metallurgical samples provided recoveries of 55% Mo and 58% Au to the third cleaner concentrate in the locked cycle tests.

Further flotation testwork should include a programme focussing on recoveries for gold and molybdenum through reagent optimisation.

13.6.4 Oxidised supergene zone

As noted previously, insufficient samples from the oxidised supergene zone containing chalcocite and high soluble copper ores, have been tested.

Flotation results from only three samples have been used to estimate recoveries (66% Cu) and concentrate grades (12% Cu) for various copper feed grades and pyrite content. At these recoveries and concentrate grades, some of this material may be sub-economic, and it would be beneficial to have more comprehensive metallurgical data from this domain prior to the operating phase of the Project.

13.6.5 Recovery of perched gold

Testwork on the recovery of perched gold has been discussed previously. Bottle roll leaches at various sizes suggested that recoveries would be low (40 to 50%) at crush sizes suitable for heap leaching, although 90% recovery could be achieved in a CIL (carbon in leach) circuit at a grind size of 80% passing 75 µm. These results suggest that a separate gold recovery circuit for this material could be considered.

- comminution testwork to define the milling circuit required
- cyanide destruction on the CIL tailings to define reagent requirements, and for environmental permitting
- sedimentation testwork on tailings from cyanide destruction to define tailings thickener sizing

Further work is required to support a feasibility level study on whether or not it would be economic to process the auriferous material from near-surface.

13.6.6 Sedimentation and filtration testwork

Sedimentation testwork to define thickener sizing was performed on samples from within the starter pit area. This work was originally undertaken using tap water and was repeated on a composite of the ten metallurgical samples in 2020 in brine.

The results in brine gave a settling rate of 1.4 $t/m^2/h$ for final tailings. However, a repeat of this work in fresh water (2021 testwork) suggested a flux of only 0.8 $t/m^2/h$. This work should be repeated to confirm the settling rate to be employed for testwork design.

13.6.7 Flocculant addition to thickeners

There are several areas of the flowsheet where concentrates require dewatering prior to further stages of flotation. These are:

- rougher concentrates prior to regrind
- bulk cleaner concentrates prior to copper-molybdenum separation

There is a concern that addition of flocculants to these thickeners to assist in settling may adversely affect subsequent stages of flotation, since both flocculation and flotation are surface phenomena. However, not using flocculants for these dewatering stages and relying on conventional thickeners would lead to large thickener area requirements.

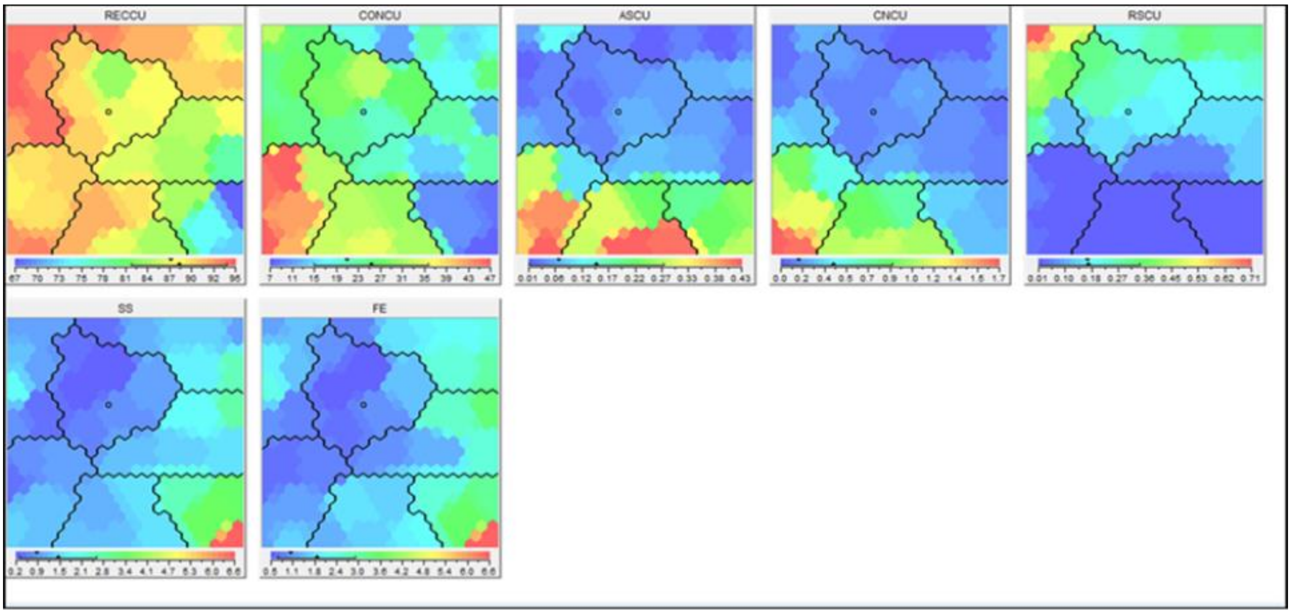
Some testwork should be undertaken to measure the effect of flocculants on flotation, and if noticeable, to investigate various methods of mitigation. Additional sedimentation testwork might be required to firm up on conventional thickener sizing requirements.

13.7 Metallurgical recovery estimates

The testwork variability samples included mineralogical information on sulphide sulphur content, iron, and sequential copper data (i.e., acid soluble copper, cyanide soluble copper and residual copper), together with recovery and concentrate grade data.

Multi-variate analysis (neural network analysis, NNA) was completed on this variability data during the course of the Mineral Resource modelling (Item 14). This highlighted distinct groupings related to recovery and concentrate grades. Figure 13-24 shows the analysis charts for recovery (RECCU), copper concentrate grade (CONCU), soluble copper grade (ASCU), cyanide soluble copper grade (CNCU), primary sulphide copper grade (RSCU), sulphide sulphur grade (SS) and iron grade (FE).

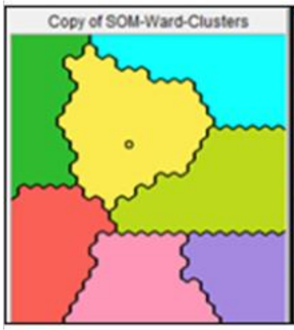
Figure 13-24 Neural network analysis charts



The analysis indicated seven clusters with attributes as shown in Figure 13-25. Delving into the detail of these interpreted clusters in relation to individual samples reveals that:

- high and low recoveries group together with concentrate grade
- lowest recovery and concentrate grade (in cluster C5) correlates with high soluble copper and high pyrite content
- highest recovery (in cluster C4) correlates with high grade primary mineralisation

Figure 13-25 Overall cluster attributes



Cluster	Description	Abs. Profil...	Frequency	RECCU	CONCU	ASCU	CNCU	RSCU	SS	FE
C1		0.689	17.50%	90.06	18.69	0.0429	0.121	0.2643	1.421	2.091
C2		1.441	15.00%	90.42	42.52	0.3083	1.192	0.0300	1.048	1.112
C3		0.787	15.00%	87.82	26.53	0.0483	0.173	0.2267	0.590	0.943
C4		1.205	17.50%	94.10	26.31	0.0557	0.206	0.4143	1.036	1.491
C5		1.932	10.00%	74.42	9.45	0.2350	0.305	0.0100	4.005	3.905
C6		1.178	12.50%	88.48	32.80	0.3300	0.968	0.0120	1.364	1.696
C7		0.000	12.50%	84.16	18.82	0.0560	0.164	0.1140	1.770	2.146

These clusters formed the basis of an understanding as to what influences recovery and concentrate grade. This understanding guided the programme of drill core logging in 2019 and a reinterpretation of mineralisation domaining. The net result of considering the interpreted C1 to C7 clusters along with the core logging work and interpretation of geological continuity, is the current definition of four broad (i.e., mixed, secondary, supergene mixed, primary) mineralisation domains as presented in Table 13-45. The groupings in Table 13-45 are an update on preliminary FQM interpretations that had existed since 2018.

This data was combined with the recent locked cycle testwork results described above to estimate recoveries and concentrate grades for the distinct ore types, copper head grades and pyrite content. These estimates are presented in Table 13-45 and have been coded into the Mineral Resource model (Item 14) and are recommended for adoption in all mine production scheduling and cashflow modelling until next updated.

The overall average copper recovery over the life of mine is expected to be 86.7% at a concentrate grade of 25.3%. The recommended recoveries and concentrate grades for the other processed metals, as applicable for both primary and supergene plant feeds, are as follows:

- Mo recovery is 45% into a concentrate with an average grade of 47%
- Au recovery (into concentrate) is 61.4% at a typical grade of 4.5 g/t

13.7.1 Comments on the estimates

Considering the results from the original Lumina testwork at the Plenge laboratories in Lima, as well as the recent locked cycle tests in brine and fresh water, several trends are evident from the groupings in Table 13-45:

- low recovery and concentrate grades correlate with high soluble copper and high pyrite content
- high pyrite content affects recovery and concentrate grade, no matter what the copper grade or mineralisation
- highest recovery correlates with high grade primary mineralisation
- recovery and concentrate grades increase with increasing copper head grade (as expected)
- highest concentrate grades are achieved from samples with high chalcocite content

It should be noted that much of the early flotation work was completed before there was a clear understanding of the material types making up the ore body, and it is difficult to assign recoveries and concentrate grades from this work to the metallurgical domains shown in Table 13-45.

Thus, for some of the ore domains there is insufficient metallurgical testwork to fully support the recoveries and concentrate grades shown.

In particular, the supergene oxidised zone is poorly represented in the metallurgical samples. However, this zone only represents 3% of the orebody, and is not currently considered in the Mineral Reserves. The supergene mixed ores are well represented by samples tested by both Plenge in the variability testwork, and by ALS. This ore type will comprise the majority of the feed to the plant in the early years of the Project. The supergene transition zone was represented in seven samples tested by Plenge in their variability work, but no samples from this zone were tested in the more recent ALS work. Confirmatory testwork on ores from this zone would be of benefit, but is not considered critical for current design purposes. Testwork has adequately covered the hypogene ores.

The previous variability testwork as documented in the October 2012 Plenge report contained three samples that were very high Fe (3.1% to 6.65%), with total Cu of 0.5% to 0.6% and high acid soluble Cu (42% to 49%). These samples were classified as oxidised, medium Cu, high pyrite and gave 66% recovery at a concentrate grade of 12% Cu (in a batch cleaner test, in tap water). Those results were used as a basis for recoveries for the oxidised zone.

Two recent metallurgical samples, MET001 and MET008, were classified as partial leach. They gave recoveries of 73.8% and 80.3% Cu, respectively, at 35% and 37% Cu concentrate grades. The assays indicated only about 12% acid soluble copper and medium Fe levels, and therefore those results were applied to the supergene mixed, medium Cu grade, and medium pyrite material.

MET005 was a very difficult sample to deal with in the testwork, giving copper recoveries of 64.4% at a concentrate grade of 14.7% Cu. A sample from the variability testwork gave 79.7% Cu recovery at a concentrate grade of 12.4% Cu. Both samples were classified as Supergene/Mixed with medium Cu grade and high Fe. As with the three samples classified as oxidised, described above, it appears that the high Fe (pyrite) content of some samples leads to lower recoveries and poor concentrate grades.

Where individual ore types (e.g. supergene transition, low grade, low pyrite content) are not covered by testwork, recoveries and concentrate grades have been estimated by interpolation from data on adjacent sample types.

Grade-recovery relationship equations for each metallurgical type cannot be derived at this time.

Additional confirmatory testwork is recommended for samples representing the metallurgical domains that are expected to be mined in the first five years of the Project life, and utilising fresh or brackish water sourced from the borefields that will be supplying water to the process.

These tests should replicate the currently recommended flowsheet and reagent additions and should include locked cycle testwork to confirm achievable recoveries and concentrate grades.

Table 13-45 Metallurgical domains, 2024 update

Geological Process	Copper species	Relative proportion	% of Total Ore	Domain (DOM) numeric value	Material type (MATTYPE) Name	% Acid Soluble Cu	Cu%	Pyrite	Fe%	Met domain	Cu Recovery, %	Cu Con Grade, % Cu	Au Recovery, % (ALS testwork)	Supporting Met Testwork (ALS)	Supporting Met Testwork (Plenge 2012 Variabilty Tests)
Supergene	Chalcocite,	CC>CPY + Sol Cu	3%	203	MIXED	>10	<0.4	low	<1.5	C2	65	15			
mixed	chalcopyrite,			203	MIXED	>10	<0.4	medium	1.5-2.4	C2	65	15			
oxide	black Cu			203	MIXED	>10	<0.4	high	>2.4	C2	65	15			
	oxides,			203	MIXED	>10	>0.4	low	<1.5	C2	74	20			
	soluble			203	MIXED	>10	>0.4	medium	1.5-2.4	C2	74	20			
	copper			203	MIXED	>10	>0.4	high	>2.4	C2	70	20			3 Samples
Supergene		CC>>CPY	19%	304, 305, 310	SECONDARY	>10	<0.4	low	<1.5	C5	85	25			2 samples
mixed				304, 305, 310	SECONDARY	>10	<0.4	medium	1.5-2.4	C5	83	25			
	Chalcocite,			304, 305, 310	SECONDARY	>10	<0.4	high	>2.4	C5	80	22	32	Met 005	
	chalcopyrite			304, 305, 310	SECONDARY	>10	0.4-1	low	<1.5	C6	88	34	55	Met 001	2 samples
				304, 305, 310	SECONDARY	>10	0.4-1	medium	1.5-2.4	C5	88	34	56	Met 008, 009	1 sample
				304, 305, 310	SECONDARY	>10	0.4-1	high	>2.4	C5	86	25			1 sample
	Chalcocite,			304, 305, 310	SECONDARY	>10	>1	low	<1.5	C6	88	35	47	Met 002, 006	5 samples
	Chalcopyrite,			304, 305, 310	SECONDARY	>10	>1	medium	1.5-2.4	C5	88	32		Met 003	4 samples
	bornite			304, 305, 310	SECONDARY	>10	>1	high	>2.4	C5	86	30			
Supergene	Chalcocite,	CC>CPY	59%	307,308, 309	SUPERGENE MIXED	~10	<0.4	low	<1.5	C7	88	27			
transition	chalcopyrite			307,308, 309	SUPERGENE MIXED	~10	<0.4	medium	1.5-2.4	C7	86	25			1 Sample
				307,308, 309	SUPERGENE MIXED	~10	<0.4	high	>2.4	C7	84	22			2 samples
	Chalcocite,			307,308, 309	SUPERGENE MIXED	~10	>0.4	low	<1.5	C1	90	30			2 samples
	chalcopyrite,			307,308, 309	SUPERGENE MIXED	~10	>0.4	medium	1.5-2.4	C1	90	30			
	bornite			307,308, 309	SUPERGENE MIXED	~10	>0.4	high	>2.4	C7	88	26			2 samples
Hypogene	Chalcopyrite,	CPY>CC	20%	306	PRIMARY	<10	<0.4	low	<1.5	C3	90	25	74	Met 010	2 samples
	chalcocite			306	PRIMARY	<10	<0.4	medium	1.5-2.4	C3	90	25			4 samples
				306	PRIMARY	<10	<0.4	high	>2.4	C3	88	22			2 samples
	Chalcopyrite,			306	PRIMARY	<10	>0.4	low	<1.5	C4	90	28			3 samples
	chalcocite,			306	PRIMARY	<10	>0.4	medium	1.5-2.4	C4	90	28	74	Met 007	5 samples
	bornite			306	PRIMARY	<10	>0.4	high	>2.4	C4	90	26	78	Met 004	1 sample

ITEM 14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

Taca Taca Mineral Resource estimates were updated for copper (Cu), gold (Au), silver (Ag), molybdenum (Mo), iron (Fe) and sulphur (S) elements. Grade estimates were completed by David Gray (QP) of FQM, using commercially available software, Datamine Studio RM (v 2.1.125) and Snowden Supervisor (v 9). Grades were estimated into a block model from drill hole sample results using ordinary kriging.

Sample assay results were exported from a secure database and used together with a geological model that relates to the spatial positions of copper, gold and molybdenum mineralisation. The estimation parameters were guided by geology, style of mineralisation, drill sample spacings, and geostatistical analysis.

Block estimates were classified according to data quality, geological and grade continuity, drill sample spacing, confidence in estimates, and a life of mine pit shell (reasonable prospects for economic extraction). Reporting has used the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy, and Petroleum (the CIM Guidelines, 2014). Declared Mineral Resources are reported within the life of mine pit shell together with a copper, gold and molybdenum equivalent cut-off grade of 0.11%.

14.2 Database data

A three-dimensional block model was limited to an upper topographic surface and a base elevation of 2,395 m (above mean sea level) to cover the extents of drill samples and the deposit extents. Drill hole samples were collected by BHP, Corriente, Rio Tinto, Lumina, and FQM (Table 14-1). Lumina samples comprise 81% of all samples and have greatest influence on the estimate results. In total, 405 holes were drilled within the modelled volume, with 164,822 m sampled from 399 holes (Figure 14-1).

Table 14-1 Drill holes with assay data as used in this Mineral Resource estimate

Year	Company	Number of holes	Hole purpose	Hole type*	Hole ID sequence	Meters drilled
1996	BHP	4	Exploration	RD	TK01 - TK04	1,651
1997	BHP	26	Exploration	RD	TK05 - TK25,TK27 - TK33	8,548
1998	Corriente	14	Exploration	DD	TK34 - TK47	3,328
1999	Corriente	64	Exploration	DD	TK48 - TK126	3,665
1999	Rio Tinto	7	Exploration	RC	CCR001 - CCR007,ARI001	2,732
2008	Rio Tinto	8	Exploration	DD	TTBJ0001 - TTBJ0008	4,877
2010	Lumina	5	Exploration	DD	TTBJ10-01 - TTBJ10-5	3,436
2011	Lumina	64	Resource development	DD/RD	TTBJ10-06,TTBJ11-07 - TTBJ11-73	42,364
2011	Lumina	1	Exploration	RC	TTEX01	200
2011	Lumina	4	Exploration	DD	TTEX02,TTEX03,TTEX07	1,851
2011	Lumina	4	Geotechnical	DD	TTGT01 - TTGT-04	2,404
2011	Lumina	16	Resource development	RC	TTRC11-01 - TTRC-17,TTRC12-88/94/95	3,496
2012	Lumina	5	Exploration	RC	FR12-05 - FR12-10	1,878
2011	Lumina	1	Water monitoring	DD	TTB11-44	300
2012	Lumina	11	Geotechnical	DD	TTTV1-TTTV11	5,905
2012	Lumina	66	Resource development	DD/RD	TTBJ11-72 - TTBJ11-76,TTBJ12-77 - 136	42,602
2012	Lumina	81	Resource development	RC	TTRC12-18 - TTRC12-97	28,559
2012	Lumina	2	Water monitoring	RC	AV-SP4S/SP5D	260
2019	FQM	3	Metallurgical	DD	TTBJ19-138 - TTBJ19-140	1,065
2022	FQM	13	Exploration	DD	TTBJ22-141 - TTBJ22-152/155/162	5,701
TOTALS		**399**				**164,822**
* DD=diamond drilled holes, RC=reverse circulation holes, RD=diamond drilled hole with RC pre-collar						

Drill hole data includes collar coordinates, downhole surveys, assays, logs of geology, weathering, minerals, alteration and structures, magnetic susceptibility and SWIR spectrometer readings. For core drilled since 2010, core recovery, RQD, point load test, and density measurements were recorded.

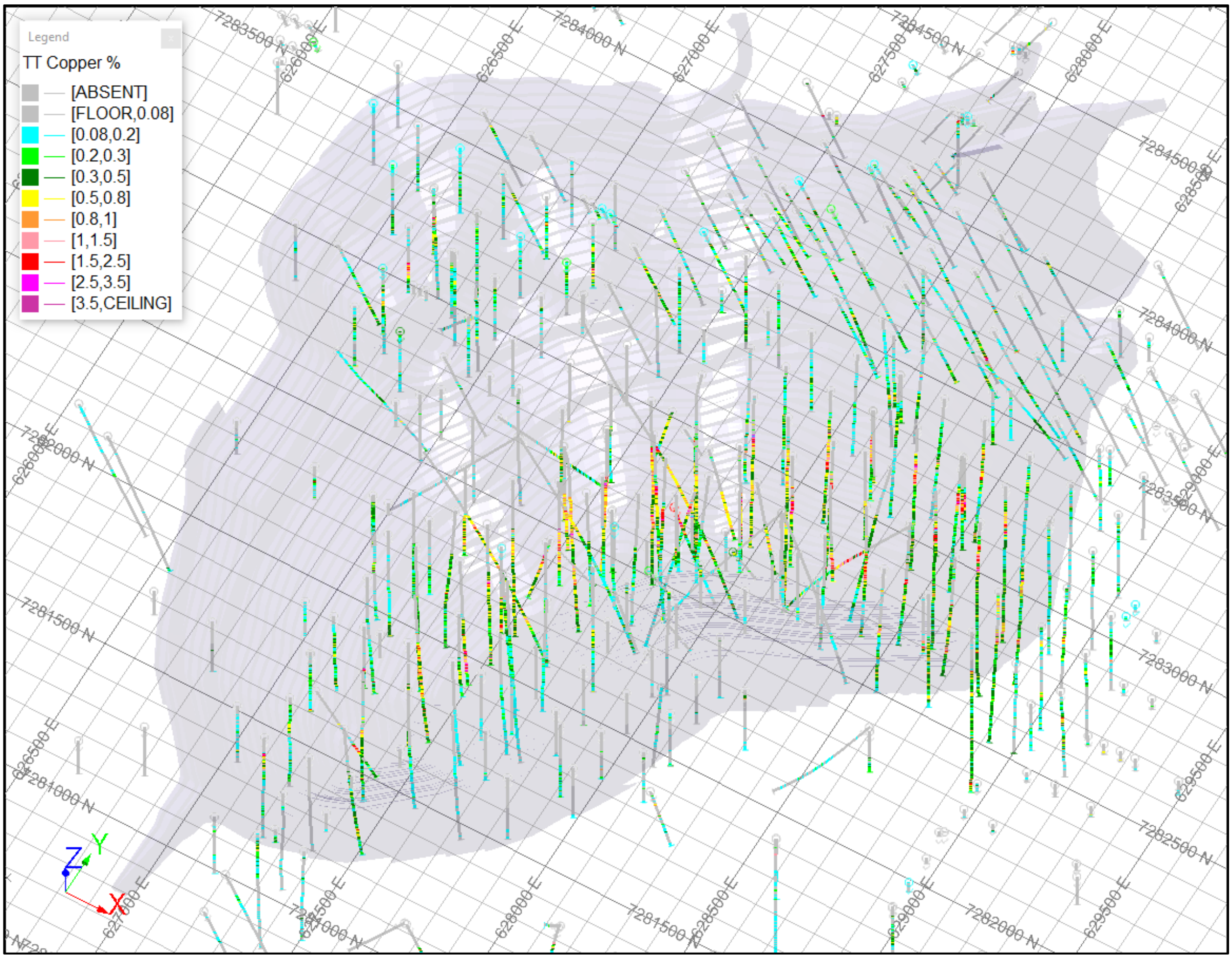
Samples were assayed for Cu, Au, Ag and Mo with more than 90% of samples having ICP multi-element analysis. Sequential leach copper assays were completed for around 10% of samples, however, these values were not estimated owing to their limited spatial distribution and poor ability to represent the in-situ styles of copper mineralisation. Correction factors were applied to subsets of the sequential copper data. Sequential leach copper assays were only used to provide an indication of dominant copper mineral species per geology domain.

QAQC results were available for samples from holes drilled since 2008. Samples were determined to have an acceptable level of precision and accuracy for use in the Mineral Resource estimate. Data were subject to routine validation checks with limited data errors or inconsistencies identified. The following adjustments were made:

- assay data with negative values were reset to values at half the lower detection limit for that element
- different logging codes used by previous owners to represent the same lithology were grouped into a merged rock-type field, to facilitate geology modelling
- sequential copper leach values were normalised to their respective total copper assay values.

Around 98% of samples were taken from a 2 m sample length. Drillhole data was not composited.

Figure 14-1 An oblique 3D view, looking northwest, of the available drill holes used in this Mineral Resource estimate. Holes are coloured by total %Cu relative to the life of mine pit shell



14.3 Topography

In 2019, a high-resolution topographic survey was acquired via WorldView3. The survey covered an area of 12 km by 23 km at 0.5 m resolution, and over the wider surrounding areas at a 3 m resolution. The resulting topography data limits the upper extents of the block model estimate and was also used to support an improved understanding of the position and extents of immediate surficial features such as outcrop, soil cover, scree slopes and the salar limits.

14.4 Geological model and domains

Geology, mineralisation and multi-element data were analysed statistically and spatially to identify and define rock type and weathering domains that have clear controls on mineralisation. The resulting domains (Table 14-3) limit grade variability to support representative grade estimates.

Analysis of multi-element assays, sequential leach copper data, and drill core logging data resulted in eleven geologically distinct domains (Table 14-3) which relate to rock type (Table 14-2), weathering, alteration, and mineralisation features. Neural Network Analysis together with self-organising feature maps (using Viscovery SOMine 7 software) was used to identify these key domains. Resulting domains were validated against logged data and then represented as three-dimensional wireframe surfaces or volumes.

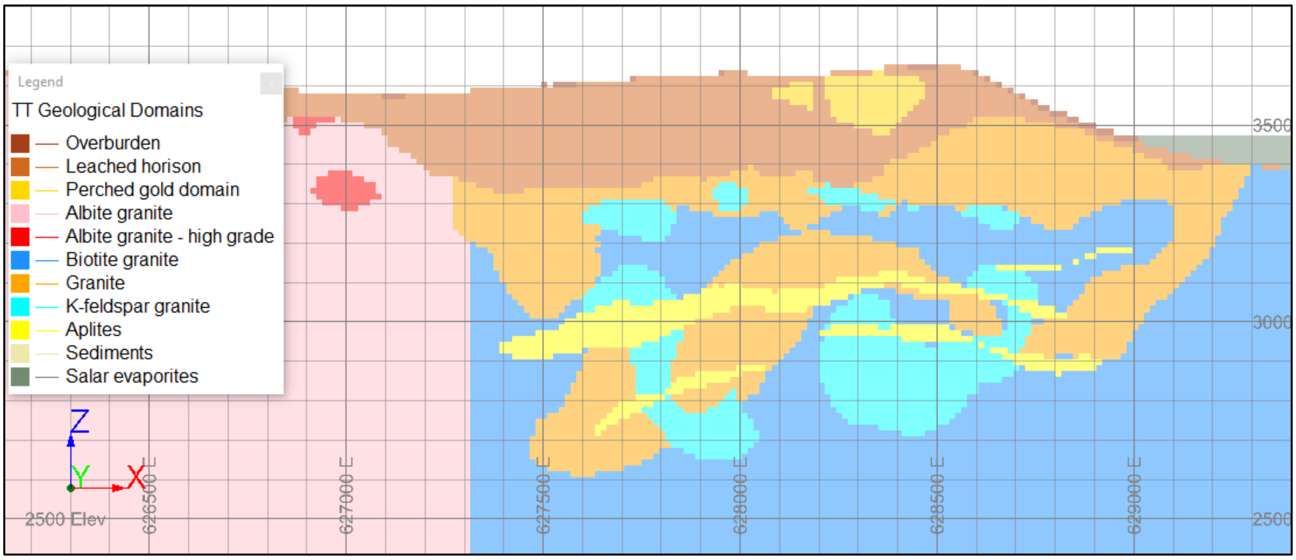
The upper 15 m to 300 m thick leached cap and partially leached zone were modelled as sub-horizontal surfaces (Figure 14-2), with the latter grouped with fresh rock due to its irregular geometry. The base of the leached horizon was identified from logging data together with copper and sequential copper sample assay values.

Variations in copper mineralogy across domains were confirmed by sequential copper data and visual inspection of drill core highlighting that supergene chalcocite is dominant in phyllic-altered granite (7 - Table 14-2) and primary sulphide minerals in biotite granite (6 - Table 14-2). A significant fault (TK2) displacing the weathering contact was included in the final geology model.

Table 14-2 Rock-type 'ROCK' field numerical codes and their associated descriptions

ROCK field number	ROCK field description
1	Leached
2	Perched gold
3	Perched copper
4	Albite granite
5	Albite granite – high grade
6	Biotite granite
7	Granite
8	K-feldspar granite
9	Aplites
10	Sediments
11	Salar Evaporites

Figure 14-2 Section 7282814N, looking north, showing geological domains used to code drillhole samples and the block model



Estimation domains (Table 14-3) were defined from combined rock and weathering domains which were spatially defined by wireframe surfaces or volumes. The block model and sample assay data was coded according to these wireframes to facilitate spatial analysis and block model estimates.

Table 14-3 Estimation domain field codes and descriptions as per rock type and weathering, where 100 denotes weathered rock and 200, fresh rock

Domain Description	Domain field value	3D Model Constraint	Approximate deposit location	Dominant Mineralisation type
Leach	101	Wireframe Surface	150m-300m surface cap	Cu depleted, narrow remnant copper oxides
Perched Gold	102	Wireframe Volume	Enriched leached material to the east and southeast	Au and Mo mineralisation
Perched Copper	203	Wireframe Volume	Remnant zone of supergene mineralisation in partially leached material to the east	Chalcocite>>soluble copper
Fresh Albite Granite	304	Outer zone external to wireframe	Rimming the porphyry to the northwest	Chalcocite >> soluble Cu > primary sulphides
Fresh Albite Granite – high grade	305	Wireframe Volume	High grade pockets internal to the fresh albite domain (304) in the northwest	Chalcocite >> soluble Cu
Fresh Biotite Granite	306	Remaining volume external to wireframes	Throughout central, south, and southeast areas and intermixed with domain 307 fresh granite	Primary sulphides >> chalcocite
Fresh Granite	307	Wireframe volumes	Phyllic altered zone to south, and southeast areas. Intermixed with domain 306 fresh biotite granite	Chalcocite>>soluble copper
Fresh K-feldspar granite	308	Wireframe volume	To southeast of the deposit associated with the potassic porphyry core	Primary sulphides>>chalcocite
Fresh Aplite	309	Wireframe volume	Largest sills are to the southest and south	Chalcocite>>soluble copper
Fresh Metasediments	310	Wireframe Surface	Overlaying the porphyry to the northwest	Soluble copper, chalcocite, Au-Cu veins

14.5 Geometallurgical domains

Additional metallurgical testwork completed in 2020, following four dedicated metallurgical drill holes from the 2019 drilling campaign, supported an update to the geometallurgical interpretation relative to the previous Technical Report (refer to Item 13).

Multivariate analysis, including neural network analysis (NNA), was applied to the testwork dataset and identified distinct groupings related to metallurgical response (Table 13-26). Interpretation of these groupings provided insight into the geological controls on metallurgical performance and informed a reinterpretation of geometallurgical domaining.

Based on this analysis, a set of geological proxies was defined to support geometallurgical domaining. Iron and sulphur contents, together with total copper, were incorporated as key variables to differentiate geometallurgical domains.

Integration of the interpreted clusters (C1 to C7) with geological continuity and these proxies resulted in the definition of four broad mineralisation domains—mixed, secondary, supergene mixed, and primary—as presented in Table 13-45. Within each domain, material is further subdivided using discrete copper grade classes (Cu% <0.4, >0.4, 0.4–1, >1), pyrite classes (low, medium, high), and iron classes (Fe% <1.5, 1.5–2, >2.4), forming the geological framework for geometallurgical domaining (Table 13-45).

14.6 Data analysis and variography

Univariate statistical analysis of sample grades per domain evaluated data distributions, population mixing, and high-grade variability. The fresh rock domains have reasonable normal distributions with some mixing evident. Top-cuts were applied to Cu (>6 % Cu), Mo (>3000 ppm Mo), and Au (>2.5 g/t Au) to limit the risk of extreme high-grade values that could affect estimates. High grade values comprised less than 0.04% of total samples available. Top-cutting marginally impacts mean grades but reduces the coefficient of variation, aiding estimation quality. The resulting domain statistics supported the development of robust variogram models for ordinary kriging.

Boundary analysis, informed by contact profiles and geological context, led to the use of hard boundaries for domains with distinct lithological or mineralogical properties (domains 101, 307, 308, 309 and 310) and soft boundaries where transitional behaviour was observed in domains 304, 305 and 306.

Spatial analysis was conducted using variography in Snowden Supervisor, with three-dimensional continuity modelled through normal scores variograms. Anisotropy directions were defined using variogram fans, and nugget effects were derived from downhole variograms. Variogram models were standardised, back transformed, and calibrated to each domain's population variance. The models showed well-defined nugget values and anisotropic ranges, with Cu, Au, and Mo sharing consistent orientations. These variograms informed the search strategy and underpinned the estimation process, as detailed in Table 14-4, and Table 14-6.

Table 14-4 Summarised variogram parameters for Cu, Au, and Mo estimates

Domain	Grade	Variogram Rotation Angles			NUGGET	Spherical model 1 ranges				Spherical model 2 ranges			
		Z	X	Z		X	Y	Z	Sill	X	Y	Z	Sill
101	CUPCT	110	90	10	0.15	88	51	96	0.22	937	286	708	0.63
102	CUPCT	-40	10	-90	0.19	252	83	21	0.29	266	241	221	0.52
203	CUPCT	-60	5	-180	0.23	358	118	12	0.29	372	279	34	0.48
304	CUPCT	155	20	20	0.13	227	177	26	0.32	248	244	212	0.55
305	CUPCT	155	20	20	0.13	218	33	49	0.35	282	177	190	0.52
306	CUPCT	-180	100	170	0.12	112	52	27	0.11	425	195	260	0.77
307	CUPCT	170	170	10	0.09	218	85	20	0.22	729	399	257	0.69
308	CUPCT	95	100	60	0.11	274	344	258	0.26	284	345	259	0.63
309	CUPCT	115	60	0	0.12	564	26	352	0.09	565	212	354	0.79
310	CUPCT	85	170	10	0.09	218	85	20	0.22	729	399	257	0.69
101	AUPPM	110	90	10	0.15	88	51	96	0.22	937	286	708	0.63
102	AUPPM	-40	5	140	0.3	209	120	7	0.25	257	159	28	0.45
203	AUPPM	-60	5	-180	0.09	217	118	6	0.12	252	179	90	0.79
304	AUPPM	155	20	20	0.13	125	177	26	0.33	307	244	212	0.54
305	AUPPM	155	20	20	0.13	162	33	22	0.37	282	151	190	0.5
306	AUPPM	-180	100	170	0.12	112	52	27	0.11	425	195	260	0.77
307	AUPPM	170	160	10	0.09	218	85	20	0.22	796	399	257	0.69
308	AUPPM	90	120	60	0.11	274	344	258	0.26	284	345	259	0.63
309	AUPPM	75	150	140	0.12	564	26	164	0.14	565	212	165	0.74
310	AUPPM	85	170	10	0.09	218	85	20	0.22	796	399	257	0.69
101	MOPPM	150	170	10	0.18	109	126	19	0.18	1382	800	353	0.64
102	MOPPM	-40	5	-15	0.32	65	68	13	0.33	225	178	103	0.35
203	MOPPM	-60	5	-180	0.23	358	272	11	0.11	372	556	96	0.66
304	MOPPM	155	20	20	0.13	68	177	26	0.34	314	244	109	0.53
305	MOPPM	155	20	20	0.13	218	33	49	0.35	282	177	268	0.52
306	MOPPM	-180	100	170	0.19	82	27	103	0.37	425	195	260	0.44
307	MOPPM	170	150	10	0.24	194	192	256	0.16	1158	399	329	0.6
308	MOPPM	95	100	60	0.35	98	190	114	0.16	487	349	395	0.49
309	MOPPM	115	60	0	0.3	86	11	267	0.23	837	212	373	0.47
310	MOPPM	85	170	10	0.24	194	192	256	0.16	1158	399	329	0.6

Table 14-5 Summarised search ellipses per domain estimate (for all grades)

DOMAIN	Search axis rotation			First pass search radius			Second pass radius multiplier
	Z	X	Z	X	Y	Z	
101	110	90	10	450	150	300	1.5
102	-40	10	-90	150	150	50	1.5
203	-60	5	-180	200	150	50	1.5
304	155	20	20	200	150	100	1.5
305	155	20	20	250	150	150	1.5
306	-180	100	170	250	140	150	1.5
307	170	170	10	200	150	130	1.5
308	95	100	60	200	175	125	1.5
309	Uses dynamic anisotropy			225	125	75	1.5
310	85	170	10	250	180	140	1.5

Table 14-6 Summarised search parameters per domain estimates (for all grades)

DOMAIN	Max. # samples per hole
101	6
102	6
203	6
304	4
305	4
306	4
307	4
308	4
309	4
310	4

Search Pass	Min. # of samples	Max. # of samples
First	8	24
Second (1.5*initial search ellipse)	8	24

14.7 Block model construction

A kriging neighbourhood analysis (KNA) was undertaken using Snowden Supervisor to optimise block size, search ellipse dimensions, and sample selection criteria (Table 14-5 and Table 14-6) for grade estimation, based on variogram models and outputs such as kriging efficiency and regression slope. A parent block size of 60 m × 60 m × 15 m was chosen to balance estimation performance with drill spacing, SMU dimensions, and the base of leach and topographic representation. The block model was built in Datamine (See Table 14-7) and coded with topography, weathering, and lithology domains. Dynamic anisotropy was applied to the sub-horizontal aplite sill (domain 309) volumes. A SMU dimension of 15 m × 15 m × 7.5 m was used to enable adequate volumetric resolution within the 60 m parent blocks as well as to best reflect grade and tonnage results at the scale of mining.

Table 14-7 Block model settings used in the Mineral Resource estimate

Model setting		Value
Origin	X	626,100
	Y	7,280,500
	Z	2,395
Maximum	X	629,640
	Y	7,284,880
	Z	3,820
Parent cell size	X	60
	Y	60
	Z	15
SMU cell size	X	15
	Y	15
	Z	7.5

14.8 Density estimation

5,363 dry density Archimedes method determinations were acquired from diamond drilling campaigns conducted between 2010 and 2012. The available density data has good spatial coverage and adequate representation across the respective geological domains. Statistical analysis of the density data identified outlier measurements below 2.00 t/m³ and above 3.50 t/m³ for exclusion. These values were deemed to be well beyond the expected range for granitic lithologies. No direct measurements were available for evaporitic units; therefore, a nominal density of 1.50 t/m³ was assigned to these materials. Recognizing the pronounced influence of weathering on rock density, estimates were derived for individual weathering domains (leached and fresh). For model blocks lacking sufficient empirical data, a default density value of 2.65 t/m³ was applied.

Table 14-8 Mean density value per weathering domain

WEATH Field	Description	Cut Mean (t/m^3)
100	Leached material (excluding evaporites)	2.56
200	Partially leached material	2.64
300	Fresh material	2.67

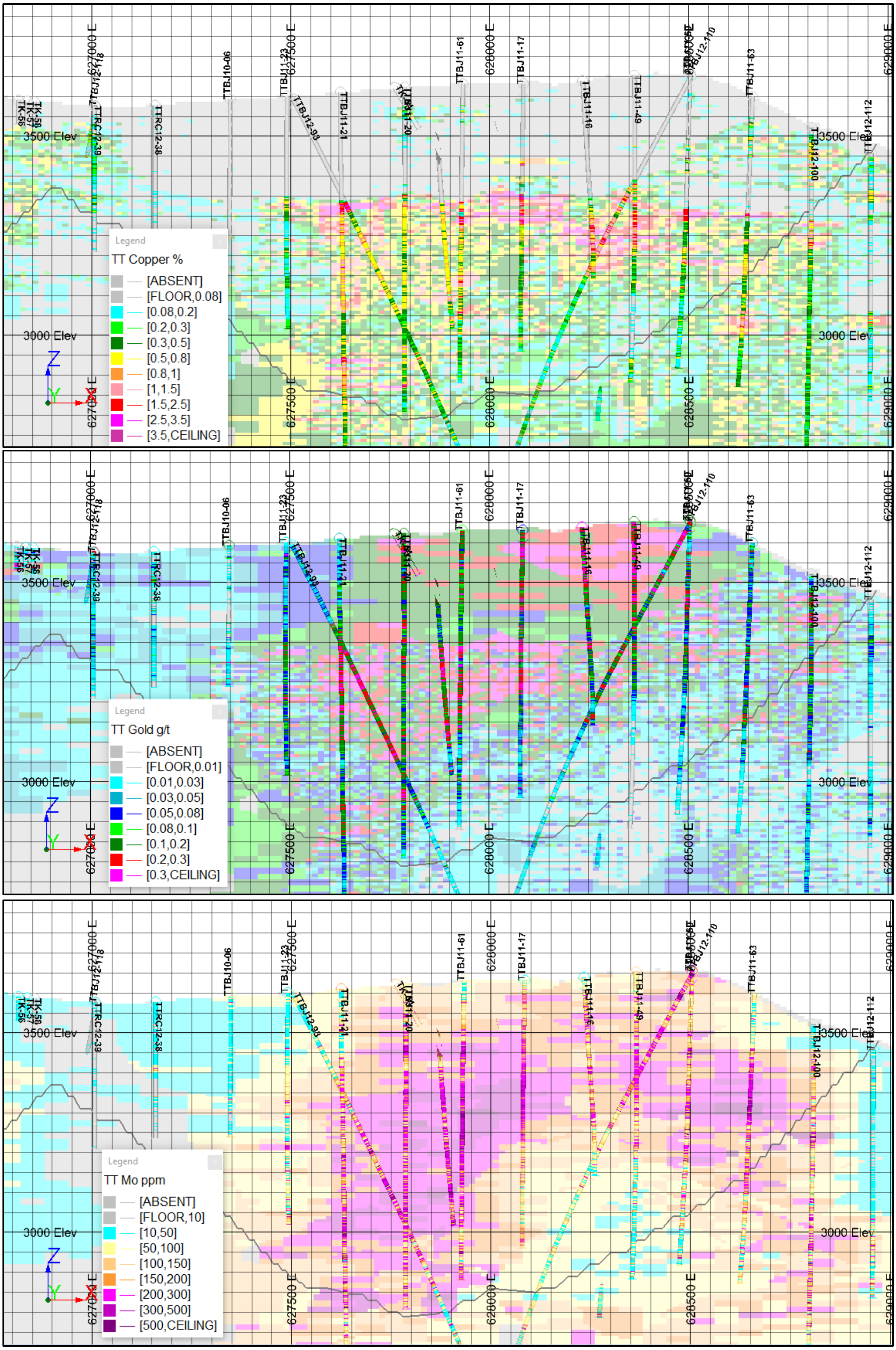
14.9 Grade estimation and model validation

All grade and density estimates were completed using ordinary kriging (OK) into parent blocks, selected as the appropriate method due to the near-normal grade distributions and minimal population mixing within mineralised domains. Estimation parameters were derived from variography, kriging neighbourhood analysis (KNA), geological continuity, and drill spacing. Estimates used top-cut sample assays. Most block grades were estimated using the first search ellipse, while peripheral zones with sparse data required an expanded second search. Discretization settings of 8×8×4 points were used for accurate volume representation. Un-estimated peripheral blocks were assigned trace grades. To generate mine-scale grade and tonnage predictions, localised uniform conditioning (LUC) was applied to fresh material blocks from the first search pass, generating SMU block estimates per parent block with total metal preserved. While LUC post processing enhances estimate grade and tonnage results at the scale of mining, the current drill spacing limits SMU-level spatial precision. Domain-level validation confirms no metal gain or loss during LUC post-processing.

A comprehensive validation process was undertaken to confirm that block grade estimates accurately reflect the input data and underlying geology. This included visual comparisons of sample and block grades on 2D cross sections, trend validations using swath plots across northing, easting, and vertical slices, and checks comparing mean sample grades to corresponding mean block model estimates within each domain.

Visual checks indicated that the estimated grades represent the input data, while swath plots showed strong alignment in well-informed areas, as illustrated for domain 306 (granite). Comparisons were made for copper and gold using both OK and LUC estimates, and for molybdenum using OK only. Overall, validation has confirmed that block model estimates adequately represent the in-situ mineralisation as per the available sample support.

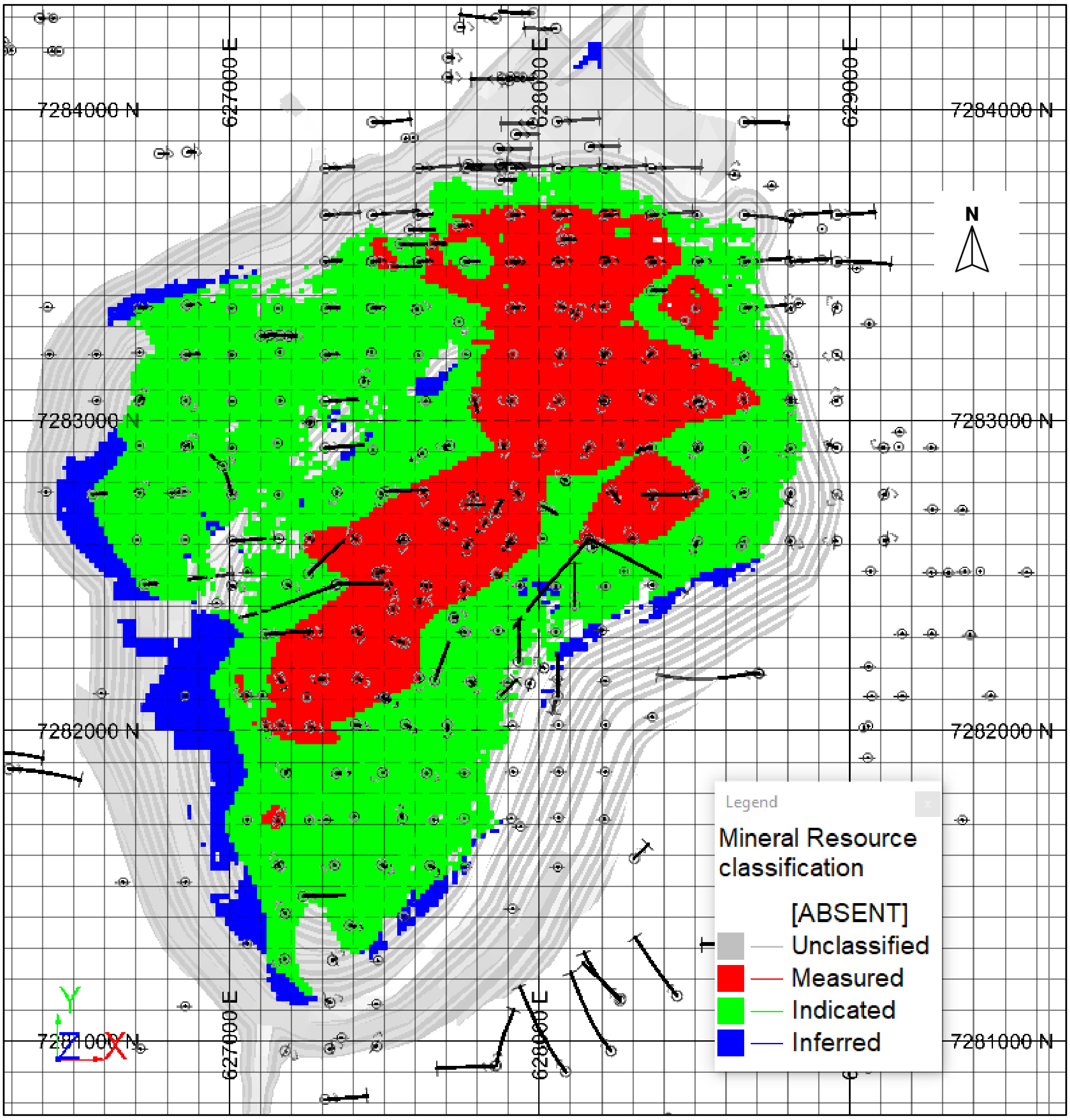
Figure 14-3 **Vertical section 7282775N, looking north. Compares drillhole sample grades and block model grades: %Cu (top); g/t Au (middle); ppm Mo (bottom)**



14.10 Mineral Resource classification

The Mineral Resource estimate was classified as Measured, Indicated, and Inferred (Figure 14-4) in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and is consistent with the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019), based on factors such as concession title verification, data quality (drilling, sampling, assaying, and geology), drill spacing, geological model reliability, in-situ dry bulk density, kriging efficiency and regression slope statistics, and validation between sample and block grades. Classification has considered reasonable prospects for economic extraction and was constrained to a life-of-mine pit shell. As such, Mineral Resources were reported within the life of mine pit shell as per the detailed Mineral Reserve studies and prevailing economics. All mineralisation located external to this pit, as well as the leached zones, perched gold, Salar de Arizaro evaporites and meta-sediments were deemed as unclassified due to their poor economic viability. Wireframes were used to delineate the volumes for each resource classification (Figure 14-4). Fresh mineralisation located outside the pit at depth, to the south and to the east, remains under review for future exploration.

Figure 14-4 Measured, Indicated, and Inferred volumes at Cu equivalent cut-off >0.11%, shown relative to drill holes and the life of mine pit shell



14.11 Mineral Resource statement

Since mineralisation consists of several metals of economic value, the gold and molybdenum grades were converted by formula and added to the grade of copper. The following equation was used to calculate copper equivalent (Cu_{eq}) grades:

$$Cu_{eq} = Cu\% + (Au\ g/t*recovery*metal\ price)/((Cu\%*\ recovery*metal\ price)/Cu\%) + (Mo\ \%\ *recovery*\ metal\ price)/((Cu\%*\ recovery*metal\ price)/Cu\%)$$

where Cu revenue is based on variable recovery and $3.50/lb Cu price, Au revenue is based on variable recovery and $1,800/oz Au price, and Mo revenue is based on 40% recovery and $12.00/lb Mo price.

The block model was reblocked into 7.5 mE by 7.5 mN by 15 mRL blocks for mine planning and Mineral Reserve estimation.

The December 2025 Mineral Resource statement was reported within the life of mine pit shell and at a 0.11% Cu_{eq} cut-off grade (Table 14-9). Mineral Resources are inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have a demonstrated economic outcome.

Table 14-9 Taca Taca December 2025 Mineral Resource statement within the life of mine pit shell and using a 0.11% copper equivalent cut-off grade

Classification	Tonnes (mt)	Density (t/m^3)	Cu (%)	Mo (%)	Au (g/t)	CuEq* (%)	Cu Metal (kt)	Mo Metal (kt)	Au metal (koz)
Measured	441	2.67	0.58	0.015	0.13	0.67	2,557	68	1,868
Indicated	1,637	2.65	0.38	0.011	0.07	0.43	6,159	185	3,847
Measured plus Indicated	**2,078**	**2.66**	**0.42**	**0.012**	**0.09**	**0.48**	**8,716**	**253**	**5,715**
Inferred	145	2.66	0.27	0.007	0.06	0.31	389	10	263

* CuEq = Cu% + (Au g/t*recovery*metal price)/((Cu%* recovery*metal price)/Cu%) + (Mo % *recovery* metal price)/((Cu%* recovery*metal price)/Cu%)

The perched gold domain (102) is located within the upper portions of the leached horizon. The reported Mineral Resource excludes this perched gold mineralisation, which will be mined as part of the waste pre-strip and stockpiled separately.

To the best knowledge of the QP, the stated Mineral Resource is not materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other issues that prevent this resource from having reasonable prospects for economic extraction.

14.12 Comparison with previous estimates

There are marginal changes to the updated FQM Mineral Resource statement when compared with the December 2021 Technical Report statement. These changes are primarily associated with the application of a life-of-mine (LOM) pit shell defining the limits for reasonable prospects for economic extraction, together with the application of updated copper-equivalent cut-off grades consistent with those used for Mineral Reserve conversion studies. The previous estimate was not constrained by an equivalent economic pit shell. As a result, Inferred Mineral Resources are now constrained to the LOM pit shell, with the greatest impact occurring where mineralisation extends beneath the Salar de Arizaro. Further technical and economic study would be required to support this mineralisation as a Mineral Resource.

It is the opinion of the QP that the resulting changes to this Mineral Resource estimate reflect RPEE (reasonable prospects of economic extraction), the confidence in the underlying data and that the estimates are representative of the prevailing mineralisation. Grade and geology knowledge has been improved from analysis of sequential copper data thereby enabling improvement to the interpreted base of leached material and the associated impacts on style of copper mineralisation.

ITEM 15 MINERAL RESERVE ESTIMATE

15.1 Introduction

Detailed technical information provided under this item relates specifically to the Mineral Reserve estimate completed for this Technical Report and derived from the Mineral Resource model and Mineral Resource estimate as reported in Item 14.

As part of the estimation process, open pit optimisation aspects and detailed pit designs were completed by FQM personnel overseen and supervised by Michael Lawlor (QP) of FQM. All operating cost, recovery and revenue information for the open pit optimisation, in addition to operational parameters for the open pit designs, were reviewed by Michael Lawlor (QP).

To conform to NI 43-101 standards, the Mineral Reserve estimate is derived from Measured and Indicated Resources only. The Measured and Indicated Mineral Resource estimate as listed in Item 14 is reported inclusive of the Mineral Reserve. Reporting has followed the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy, and Petroleum (the CIM Guidelines, 2014).

15.2 Methodology

The conversion of the Mineral Resource estimate to a Mineral Reserve estimate has followed a conventional approach, commencing with open pit optimisation techniques incorporating economic parameters and other "modifying" factors as described in the following commentary.

Pit optimisation by FQM was completed on an undiscounted cashflow basis, and with recoveries to copper metal (plus molybdenum and gold), in concentrate, determined from process recovery values associated with several metallurgical domains.

A selected ultimate (optimal) pit optimisation shell derived from the use of Whittle Four-X pit optimisation software then guided the creation of practical and detailed open pit stage designs accounting for batters, berms and haul roads.

These pit designs then provided the bench by bench ore and waste mining inventories for the detailed production schedule that demonstrates viable open pit mining. This schedule, which in turn provides the physical basis for 40 Mtpa (Stage 1) cashflow modelling, is described in Item 16.2.5.

15.3 Open pit optimisation

Optimisations (including sensitivity analyses) were completed in late 2024 using a (reblocked) mine planning model derived from the Mineral Resource estimate model described in Item 14. The optimal pit shell was selected on an undiscounted cashflow basis, and with recoveries to copper metal (plus molybdenum and gold) in concentrate.

Consistent with the approach taken in the 2021 Technical Report, the optimisation was constrained by a polygon boundary from encroaching into the mineralisation under the Salar de Arizaro. An additional infrastructure constraint was also applied.

Also consistent with the approach taken in 2021, the optimisations assumed the Stage 2 60 Mtpa production throughput. A retrospective optimisation was completed to test the impact of higher operating costs associated with the Stage 1 40 Mtpa scenario; findings of this analysis are presented in Item 15.4.

15.3.1 Pit slope design criteria

Pit optimisation input included overall slope design angles as indicated in Table 15-1. The geotechnical engineering basis for these design angles, relative to the listed domains, is outlined in Item 16.2.2.

Table 15-1 Pit slope design criteria

Domain	RL range (m) (m)	O'all slope angle (deg)
1	< 3220	36.0
	3220 - 3340	39.0
	> 3340	38.0
2		47.5
3	< 3190	47.0
	3190 - 3370	48.0
	> 3370	45.5
4-1	< 2950	43.0
	2950 - 3250	48.0
	3250 - 3430	47.0
	> 3430	38.0
4-2	< 2860	57.0
	2860 - 3040	44.0
	3040 - 3280	49.0
	3280 - 3460	47.0
	> 3460	39.0
4-3	< 3400	47.5
	> 3400	39.0
5	< 2920	46.0
	2920 - 3070	40.0
	3070 - 3310	50.0
	3310 - 3430	41.0
	> 3430	50.0
6	< 2950	50.0
	2950 - 3250	45.0
	> 3250	45.0
7	< 2875	45.0
	2875 - 2980	37.0
	2980 - 3130	49.0
	3130 - 3250	25.5
	3250 - 3340	49.0
	3340 - 3490	40.5
	> 3490	28.0

15.3.2 Physical optimisation constraints

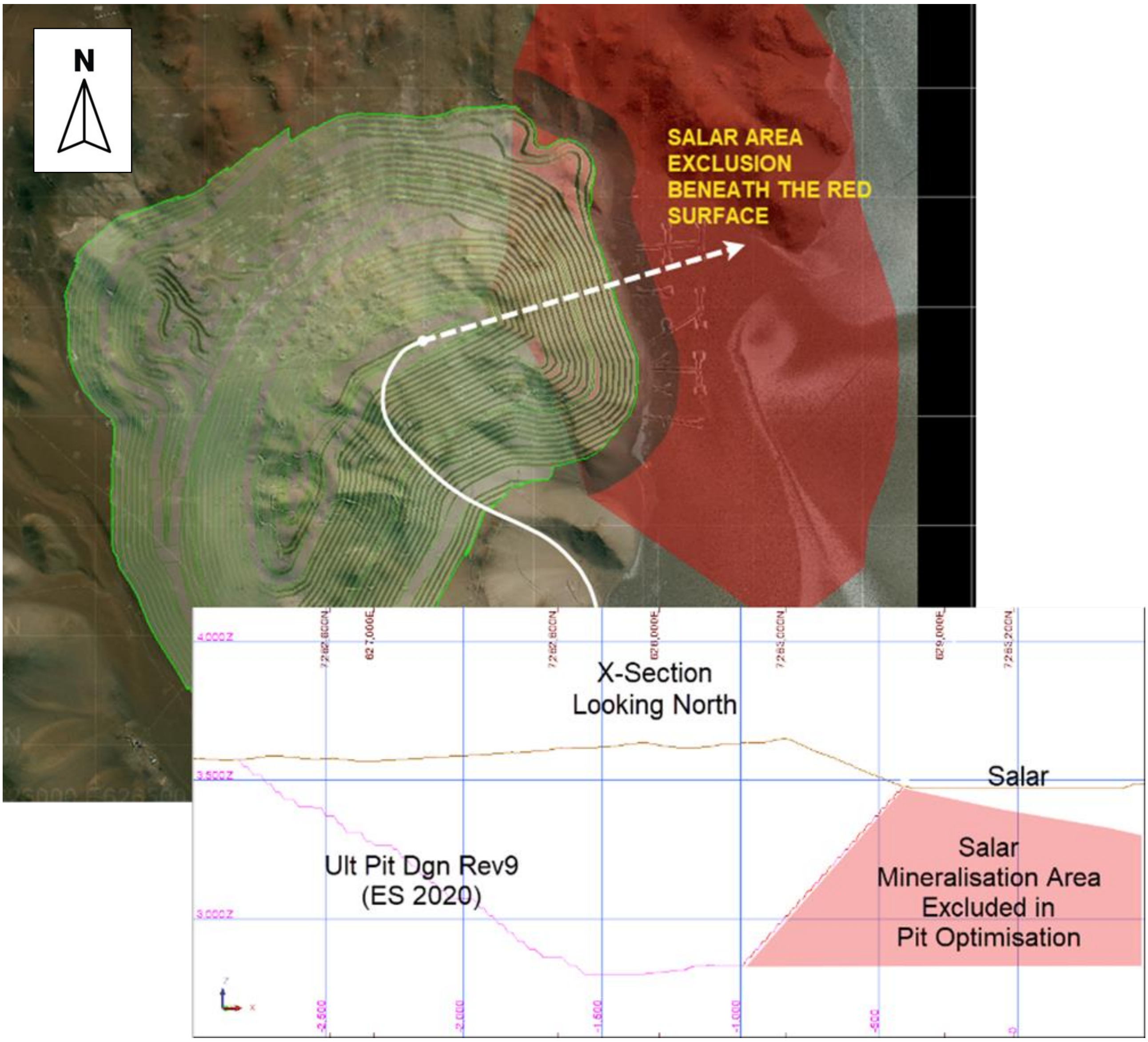
Further to comments made by geotechnical engineers Wyllie and Norrish (W&N, 2016), the optimisation was constrained to prevent pit shells from daylighting into the adjacent salar, with a consequent risk of brine ingress[11]. Figure 15-1 shows the surface projection of this exclusion zone which extends to the base of the pit below the eastern wall.

15.3.3 Mining dilution and mining recovery factors

In the optimisation inputs, "unplanned mining dilution" of 5% (at zero grade) and a mining recovery factor of 95% were included to emulate practical mining losses.

[11] This is a geotechnical aspect that will be addressed further as the Project engineering phase proceeds. A planned geotechnical drilling and investigation programme is proposed to address the proximity of the pit eastern wall to the Salar de Arizaro. This investigation commencing in Q2 2026 will be completed in conjunction with hydrogeological modelling of the salar and the brine draw down response to open pit mining.

Figure 15-1 Pit optimisation, salar incursion constraint



15.3.4 Metal prices

From the 2021 Technical Report the adopted longer term metal prices were:

- changed from \$3.00/lb Cu to \$3.50/lb Cu
- changed from \$1,200/oz Au to \$1,800/oz Au
- molybdenum price was unchanged at \$12.00/lb.

The molybdenum price was unchanged at \$12.00/lb.

15.3.5 Metal recoveries

The changed water balance no longer relies on brine water make-up in the rougher flotation process, and hence the metallurgical recovery projections for all metals have changed.

To reflect the increased consumption of fresh water during rougher and cleaner processing, the updated process recovery projections were as listed in Table 15-2. Note that the recovery values for molybdenum are fixed values, irrespective of MATTYPE.

Table 15-2 Updated process recovery projections

Geological Process	Copper species	Relative proportion	% of Total Ore	Domain (DOM) numeric value	Material type (MATTYPE) Name	% Acid Soluble Cu	Cu%	Pyrite	Fe%	Met domain	Cu Recovery, %	Cu Con Grade, % Cu	Au Recovery, % (ALS testwork)	Supporting Met Testwork (ALS)	Supporting Met Testwork (Plenge 2012 Variabilty Tests)
Supergene	Chalcocite, chalcopyrite, black Cu oxides, soluble copper	CC>CPY + Sol Cu	3%	203	MIXED	>10	<0.4	low	<1.5	C2	65	15			
mixed				203	MIXED	>10	<0.4	medium	1.5-2.4	C2	65	15			
oxide				203	MIXED	>10	<0.4	high	>2.4	C2	65	15			
				203	MIXED	>10	>0.4	low	<1.5	C2	74	20			
				203	MIXED	>10	>0.4	medium	1.5-2.4	C2	74	20			
				203	MIXED	>10	>0.4	high	>2.4	C2	70	20			3 Samples
Supergene	Chalcocite, chalcopyrite	CC>>CPY	19%	304, 305, 310	SECONDARY	>10	<0.4	low	<1.5	C5	85	25			2 samples
mixed				304, 305, 310	SECONDARY	>10	<0.4	medium	1.5-2.4	C5	83	25			
				304, 305, 310	SECONDARY	>10	<0.4	high	>2.4	C5	80	22	32	Met 005	
				304, 305, 310	SECONDARY	>10	0.4-1	low	<1.5	C6	88	34	55	Met 001	2 samples
				304, 305, 310	SECONDARY	>10	0.4-1	medium	1.5-2.4	C5	88	34	56	Met 008, 009	1 sample
				304, 305, 310	SECONDARY	>10	0.4-1	high	>2.4	C5	86	25			1 sample
	Chalcocite, Chalcopyrite, bornite			304, 305, 310	SECONDARY	>10	>1	low	<1.5	C6	88	35	47	Met 002, 006	5 samples
				304, 305, 310	SECONDARY	>10	>1	medium	1.5-2.4	C5	88	32		Met 003	4 samples
				304, 305, 310	SECONDARY	>10	>1	high	>2.4	C5	86	30			
Supergene	Chalcocite, chalcopyrite	CC>CPY	59%	307,308, 309	SUPERGENE MIXED	~10	<0.4	low	<1.5	C7	88	27			
transition				307,308, 309	SUPERGENE MIXED	~10	<0.4	medium	1.5-2.4	C7	86	25			1 Sample
				307,308, 309	SUPERGENE MIXED	~10	<0.4	high	>2.4	C7	84	22			2 samples
	Chalcocite, chalcopyrite, bornite			307,308, 309	SUPERGENE MIXED	~10	>0.4	low	<1.5	C1	90	30			2 samples
				307,308, 309	SUPERGENE MIXED	~10	>0.4	medium	1.5-2.4	C1	90	30			
				307,308, 309	SUPERGENE MIXED	~10	>0.4	high	>2.4	C7	88	26			2 samples
Hypogene	Chalcopyrite, chalcocite	CPY>CC	20%	306	PRIMARY	<10	<0.4	low	<1.5	C3	90	25	74	Met 010	2 samples
				306	PRIMARY	<10	<0.4	medium	1.5-2.4	C3	90	25			4 samples
				306	PRIMARY	<10	<0.4	high	>2.4	C3	88	22			2 samples
	Chalcopyrite, chalcocite, bornite			306	PRIMARY	<10	>0.4	low	<1.5	C4	90	28			3 samples
				306	PRIMARY	<10	>0.4	medium	1.5-2.4	C4	90	28	74	Met 007	5 samples
				306	PRIMARY	<10	>0.4	high	>2.4	C4	90	26	78	Met 004	1 sample

Table 15-3 lists the overall average process recovery projections evident in the mine planning model as adopted for pit optimisation.

Table 15-3 Pit optimisation input, average process recovery projections

	Cu rec (%)	Mo rec (%)	Au rec (%)
Secondary	83.2%	40.0%	41.6%
Mixed	88.3%	40.0%	55.0%
Primary	89.5%	60.0%	66.6%

15.3.6 Mining costs

The estimated incremental mining costs for pit optimisation were updated after the 2021 Technical Report estimate (Table 15-4).

The estimate was completed in detail, by means of:

- updated ore and waste haul profiles (distance and travel time), relative to preliminary staged pit designs, waste dump and stockpile layouts
- updated mining equipment productivity parameters
- updated diesel fuel and electrical power consumption rates
- tyre consumption rates
- updated mining labour rates

The overall average mining cost estimated for pit optimisation, for ore and waste, is $2.00/t mined.

15.3.7 Operating and other costs

The process plant operating costs adopted for the updated 60 Mtpa pit optimisation were the same as those derived for cashflow modelling in 2021 (i.e., as distinct from the costs used for the preceding optimisation). This information is listed in Table 15-5.

In addition to the mining and plant operating costs, the following optimisation inputs were adopted:

- general and administration charges = $1.05/t ore
- water supply tariff = $0.02/t ore
- rail infrastructure maintenance = $0.06/t ore

No allowance for sustaining costs (as equivalent operating costs) was made at the time of pit optimisation. With a production and equipment replacement schedule subsequently available, sustaining cost provisions were included in the economic analysis (Item 22).

15.3.8 Metal costs

Metal costs in the terminology of pit optimisation refer to concentrate treatment charges, refining charges, concentrate freight charges and royalties. In this instance, the metal costs adopted for pit optimisation were the same as those derived for cashflow modelling in 2021 (i.e., as distinct from the costs used for the preceding optimisation). This information is listed in Table 15-6.

The net return information listing copper, molybdenum and gold on a copper equivalent return basis, is listed in Table 15-7.

Table 15-4 Pit optimisation input, updated incremental mining costs

	ORE			WST		
Bench	Ore (Direct) ($/t)	Ore (O'head) ($/t)	Nov. 2024 Opt. Input ($/t)	Waste (Direct) ($/t)	Waste (O'head) ($/t)	Nov. 2024 Opt. Input ($/t)
3705	$1.94	$0.22	$2.16	$2.42	$0.22	$2.65
3690	$1.94	$0.22	$2.16	$2.50	$0.22	$2.73
3675	$1.94	$0.22	$2.16	$2.28	$0.22	$2.51
3660	$1.94	$0.22	$2.16	$2.25	$0.22	$2.47
3645	$1.91	$0.22	$2.13	$2.26	$0.22	$2.48
3630	$1.92	$0.22	$2.14	$2.14	$0.22	$2.36
3615	$2.15	$0.22	$2.37	$2.07	$0.22	$2.29
3600	$2.18	$0.22	$2.40	$2.04	$0.22	$2.26
3585	$2.18	$0.22	$2.40	$1.94	$0.22	$2.16
3570	$1.96	$0.22	$2.18	$1.92	$0.22	$2.14
3555	$1.86	$0.22	$2.09	$1.89	$0.22	$2.11
3540	$1.53	$0.22	$1.75	$1.40	$0.22	$1.62
3525	$1.50	$0.22	$1.72	$1.41	$0.22	$1.64
3510	$1.50	$0.22	$1.72	$1.39	$0.22	$1.62
3495	$1.54	$0.22	$1.76	$1.46	$0.22	$1.68
3480	$1.59	$0.22	$1.81	$1.46	$0.22	$1.69
3465	$1.64	$0.22	$1.86	$1.48	$0.22	$1.71
3450	$1.67	$0.22	$1.89	$1.49	$0.22	$1.71
3435	$1.67	$0.22	$1.90	$1.52	$0.22	$1.74
3420	$1.73	$0.22	$1.95	$1.57	$0.22	$1.79
3405	$1.76	$0.22	$1.99	$1.61	$0.22	$1.83
3390	$1.78	$0.22	$2.00	$1.68	$0.22	$1.90
3375	$1.75	$0.22	$1.97	$1.70	$0.22	$1.93
3360	$1.72	$0.22	$1.94	$1.76	$0.22	$1.98
3345	$1.75	$0.22	$1.97	$1.76	$0.22	$1.99
3330	$1.67	$0.22	$1.89	$1.66	$0.22	$1.88
3315	$1.74	$0.22	$1.96	$1.69	$0.22	$1.91
3300	$1.68	$0.22	$1.90	$1.68	$0.22	$1.90
3285	$1.72	$0.22	$1.95	$1.67	$0.22	$1.89
3270	$1.72	$0.22	$1.95	$1.68	$0.22	$1.90
3255	$1.74	$0.22	$1.97	$1.70	$0.22	$1.92
3240	$1.75	$0.22	$1.97	$1.73	$0.22	$1.96
3225	$1.79	$0.22	$2.02	$1.77	$0.22	$1.99
3210	$1.84	$0.22	$2.07	$1.80	$0.22	$2.02
3195	$1.88	$0.22	$2.10	$1.85	$0.22	$2.07
3180	$1.91	$0.22	$2.13	$1.88	$0.22	$2.10
3165	$1.92	$0.22	$2.15	$1.91	$0.22	$2.14
3150	$1.98	$0.22	$2.20	$1.94	$0.22	$2.17
3135	$2.02	$0.22	$2.25	$1.98	$0.22	$2.20
3120	$2.05	$0.22	$2.28	$2.05	$0.22	$2.27
3105	$2.07	$0.22	$2.30	$2.12	$0.22	$2.34
3090	$2.10	$0.22	$2.33	$2.13	$0.22	$2.36
3075	$2.16	$0.22	$2.38	$2.14	$0.22	$2.36
3060	$2.19	$0.22	$2.41	$2.17	$0.22	$2.39
3045	$2.21	$0.22	$2.44	$2.20	$0.22	$2.42
3030	$2.24	$0.22	$2.46	$2.32	$0.22	$2.54
3015	$2.24	$0.22	$2.46	$2.33	$0.22	$2.56
3000	$2.28	$0.22	$2.50	$2.37	$0.22	$2.59
2985	$2.32	$0.22	$2.54	$2.41	$0.22	$2.63
2970	$2.34	$0.22	$2.57	$2.43	$0.22	$2.65
2955	$2.38	$0.22	$2.60	$2.46	$0.22	$2.68
2940	$2.39	$0.22	$2.61	$2.49	$0.22	$2.72
2925	$2.39	$0.22	$2.61	$2.53	$0.22	$2.75
2910	$2.42	$0.22	$2.64	$2.56	$0.22	$2.78
2895	$2.45	$0.22	$2.67	$2.59	$0.22	$2.81
2880	$2.50	$0.22	$2.72	$2.63	$0.22	$2.86
2865	$2.53	$0.22	$2.75	$2.66	$0.22	$2.89
2850	$2.58	$0.22	$2.81	$2.73	$0.22	$2.96
2835	$2.67	$0.22	$2.89	$3.20	$0.22	$3.42
2820	$2.71	$0.22	$2.93	$3.23	$0.22	$3.46
2805	$2.74	$0.22	$2.96	$3.21	$0.22	$3.43
			$2.13			**$1.89**

Table 15-5 Pit optimisation input, processing costs

	60 Mtpa, 7500 tph	
	$pa	$/t
Mill and crusher liners	$16,485,240	$0.275
Grinding media	$52,172,400	$0.870
Reagents	$57,042,549	$0.951
Miscellaneous	$1,152,948	$0.019
Subtotal consumables	**$126,853,137**	**$2.114**
Energy	$94,380,000	$1.573
Maintenance	$51,673,065	$0.861
Process labour	$5,795,300	$0.097
Engineering labour	$2,724,500	$0.045
Concentrate transport	n/a	n/a
Total unit processing cost	**$281,426,002**	**$4.690**

Table 15-6 Pit optimisation input, metal costs

Metal Costs	Units	Primary	Secondary	Mixed
Copper concentrate charges:				
Copper grade (in ore)	%	0.45	0.45	0.45
Copper payable rate	%	96.2%	96.2%	96.2%
Cu concentrate grade	%	25.4%	25.4%	25.4%
Moisture content	%	10%	10%	10%
Transport and freight charges:		$0.00	$0.00	$0.00
Concentrate rail transport	$/wmt	$39.50	$39.50	$39.50
Port charges	$/wmt	$7.00	$7.00	$7.00
Sea freight charges	$/wmt	$48.50	$48.50	$48.50
subtotal	**$/wmt**	**$95.00**	**$95.00**	**$95.00**
subtotal	**$/dmt**	**$105.55**	**$105.55**	**$105.55**
Copper treatment charge	$/dmt	$90.00	$90.00	$90.00
Cu refining (on payable)	$/lb	$0.09	$0.09	$0.09
Copper metal cost	**$/lb**	**$0.45**	**$0.45**	**$0.45**
Molybdenum concentrate charges:				
Molybdenum grade (in ore)	%	0.012	0.012	0.012
Molybdenum payable rate	%	86.0%	86.0%	86.0%
Molybdenum concentrate grade	%	47.0%	47.0%	47.0%
Moisture content	%	10%	10%	10%
Transport and freight charges:				
Concentrate rail transport	$/wmt	$48.00	$48.00	$48.00
Port charges	$/wmt	$7.00	$7.00	$7.00
Sea freight charges	$/wmt	$48.50	$48.50	$48.50
subtotal	**$/wmt**	**$103.50**	**$103.50**	**$103.50**
subtotal	**$/dmt**	**$115.00**	**$115.00**	**$115.00**
Molybdenum treatment charge	$/dmt	$68.19	$68.19	$68.19
Molybdenum refining (on payable)	$/lb	$0.00	$0.00	$0.00
Molybdenum metal cost	**$/lb**	**$0.21**	**$0.21**	**$0.21**
Gold charges:				
Gold grade (in ore)	g/t Au	0.092	0.092	0.092
Gold grade (in ore)	%	0.000	0.000	0.000
Gold payable rate	%	90.0%	90.0%	90.0%
Gold refining (on payable)	$/oz	$5.10	$5.10	$5.10
Gold metal cost	$/oz	**$4.59**	**$4.59**	**$4.59**
Royalty charges:				
Copper	$/lb	$0.16	$0.16	$0.16
Molybdenum	$/lb	$0.54	$0.54	$0.54
Gold	$/oz	$81.00	$81.00	$81.00
Total metal costs:				
Total copper metal cost	**$/lb**	$0.61	$0.61	$0.61
Total molybdenum metal cost	**$/lb**	$0.75	$0.75	$0.75
Total gold metal cost	**$/oz**	$85.59	$85.59	$85.59
Total metal costs as %age of price:				
Copper metal cost	%	17.4%	17.4%	17.4%
Molybdenum metal cost	%	6.2%	6.2%	6.2%
Gold metal cost	%	4.8%	4.8%	4.8%

Table 15-7 Pit optimisation input, net return

Net Return	Units	Primary	Secondary	Mixed
Processing Parameters:		av. recovery revenue factor 1 pit shell		
Cu recovery	%	89.50%	83.15%	88.34%
Mo recovery (thru Mo con)	%	60.00%	40.00%	40.00%
Au recovery	%	66.60%	41.55%	55.00%
Net return				
Copper metal cost	$/lb	$0.61	$0.61	$0.61
Copper net return	$/lb	$2.89	$2.89	$2.89
Copper net return (recovered)	$/lb	$2.59	$2.40	$2.55
Molybdenum metal cost	$/lb	$0.75	$0.75	$0.75
Molybdenum net return	$/lb	$11.25	$11.25	$11.25
Cueq net return (recovered)	$/lb	$0.07	$0.06	$0.06
Gold metal cost	$/oz	$85.59	$85.59	$85.59
Gold net return	$/oz	$1,714.41	$1,714.41	$1,714.41
Cueq net return (recovered)	$/lb	$0.25	$0.11	$0.18
Total Net Return (recovered)	**$/lb**	$2.91	$2.57	$2.78
Total Net Return (recovered)	**$/10kg**	**$64.16**	**$56.63**	**$61.38**

15.3.9 Marginal cut-off grades

The updated marginal cut-off grades, emulating calculations performed within the Whittle optimisation process, are listed in Table 15-8.

Table 15-8 Overall average marginal cut-off grades

Marginal cut-off grade	Units	Primary	Secondary	Mixed
PROCOST	$/t ore	$5.82	$5.82	$5.82
MINDIL		1.05	1.05	1.05
TOTAL NET RETURN (recovered)	$/10kg	$64.16	$56.63	$61.38
C/O equivalent grade	**%Cu**	**0.10**	**0.11**	**0.10**

15.3.10 Optimisation results

The pit optimisation results are shown in Figure 15-2 and listed in Table 15-9. The selected optimal pit shell is at revenue factor 1.0, yielding a total pit size of 4,865 Mt, plant feed of 2,040 Mt at 0.41%Cu and 2,825 Mt of waste (at a strip ratio of 1.4 : 1). Pit shells beyond revenue factor 1.0 offer no better undiscounted cashflow.

The undiscounted operating cashflow from the revenue factor 1.00 pit shell (no. 14) is $29.7B.

Comparison against the 2021 Technical Report optimisation results

The pit optimisation results reflect several input changes relative to those adopted for the optimisation reported in 2021, specifically:

- a Mineral Resource model update
- revised processing recovery values
- increased metal prices
- revised pit slope design criteria
- revised mining, processing and G&A costs
- changed infrastructure confines

The impact of these changes is shown in a series of waterfall charts, in Figure 15-3 to Figure 15-6.

Table 15-9 Summary of optimisation results

Pit Shell	Revenue Factor x $3.5/lb Cu x 96.2% payabiity	Pit Size Mt	Waste Mt	SR waste/ore	Total Mt Plant Feed	Mt Plant Feed		Mt Plant Feed			Plant Feed Grade (diluted)			Recovered Metal		
						Measured	Indicated	Primary	Secondary	Mixed	%Cu	gr/t Au	ppm Mo	Mt Cu	ktroy oz Au	kt Mo
1	0.35	640.8	446.2	2.3	194.7	115.3	79.3	25.5	0.0	169.2	0.74	0.13	138.93	1.27	471.1	11.6
2	0.4	1,096.9	743.2	2.1	353.7	192.0	161.7	71.6	0.3	281.8	0.66	0.13	159.47	2.06	855.8	24.9
3	0.45	1,892.9	1,202.8	1.7	690.1	344.1	345.9	153.0	16.3	520.8	0.55	0.12	147.79	3.39	1,490.2	45.6
4	0.5	2,937.0	1,751.7	1.5	1,185.3	408.8	776.6	238.4	139.4	807.5	0.48	0.10	136.46	4.98	2,228.6	71.6
5	0.55	3,496.9	2,039.2	1.4	1,457.7	425.9	1031.8	310.6	196.5	950.6	0.45	0.10	130.61	5.78	2,543.3	84.7
6	0.6	4,100.5	2,381.3	1.4	1,719.2	432.2	1286.9	366.5	288.7	1,064.0	0.43	0.09	122.56	6.54	2,782.9	93.9
7	0.65	4,350.0	2,515.4	1.4	1,834.6	432.2	1402.4	396.7	316.1	1,121.7	0.42	0.09	120.68	6.83	2,886.9	98.7
8	0.7	4,469.1	2,579.1	1.4	1,890.1	432.2	1457.8	409.9	333.3	1,146.9	0.42	0.09	119.58	6.96	2,928.5	100.8
9	0.75	4,599.4	2,654.4	1.4	1,945.0	432.2	1512.8	424.3	342.0	1,178.7	0.41	0.09	119.06	7.08	2,971.4	103.3
10	0.8	4,675.3	2,701.1	1.4	1,974.2	432.2	1541.9	429.9	351.0	1,193.2	0.41	0.08	118.65	7.14	2,990.5	104.5
11	0.85	4,769.5	2,763.3	1.4	2,006.2	432.2	1574.0	442.3	356.1	1,207.8	0.41	0.08	118.21	7.21	3,016.2	105.9
12	0.9	4,800.8	2,782.4	1.4	2,018.4	432.2	1586.2	442.8	359.3	1,216.3	0.41	0.08	118.13	7.23	3,022.9	106.4
13	0.95	4,845.5	2,811.9	1.4	2,033.6	432.2	1601.4	445.7	360.1	1,227.7	0.41	0.08	118.04	7.26	3,032.7	107.1
14	1	4,864.8	2,824.5	1.4	2,040.4	432.2	1608.1	447.3	363.8	1,229.3	0.406	0.08	117.84	7.27	3,036.5	107.3
15	1.05	4,891.4	2,842.5	1.4	2,048.9	432.2	1616.7	448.7	365.3	1,234.9	0.41	0.08	117.73	7.29	3,040.4	107.6
16	1.1	4,908.3	2,855.6	1.4	2,052.7	432.2	1620.5	450.6	366.7	1,235.4	0.41	0.08	117.63	7.30	3,043.2	107.8
17	1.15	4,938.7	2,879.0	1.4	2,059.7	432.2	1627.4	452.6	368.1	1,238.9	0.40	0.08	117.57	7.31	3,047.7	108.1
18	1.2	4,951.3	2,888.2	1.4	2,063.1	432.2	1630.8	454.0	368.9	1,240.1	0.40	0.08	117.48	7.31	3,049.9	108.2
19	1.25	4,967.2	2,901.3	1.4	2,065.9	432.2	1633.7	455.3	369.8	1,240.8	0.40	0.08	117.43	7.32	3,052.6	108.3
20	1.3	4,983.8	2,914.6	1.4	2,069.2	432.2	1637.0	455.8	370.0	1,243.4	0.40	0.08	117.42	7.33	3,054.2	108.5
21	1.35	5,001.0	2,928.5	1.4	2,072.5	432.2	1640.3	456.2	370.7	1,245.6	0.40	0.08	117.39	7.33	3,056.1	108.6
22	1.4	5,005.7	2,932.2	1.4	2,073.5	432.2	1641.3	456.5	371.1	1,245.9	0.40	0.08	117.37	7.33	3,056.6	108.6
23	1.45	5,013.9	2,939.5	1.4	2,074.4	432.2	1642.2	457.1	371.2	1,246.1	0.40	0.08	117.36	7.33	3,057.5	108.7
24	1.5	5,028.2	2,951.4	1.4	2,076.8	432.2	1644.5	457.7	372.0	1,247.1	0.40	0.08	117.33	7.34	3,058.7	108.8
25	1.55	5,037.1	2,959.3	1.4	2,077.9	432.2	1645.7	458.3	372.1	1,247.4	0.40	0.08	117.31	7.34	3,059.6	108.8
26	1.6	5,042.3	2,963.5	1.4	2,078.8	432.2	1646.6	458.4	372.4	1,248.0	0.40	0.08	117.30	7.34	3,059.9	108.9
27	1.65	5,047.4	2,967.8	1.4	2,079.6	432.2	1647.4	458.6	372.8	1,248.2	0.40	0.08	117.27	7.34	3,060.3	108.9
28	1.7	5,056.4	2,975.7	1.4	2,080.7	432.2	1648.5	458.7	373.0	1,249.1	0.40	0.08	117.26	7.34	3,060.7	108.9
29	1.75	5,060.9	2,979.7	1.4	2,081.2	432.2	1649.0	458.8	373.1	1,249.3	0.40	0.08	117.26	7.35	3,061.1	109.0
30	1.8	5,069.6	2,987.2	1.4	2,082.4	432.2	1650.2	458.9	373.2	1,250.2	0.40	0.08	117.25	7.35	3,061.5	109.0
31	1.85	5,076.1	2,993.2	1.4	2,082.9	432.2	1650.7	459.1	373.5	1,250.4	0.40	0.08	117.23	7.35	3,061.8	109.0
32	1.9	5,078.1	2,994.9	1.4	2,083.2	432.2	1650.9	459.1	373.6	1,250.4	0.40	0.08	117.22	7.35	3,061.9	109.0
33	1.95	5,080.5	2,997.2	1.4	2,083.4	432.2	1651.1	459.2	373.7	1,250.4	0.40	0.08	117.21	7.35	3,062.0	109.0
34	2	5,088.9	3,004.6	1.4	2,084.3	432.2	1652.1	459.6	373.8	1,251.0	0.40	0.08	117.20	7.35	3,062.6	109.1

Figure 15-2 Pit optimisation, graphical result

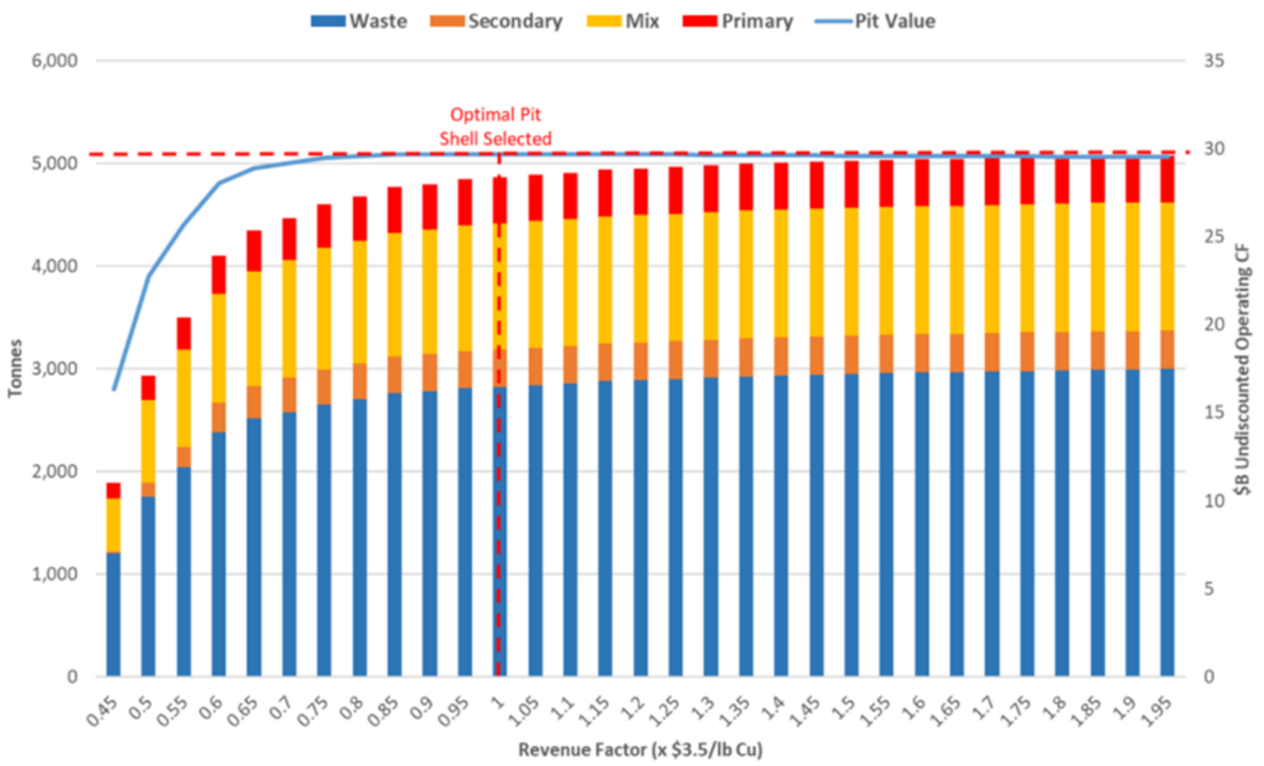


Figure 15-3 Pit optimisation; impact of changed inputs upon total pit size

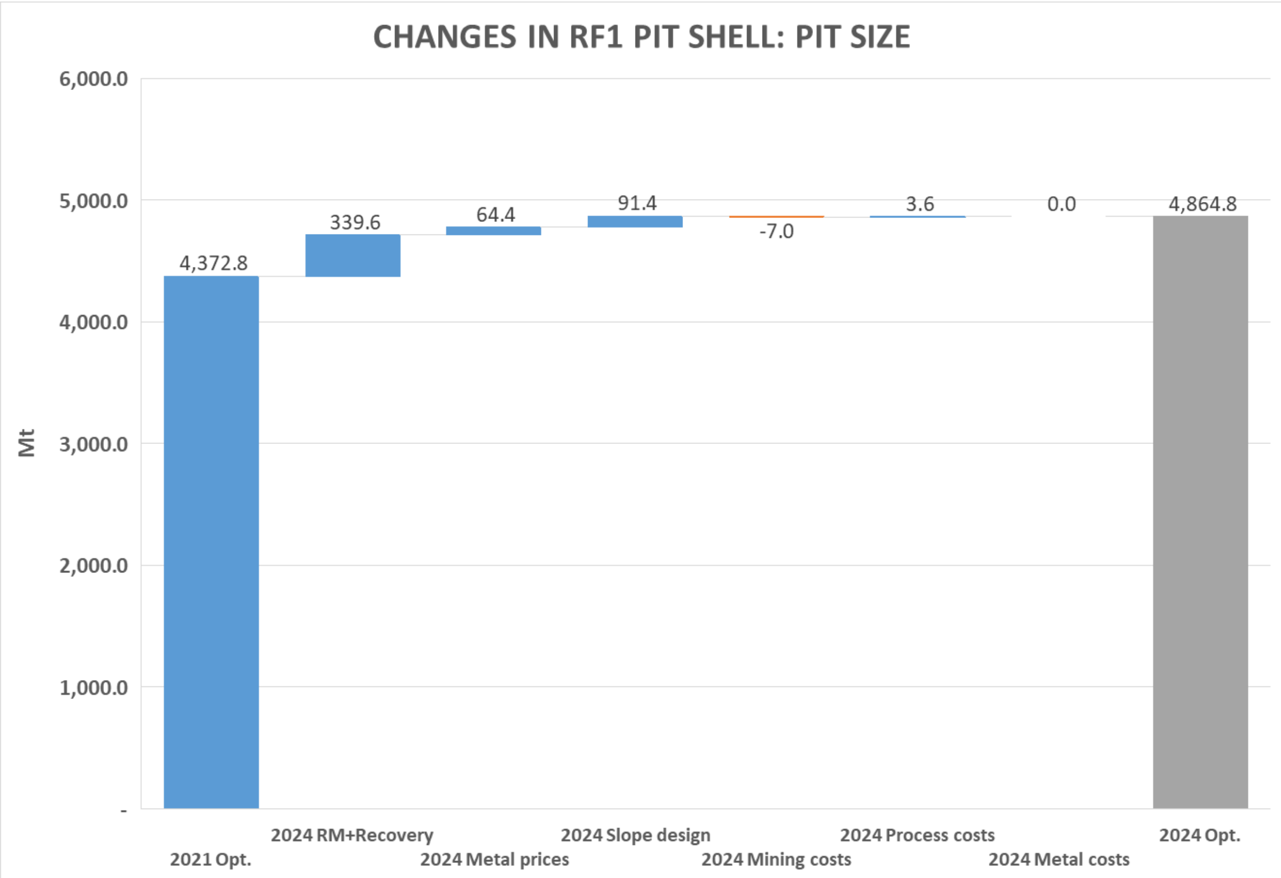


Figure 15-4 Pit optimisation; impact of changed inputs upon LOM plant feed

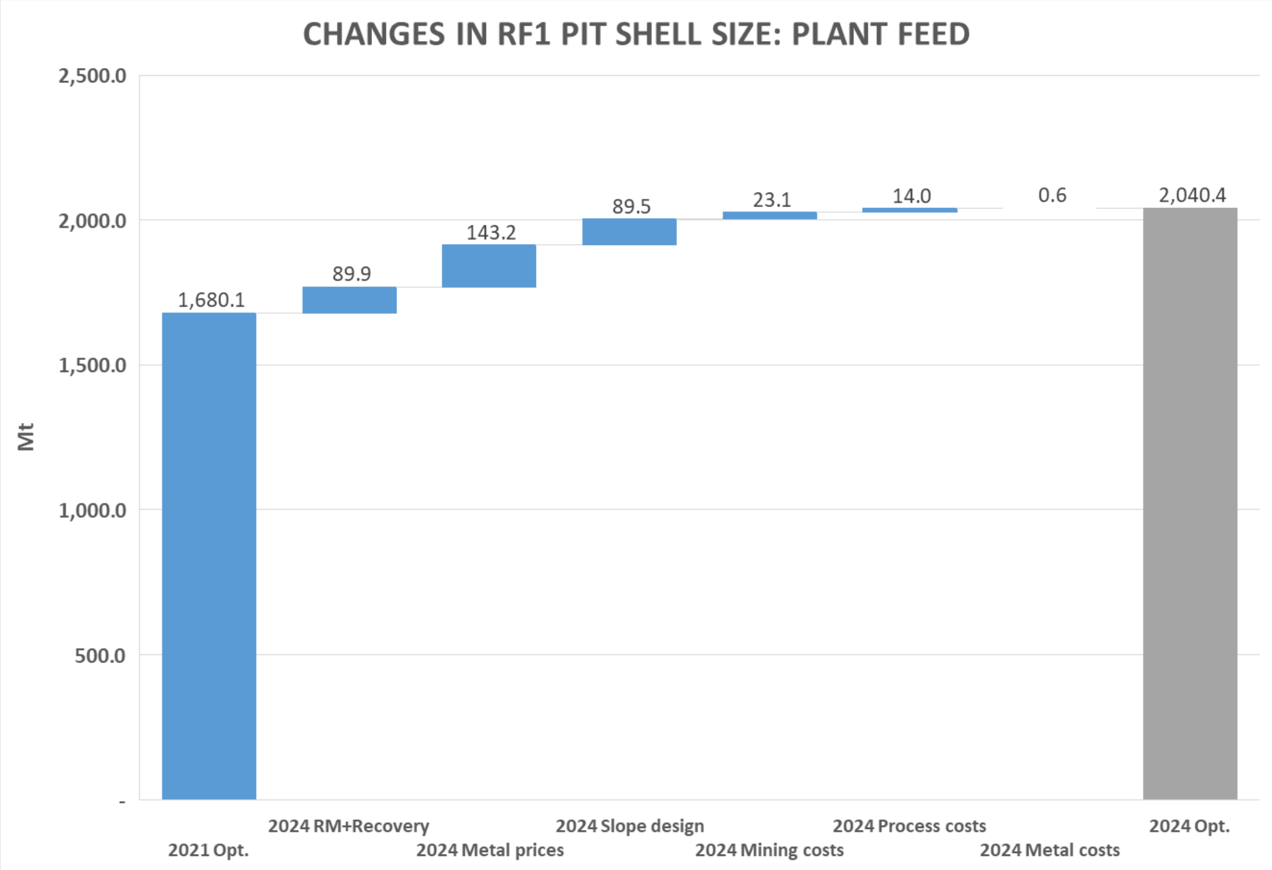


Figure 15-5 Pit optimisation; impact of changed inputs upon recovered copper

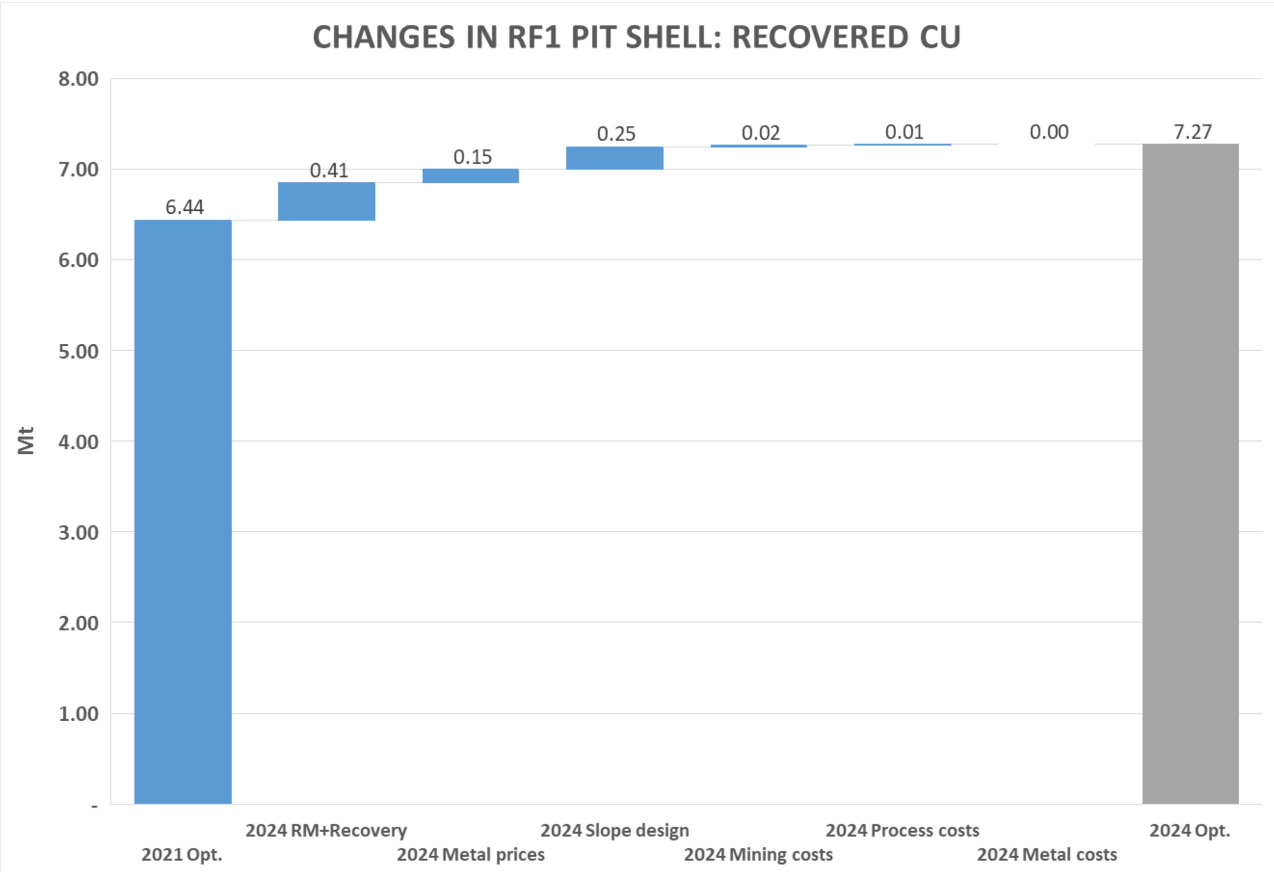
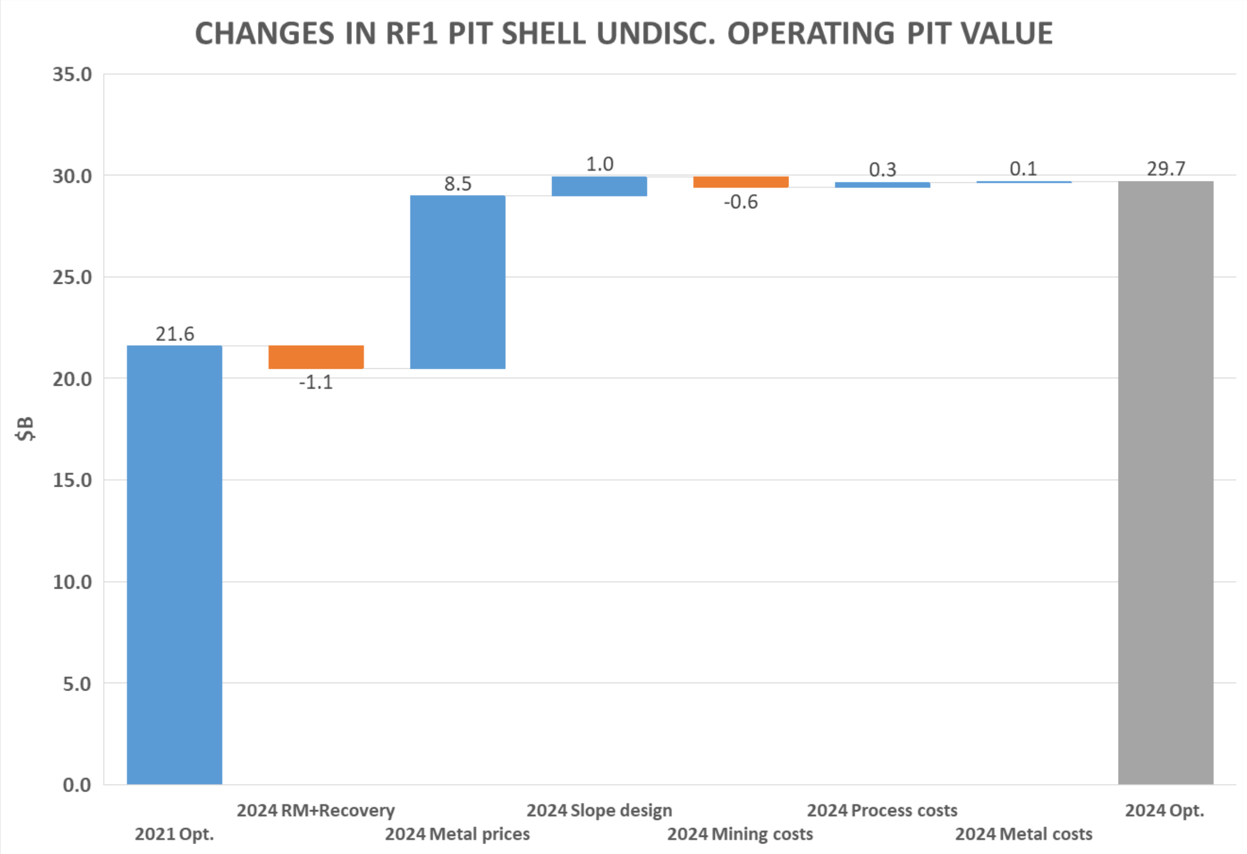


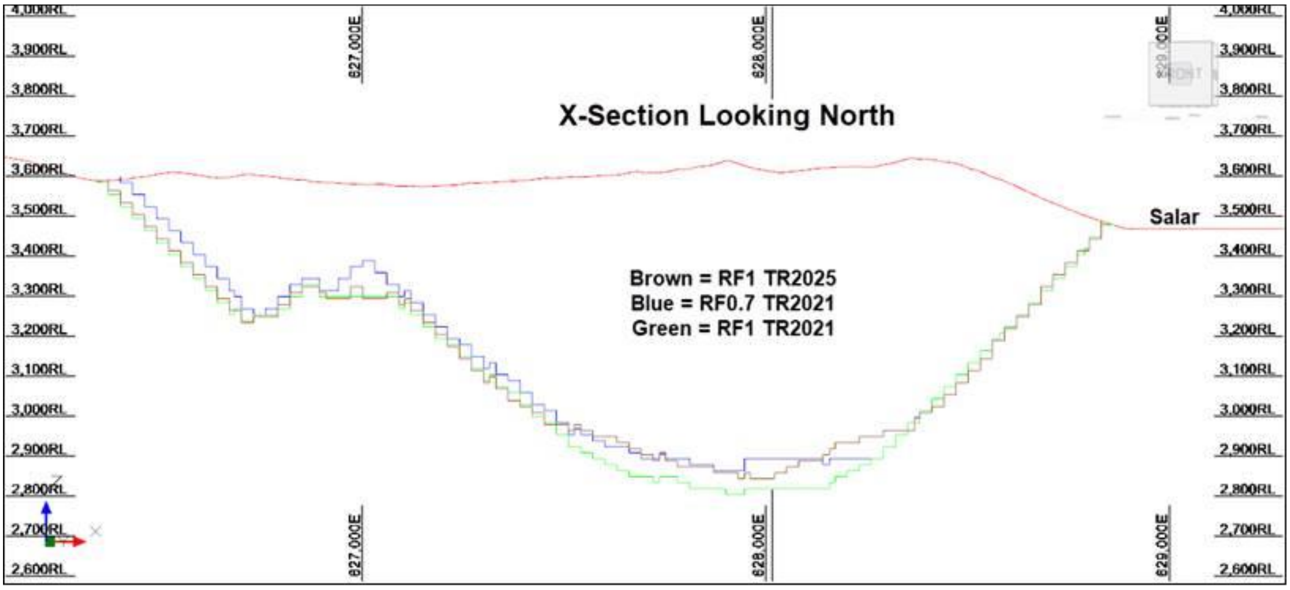
Figure 15-6 Pit optimisation; impact of changed inputs upon pit value (undiscounted LOM cashflow)



Comments on these waterfall charts, when comparing the latest optimisation outcomes relative to those reported in 2021, are as follows:

- the selected optimal pit shell size is 11% larger, impacted most significantly by the new Resource model and the improved processing recovery coded into the model. This can be appreciated in Figure 15-7 which shows optimisation pit shells in cross section where;
  - the brown line is the current selected revenue factor 1 shell
  - the blue line is the revenue factor 0.7 shell selected for the 2021 Technical Report
- the plant feed tonnage is 21% higher, impacted most significantly by the increased metal prices and revised slope design parameters and the model/recovery updates contribute to the improvement
- the recovered copper metal is 13% higher, impacted most significantly by the revised slope design parameters and the model/recovery updates
- the undiscounted value (revenue and operating/metal costs basis) is 38% higher, primarily due to the higher metal prices (i.e., $3.50/lb Cu *vs* $3.00/lb Cu, and $1,800/oz Au *vs* $1,200/oz Au)

Figure 15-7 Pit shell comparisons, cross section looking north



15.4 Optimisation sensitivity analyses

15.4.1 Generic analyses

Table 15-10 summarises the results of generic 60 Mtpa optimisation sensitivity analyses, assessing the impact of:

- varying the copper metal price by +/- $1.00/lb
- varying processing recovery, mining and operating costs, and metal costs by +/- 5%

This table shows that the primary enhancements to inventory arise from grade related aspects such as recovery and metal price. Cost increases up to plus or minus 5% have a minor impact on the selected optimal shell.

In relation to the most sensitive variable, i.e. the copper metal price, Table 15-10 figures indicate that a copper price of $4.50/lb (in line with recent trends since late 2024) could yield an additional 235 Mt of potential plant feed at a grade of 0.18%Cu.

15.4.2 Retrospective analyses

Between the times of completing the pit optimisation and subsequently carrying out the economic analysis described in Item 22, metal price projections and operating cost estimates were revised. A case in point is that the (lower) operating costs in the optimisation reflected a Stage 2 60 Mtpa processing scenario (rather than a Stage 1 40 Mtpa cashflow model scenario). Consequently, additional optimisation sensitivity analyses were carried out to test the combined impact of higher metal prices and the higher operating costs associated with 40 Mtpa processing. Table 15-11 lists the original 60 Mtpa optimisation inputs and those for the additional analyses. Specifically:

- the copper metal price for the original optimisation was $3.50/lb, whereas in the economic analysis it is a LOM fixed price of $4.50/lb
- the molybdenum metal price at optimisation was $12.00/lb, whereas in the economic analysis it is a LOM fixed price of $18.00/lb
- the gold metal price at optimisation was $1,800/oz, whereas in the economic analysis it is a LOM fixed price of $3,000/oz

- mining, processing and G&A (at 40 Mtpa) unit costs were re-estimated post optimisation according to the commentary in 20.7; further adjustments were also made in the cashflow modelling to reflect the inclusion of certain indirect costs

In Table 15-11 there is also a listing of estimated unit processing and G&A costs applicable to 60 Mtpa processing. These inputs (along with the higher metal prices) were tested in another sensitivity analysis to gauge the impact relative to the original pit shell selection.

Table 15-12 summarises the compounded impact of these additional optimisation input differences. In the first instance, when comparing the 60 Mtpa original optimisation with the results of the optimisation scenarios at 40 Mtpa reflecting cashflow model related inputs:

- the overall pit size is almost unaffected by the combined effects of the higher metal prices and higher operating costs
- correspondingly, there is a ~2% reduction in the plant feed tonnage and a negligible reduction in the recovered copper metal
- the higher metal prices have a significant positive impact on the relative undiscounted cashflow value

In the second instance, when comparing the 60 Mtpa original optimisation with the results of optimisation scenarios at 60 Mtpa reflecting relative cashflow model related inputs, the outcomes are similar to the above.

An overall conclusion from the results listed in Table 15-12 is that, when comparing the retrospective optimisation results against the original 60 Mtpa optimisation shell, there is no basis for a redesign of the ultimate pit.

In this instance, when considering the elapsed time between optimisation and economic analysis, the higher metal prices appear to compensate for the higher operating costs.

As an aside to the above commentary, considerable upside is indicated when constraints are relaxed allowing the optimisation to enclose mineralisation under the base of the salar. This result underscores a recommendation for future drilling programme(s) beyond the currently constrained eastern limits of the proposed open pit.

Table 15-10 Summary results for a generic optimisation sensitivity analysis

Sensitivity Scenario	Pit Size	Waste	Strip	Plant Feed			Recovery	Rec'd Metal	Undisc. Value	% changes relative to Base Case		
	(Mt)	(Mt)	Ratio	(Mt)	(%Cu)	(Insitu Cu kt)	(%)	(Rec. Cu kt)	($B)	Plant Feed	Rec. Cu	Value
Cu Price $2.50/lb	4,984.8	3,202.8	1.8	1,782.0	0.45	8,036.8	88.0%	7,069.4	**16.29**	-12.7%	-2.8%	-45.2%
Cu Recovery -5%	4,846.6	2,844.7	1.4	2,001.8	0.41	8,227.5	83.6%	6,875.2	**27.15**	-1.9%	-5.5%	-8.6%
Process costs +5%	4,856.1	2,849.1	1.4	2,007.0	0.41	8,248.8	87.8%	7,244.5	**29.12**	-1.6%	-0.4%	-2.0%
Metal Costs +5%	4,860.5	2,827.5	1.4	2,033.0	0.41	8,274.3	87.8%	7,266.2	**29.21**	-0.4%	-0.1%	-1.7%
Mining Costs +5%	4,856.1	2,818.5	1.4	2,037.7	0.41	8,272.9	87.9%	7,268.4	**29.22**	-0.1%	-0.1%	-1.6%
Base Case 60 Mtpa; Cu Price $3.50/lb	**4,864.8**	**2,824.5**	**1.4**	**2,040.4**	**0.41**	**8,283.9**	**87.8%**	**7,273.4**	**29.71**			
Mining Cost -5%	4,871.6	2,829.6	1.4	2,042.0	0.41	8,290.5	87.8%	7,276.7	**30.20**	0.1%	0.0%	1.6%
Metal Costs -5%	4,866.1	2,819.4	1.4	2,046.8	0.41	8,289.4	87.8%	7,278.8	**30.21**	0.3%	0.1%	1.7%
Process costs -5%	4,871.4	2,796.8	1.3	2,074.6	0.40	8,319.2	87.8%	7,300.8	**30.31**	1.7%	0.4%	2.0%
Cu Recovery +5%	4,891.0	2,811.3	1.4	2,079.7	0.40	8,318.6	92.3%	7,676.5	**32.28**	1.9%	5.5%	8.7%
Cu Price $4.00/lb	5,342.4	3,144.1	1.4	2,198.3	0.39	8,639.2	87.8%	7,581.6	**39.69**	7.7%	4.2%	33.6%
Cu Price $4.50/lb	5,390.7	3,115.8	1.4	2,274.9	0.38	8,712.8	87.7%	7,644.2	**47.72**	11.5%	5.1%	60.6%

Table 15-11 Input parameters for retrospective optimisation sensitivity analyses

Sensitivity Scenario	Metal prices			Mining Waste	Mining Ore	Processing	Other	G&A	Total Procost	Metal Costs		
	Cu ($/lb)	Mo ($/lb)	Au ($/oz)	$/t rock	$/t rock	$/t ore	$/t ore	$/t ore	$/t ore	$/lb Cu	$/lb Mo	$/oz Au
Base case optimisation: 60 Mtpa	**$3.50**	**$12.00**	**$1,800**	**$2.13**	**$1.89**	**$4.69**	**$0.08**	**$1.05**	**$5.82**	**$0.45**	**$0.21**	**$4.59**
1. Optimisation: 40 Mtpa												
increased prices only	**$4.50**	**$18.00**	**$3,000**	$2.13	$1.89	$4.69	$0.08	$1.05	$5.82	$0.45	$0.21	$4.59
and adjusted op. costs	$4.50	$18.00	$3,000	**$2.00**	**$2.00**	**$7.25**	**$0.08**	**$1.52**	**$8.85**	**$0.48**	**$0.22**	**$4.59**
2. Optimisation: 60 Mtpa												
increased prices only	**$4.50**	**$18.00**	**$3,000**	$2.13	$1.89	**$4.69**	$0.08	$1.05	$5.82	$0.45	$0.21	$4.59
and adjusted op. costs	$4.50	$18.00	$3,000	**$2.00**	**$2.00**	**$6.76**	**$0.08**	**$1.40**	**$8.24**	**$0.48**	**$0.22**	**$4.59**

Table 15-12 Summary of retrospective optimisation sensitivity results

Sensitivity Scenario	Pit Size	Waste	Strip	Plant Feed			Recovery	Rec'd Metal	Undisc. Value	% changes relative to Base Case					MCOG
	(Mt)	(Mt)	Ratio	(Mt)	(%Cu)	(Insitu Cu kt)	(%)	(Rec. Cu kt)	($B)	Pit size	Waste	Plant Feed	Rec. Cu	Value	Cu_{eq}(%)
Base case optimisation: 60 Mtpa	**4,864.8**	**2,824.5**	**1.4**	**2,040.4**	**0.41**	**8,283.9**	**87.8%**	**7,273.4**	**29.71**						**0.11**
1. Optimisation: 40 Mtpa															
increased prices only	4,997.4	2,735.7	1.2	2,261.7	0.38	8,481.5	87.8%	7,449.0	48.99	2.7%	-3.1%	10.8%	2.4%	64.9%	0.07
and adjusted op. costs	4,922.2	2,930.4	1.5	1,991.8	0.41	8,246.1	87.9%	7,246.6	41.94	1.2%	3.8%	-2.4%	-0.4%	41.2%	0.11
Delta	**57.4**	**105.9**		**-48.5**	**0.08**	**-37.7**		**-26.7**	**12.2**						
2. Optimisation: 60 Mtpa															
increased prices only	4,997.4	2,735.7	1.2	2,261.7	0.38	8,481.5	87.8%	7,449.0	48.99	2.7%	-3.1%	10.8%	2.4%	64.9%	0.07
and adjusted op. costs	4,940.0	2,896.2	1.4	2,043.8	0.41	8,297.7	87.9%	7,293.0	43.17	1.5%	2.5%	0.2%	0.3%	45.3%	0.11
Delta	**75.2**	**71.7**		**3.4**	**0.41**	**13.9**		**19.6**	**13.5**						

15.5 Open pit design

15.5.1 Initial pit design and validation

Following on from the original optimisation, a new ultimate pit design (Figure 15-8) was completed conforming to the updated geotechnical and general mine design parameters. This design is similar to that upon which the geotechnical design domain boundaries are overlain in Figure 15-9.

The minor differences are essentially with respect to the in-pit primary crusher and conveyor alignment, and to the haul ramp layout in geotechnical domains (zones) 3, 4-1 to 4-3, and 5.

Table 15-13 lists a comparison between the various optimisation inventories and the new ultimate pit design. 'Delta 1' refers to differences between the originally selected pit shell and the pit design, whereas 'Delta 2' and 'Delta 3' refer to differences between the two retrospective pit shells and the pit design.

The validation results in Table 15-13 show a reasonable level of correspondence. This is corroborated by the validation plans and cross sections shown in Figure 15-10 to Figure 15-12.

Table 15-13 Validation between pit optimisations and ultimate pit design

	Units	Pit shell (60 Mtpa)	Retro shell 1 (40 Mtpa)	Retro shell 2 (60 Mtpa)	*Design pit*	Delta 1	Delta 2	Delta 2
Pit size	**Mt**	**4,864.8**	**4,922.2**	**4,940.0**	***4,936.4***	**101.5%**	**100.3%**	**99.9%**
Waste	Mt	2,824.5	2,930.4	2,896.2	***2,945.9***	104.3%	100.5%	101.7%
Strip ratio		1.4	1.5	1.4	***1.5***			
Plant feed	**Mt**	**2,040.4**	**1,991.8**	**2,043.8**	***1,990.5***	**97.6%**	**99.9%**	**97.4%**
Feed grade	%Cu	0.41	0.41	0.41	***0.43***			
Insitu Cu	kt	8,283.9	8,246.1	8,297.7	***8,548.6***	103.2%	103.7%	103.0%
Recovery	%	87.8%	87.9%	87.9%	***87.8%***			
Rec'd Cu	**kt**	**7,273.4**	**7,246.6**	**7,293.0**	***7,505.8***	**103.2%**	**103.6%**	**102.9%**

Figure 15-8 Ultimate pit design

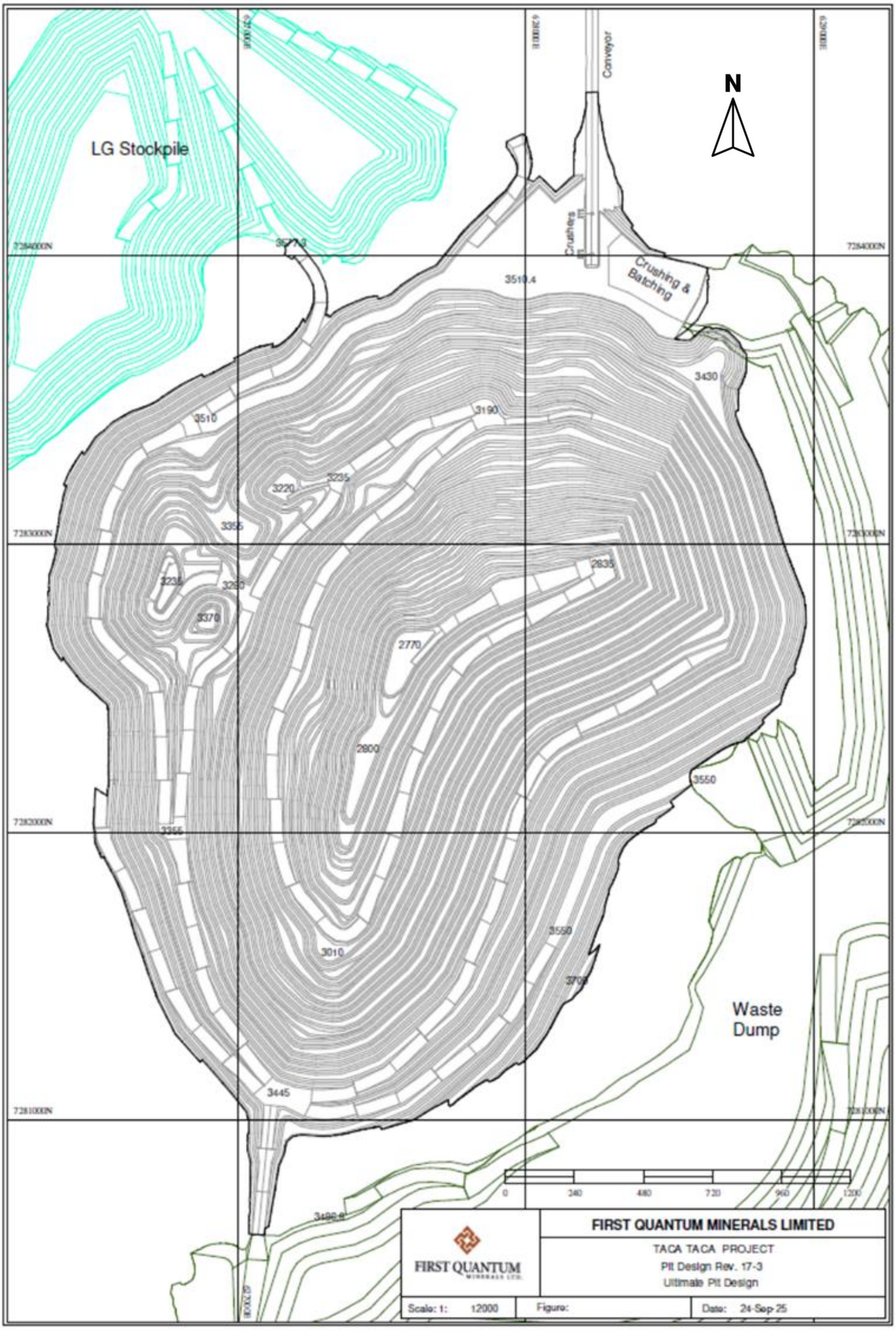


Figure 15-9 Revised pit slope design parameters

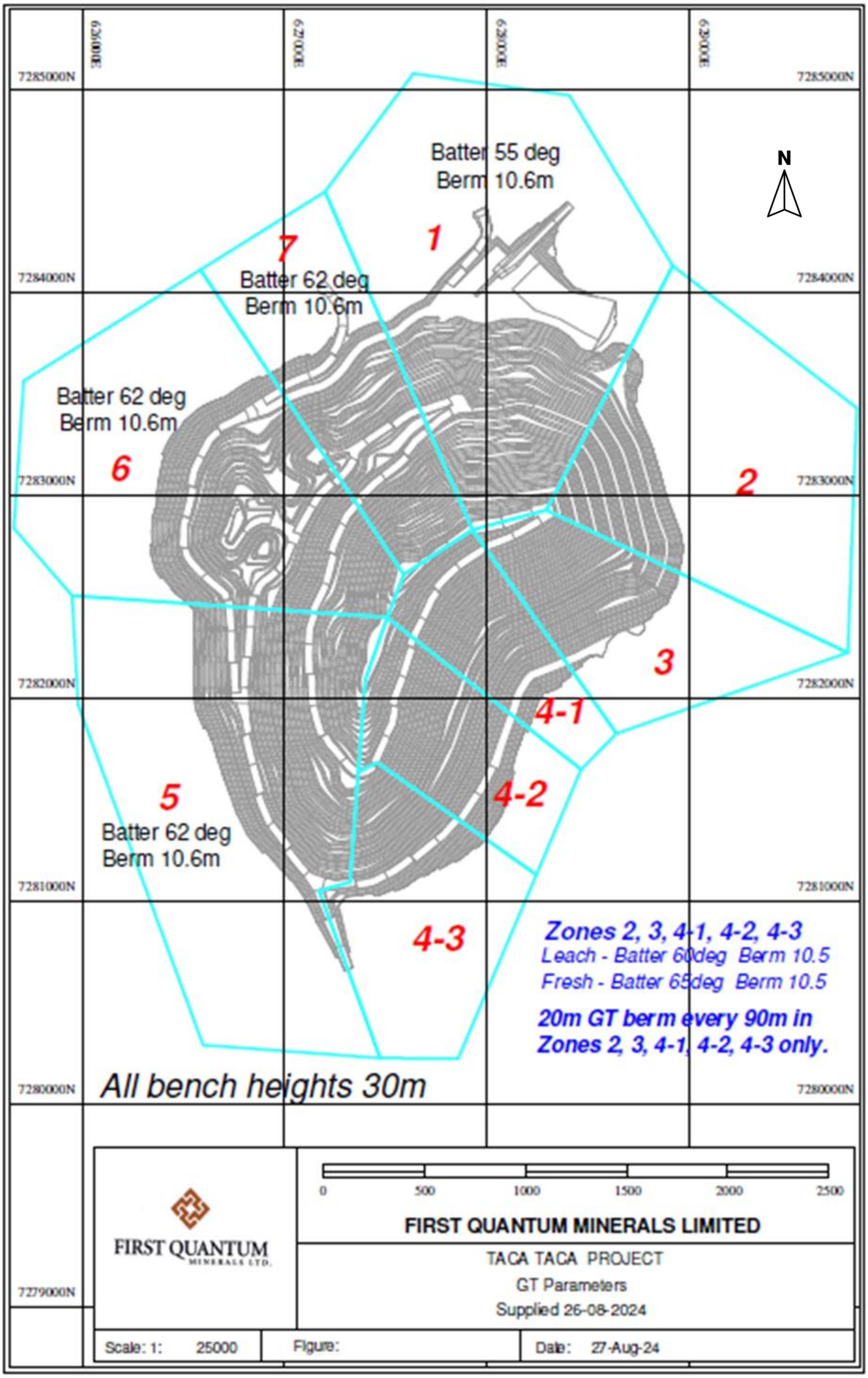


Figure 15-10 Cross section plan; optimal pit shell and ultimate design

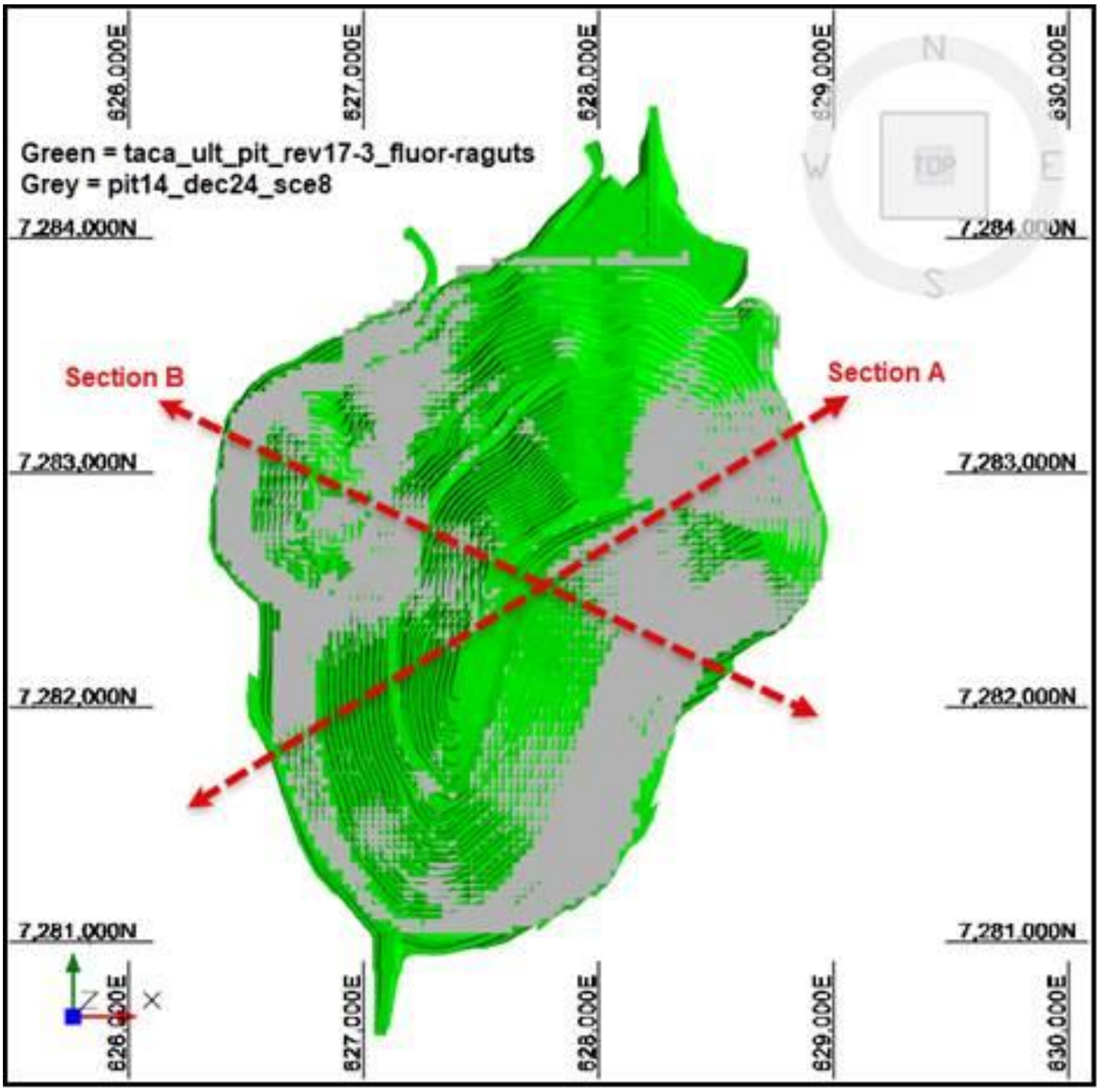


Figure 15-11 Cross section along AA; optimal pit shell and ultimate design

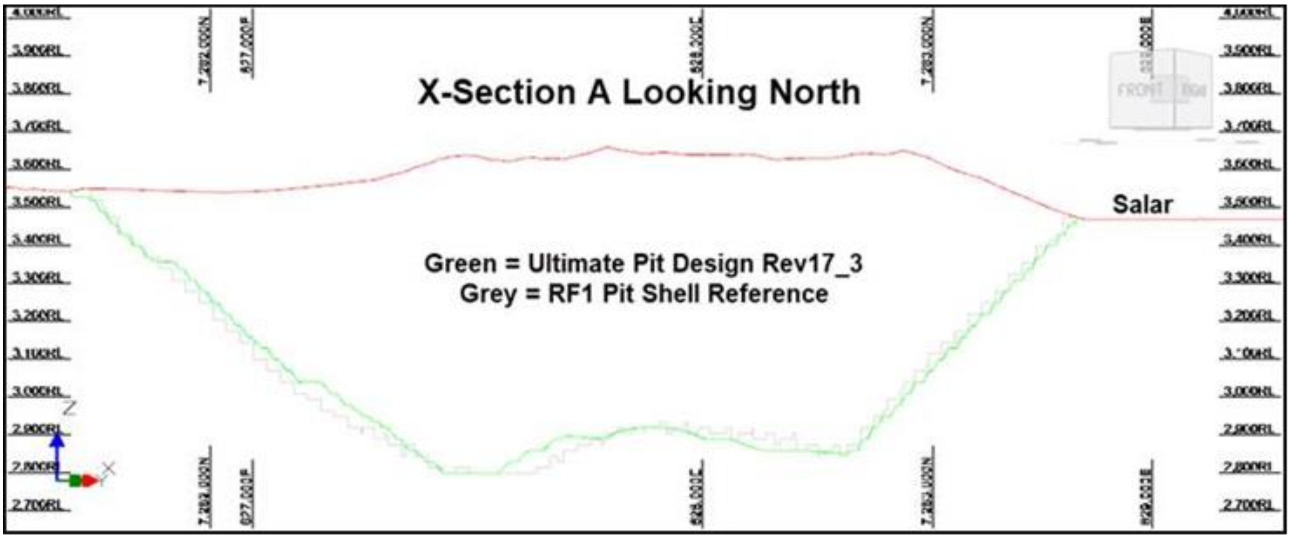
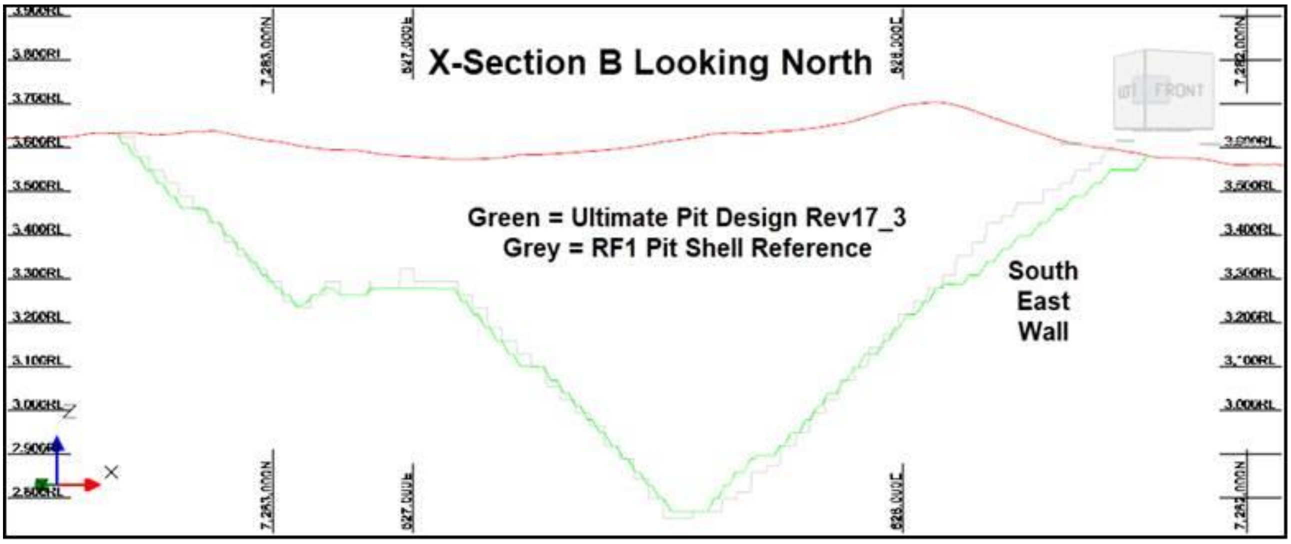


Figure 15-12 Cross section along BB; optimal pit shell and ultimate design



15.5.2 Staged and revised ultimate pit designs

Upon completion of the site layout designs, a series of updated stage pit designs and a revised ultimate pit design were produced as shown in Figure 15-13 to Figure 15-16. Figure 15-17 shows the four pit stages overlain on one another.

By comparison with the set of designs in the 2021 Technical Report, which featured six discrete stages:

- The emphasis of the update was to establish only four stages of mining, each with a broader footprint for the deployment of a combined equipment fleet (i.e., transitioning from the waste pre-strip) and for improved operability.
- In conjunction with and during the production scheduling process, the integration of wide terraces in stripping zones enabled the maintenance of reasonable stockpile levels and the feeding of the minimum quantity of low grade ore during the initial years of mining.
- During the first three stages of mining, a wider ramp than usual has been provided on the east wall, immediately adjacent to the Salar de Arizaro. This is a geotechnical contingency allowance until operational scrutiny and stability monitoring enables the ultimate design to be revised if and as required.

Stage 1 pit design (Figure 15-13)

- Stage 1 is no longer subdivided into 1a and 1b.
- Stage 1 is now deeper and wider at the base, at 3220 mRL, as opposed to the 2020 version at 3400 mRL / 3250 mRL.
- Two access ramps for waste haulage are designed, down to the 3370 mRL bench.
- The design now includes a platform that enhances the interoperability and connection between stages at the 3460 mRL bench level.
- The same platform can be used for the purposes of:
  - parking equipment during shift changes, during blasting delays, etc; thereby reducing fuel consumption.
  - performing equipment maintenance in-pit
- An additional ramp exit has been established from Stage 1 to the waste dump in the south-east section, which will facilitate the reduction of cycle time and ramp congestion.

- Stage 1 includes a wider ramp on the eastern wall adjacent to the salar, designed as a safety and operational risk mitigation measure in the event of any geotechnical issues.

Stage 2 pit design (Figure 15-14)

Relative to the 2021 designs, this stage is now significantly wider, and deeper, better enabling the deployment of combinations of electric shovels and auxiliary equipment whilst also providing improved ore exposure and reducing an operational risk that would otherwise be imposed by a confined working area.

By comparison with the 2021 Stage 2 design:

- The bottom bench level is now at 3115 mRL, as opposed to 3225 mRL in the 2020 version.
- The completed stage dimensions are now 2.1 km long and 1.2 km wide, as opposed to 1.4 km long and 1.3 km wide in the 2020 version.
- Two access ramps for waste haulage are designed, down to the 3325 mRL bench.
- The platform described for Stage 1 remains in place, enhancing the interoperability and connection between stages.
- The number of waste haulage exits has been raised from two to three, while the connection (bridge) between the pit and the waste dump located south-east of the pit remains in place.
- The wide ramp on the east wall remains.

Stage 3 pit design (Figure 15-15)

The improved operational considerations and approach are similar to those as described for Stages 1 and 2. Specifically:

- The bottom bench level is now at 2995 mRL, as opposed to 3100 mRL in the 2020 version.
- The completed stage dimensions have been increased to 2.7 km by 1.5 km, from 2.1 km by 1.3 km.
- Two access ramps for waste haulage are designed, down to the 3070 mRL bench.

Ultimate Pit design (Figure 15-16)

By comparison to the 2021 design, the revised design of the ultimate pit has undergone only minor alterations. These adjustments have been influenced to a lesser extent by changes in metal prices and pit optimisation inputs, and to greater extent by the addition of geotechnical berms at every 90 m (vertical) as previously outlined, in addition to the modifications to the haul ramp layout to allow greater flexibility for the hauling of waste rock.

Figure 15-13 Taca Taca Stage 1 design pit

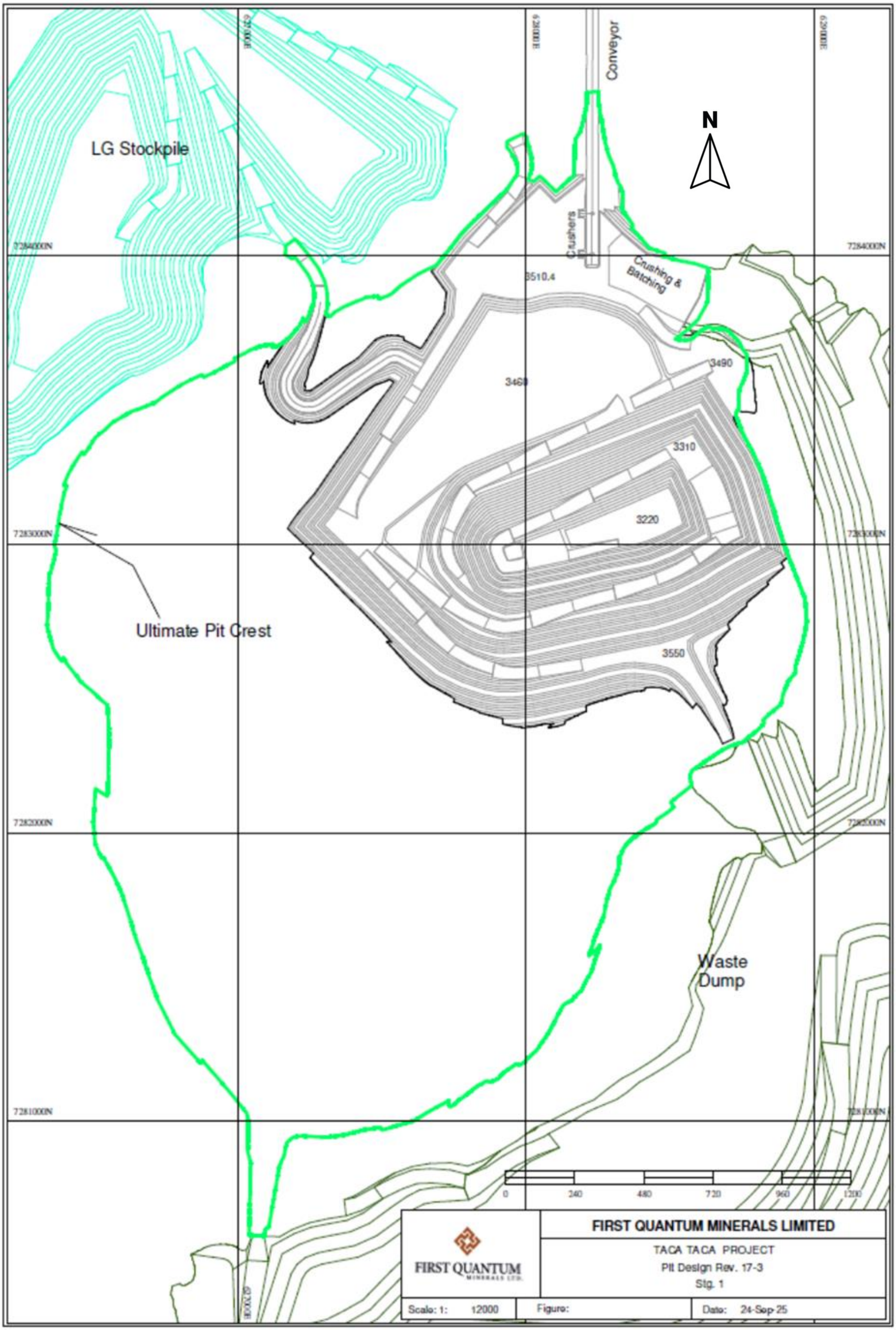


Figure 15-14 Taca Taca Stage 2 design pit

Figure 15-15 Taca Taca Stage 3 design pit

Figure 15-16 Taca Taca Stage 4 (ultimate) design pit

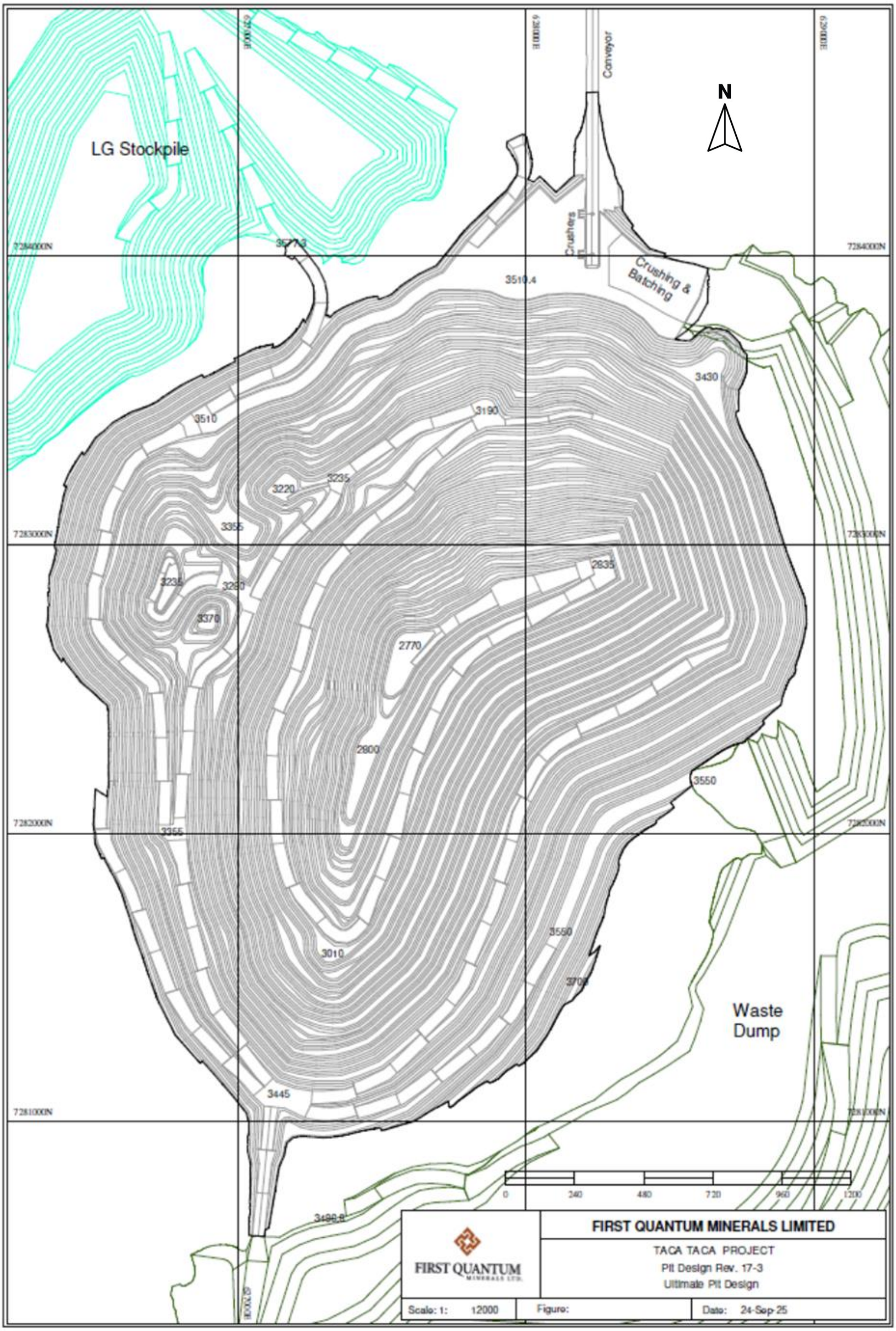
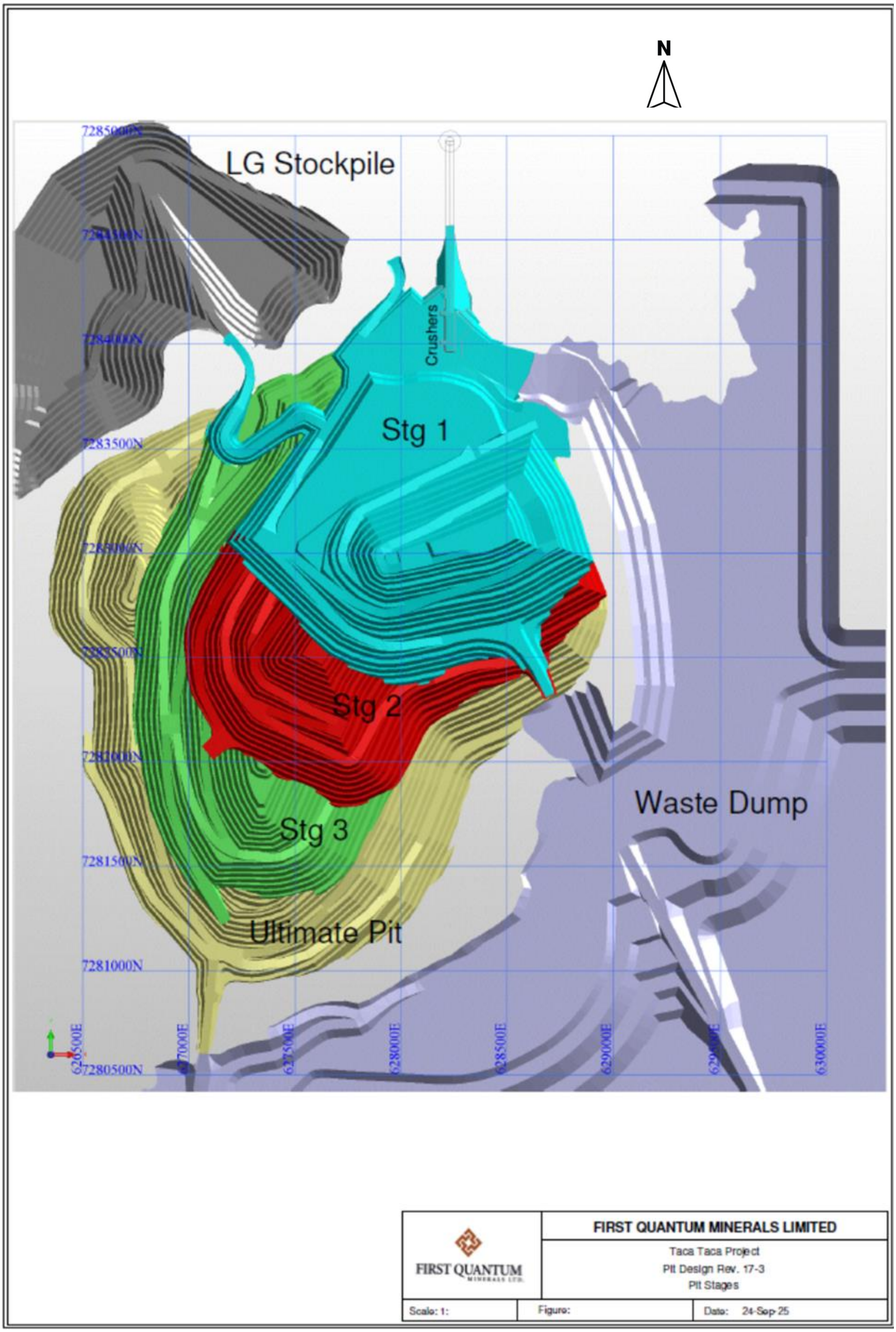


Figure 15-17 Taca Taca overlain stage and ultimate pit designs



15.6 Waste dump design

15.6.1 Location

During the selection of the preferred location for the waste dump, the following aspects were taken into account:

- the dump is located on the Salar de Arizaro, to the east of the pit, and along some of its length there will be a rock ridge between the dump and the pit
- the ultimate overall slope angle of the dump is <25° to 30°; very flat and not atypical for a waste dump design
- the slopes are unlikely to have any pore water pressure in such an arid environment
- the bearing capacity of the Salar surface indicates that subsidence could be expected at dump heights approaching 20 m to 50 m, depending on the conditions below the Salar crust (GP, September 2019, refer Item 16.1.6):
  - the ultimate dump height is in the order of 250 m to 300 m
  - if the underlying sediments were to compact resulting in slope toe movement, then at the dump height and design overall slope angle involved, any slumping would likely be contained within the slope geometry
  - compaction and subsidence would likely be gradual, and should there be any impact to the dump slope surface, it is unlikely to be sudden or especially hazardous
  - if the eastern, northern and southern walls of the dump were to slump, then apart from a visual impact, there would be no hazard to infrastructure
  - if the western wall of the dump was to slump, it is sufficiently distant from the pit to be not a hazard

Figure 15-18 shows the location of the waste dump, covering an area of approximately 1,393.6 ha. Of this area, approximately 1,078.4 ha would cover the Salar de Arizaro.

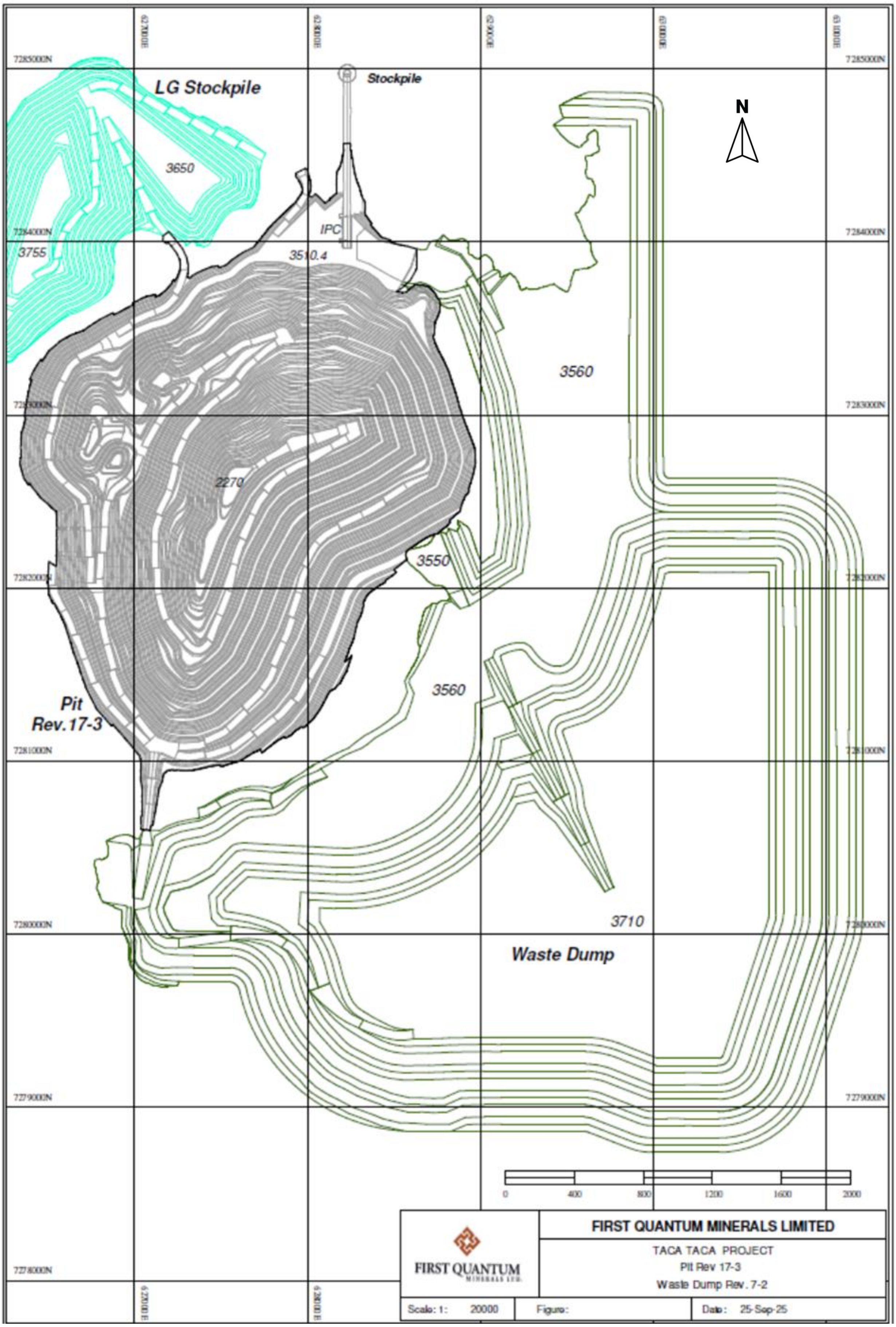
15.6.2 Capacity

Waste rock generated from mining the pits would be stored within a single large dump located to the east of the pit area. Excluding potential auriferous material mined but including mineralised waste and (currently) Inferred Resource, the total mined material quantity expected to be dumped is approximately 2,903.4 Mt and will require a waste landform that can receive about 1,523 Mm^3 (assuming an insitu to placed bulking factor of 1.35).

The dump will be sufficient for storage of 497 Mlcm of mined non-acid forming (NAF) waste and 1,026 Mlcm of potentially acid forming (PAF) waste. The actual designed capacity of the dump is 2,069 Mm^3, which can be used to accommodate mineralised waste and auriferous rock stockpiles.

Although the waste dump design can reach up to 250 m and 300 m in other scenarios evaluated, the haulage simulation reaches a height of ~210 m.

Figure 15-18 Waste dump location and ultimate extents



15.7 Ore stockpile design

A surface stockpile provided for active ore stockpiling/reclaim is shown in Figure 15-19. The maximum size of this ore stockpile is up to 204 Mt (in Year 38) with a capacity of 103 Mlcm. A ROM pad “surge” stockpile allows for short-term rehandling with the intention of providing flexibility at the crusher tipping points and avoiding prolonged truck queuing times.

There is another stockpile located within the waste dump, which for the most part, will be allocated low-grade mineralised waste material and reaching a maximum size of 362 Mt in Year 41.

15.8 Project site layout

Figure 15-20 shows the layout of the mine site, and the waste dump and stockpile site, relative to the proposed location of the processing plant and other facilities.

Figure 15-19 Ore stockpile layout

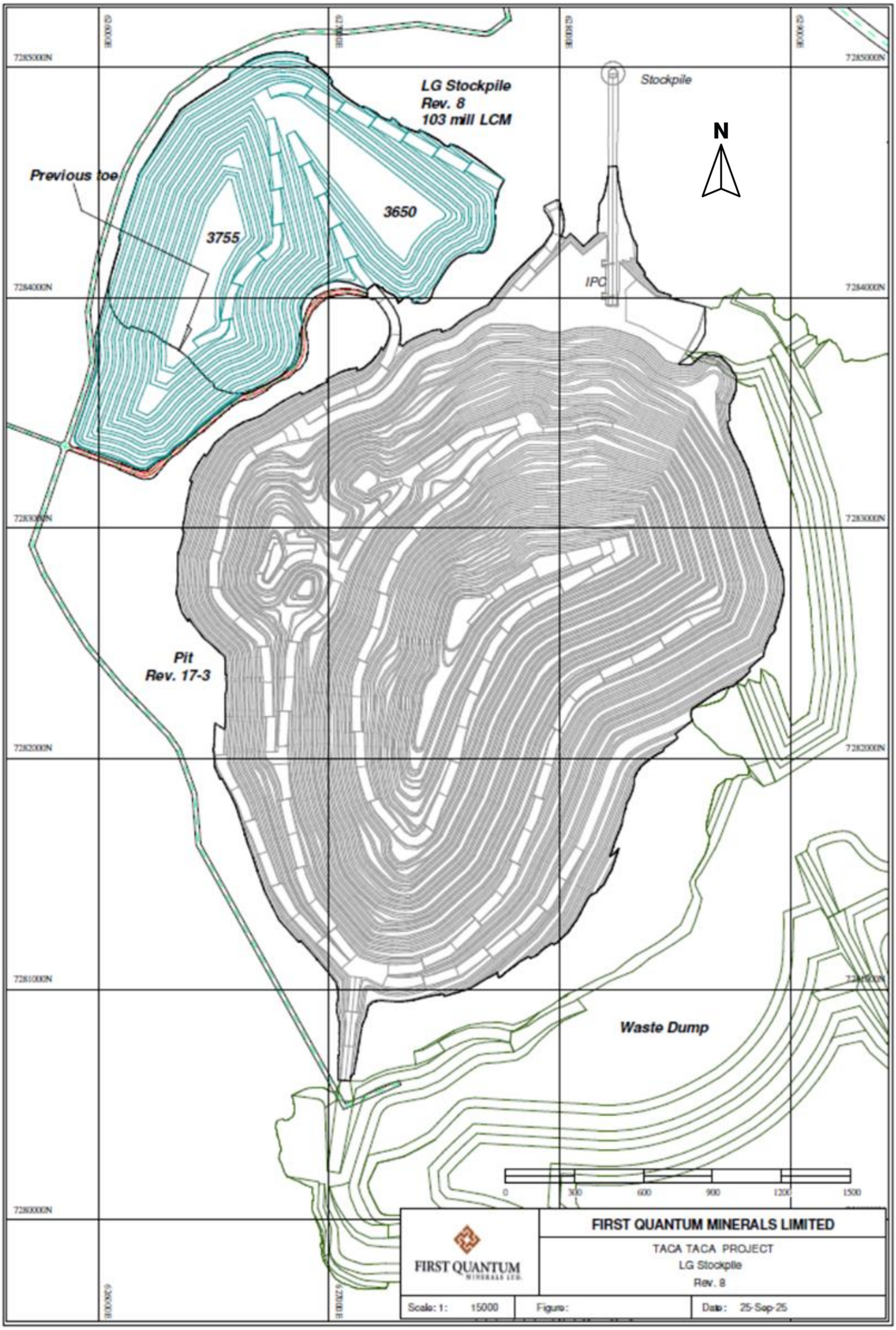


Figure 15-20 Site layout plan

15.9 Mineral Reserve statement

The Mineral Reserve statement provided in Table 15-14 is derived from a conventional pit optimisation and detailed pit design approach, supported by a production schedule for the ore and waste mining inventory within that pit design. The pit optimisation process adopted only the Measured and Indicated Mineral Resource, with Inferred Mineral Resource allocated to waste. Mining assumes conventional open pit operations using truck-and-shovel technology. The estimate in Table 15-14 accounts for mining dilution and mining recovery losses.

The actual marginal cut-off grade for the Mineral Reserve varies according to the copper recovery assigned to the various mineralogical groupings. The overall average marginal copper cut-off grade reflects long term metal price projections of $3.50/lb Cu, $12.00/lb Mo and $1,800/oz Au.

Table 15-14 Taca Taca Mineral Reserve estimate, at December 2025

Classification	Tonnes (Mt)	Cu (%)	Mo (%)	Au (g/t)	Cu metal (kt)	Mo metal (kt)	Au metal (koz)
Proven	432.1	0.58	0.015	0.13	2,509.5	66.8	1,835.6
Probable	1,558.0	0.38	0.011	0.07	5,919.0	177.6	3,696.6
Prov. + Prob.	**1,990.1**	**0.42**	**0.012**	**0.09**	**8,428.5**	**244.4**	**5,532.2**

As part of the ultimate pit design for the Project, there is a small area in the north-west that crosses over into the *Mina Francisco* joint venture concession. This encroachment, included in the Table 1-3 inventory, amounts to approximately 3.1 Mt of ore at an average grade of 0.41% Cu.

Relative to the Mineral Reserve statement reported in 2021 (FQM, 2021), the above statement indicates approximately:

- a 6% increase in Proven Reserve tonnes
- a 15% increase in Probable Reserve tonnes
- a 13% increase in combined Proven + Probable Reserve (232 Mt at an average grade of 0.30%)
- a 9% increase in insitu copper metal
- a 15% increase in insitu molybdenum metal
- a 9% increase in insitu gold metal

The waterfall charts produced from the pit optimisation results provide a good indication of the reasons behind these increases, most notably from the adoption of increased metal prices, improved process recoveries (i.e. owing to the change to milling and flotation in fresh water), and revisions to the pit slope design.

15.10 Potential impacts on Mineral Reserve estimates

The updated Mineral Reserve yields an additional 230 Mt of plant feed relative to the Reserve reported in the 2021 Technical Report. The Stage 1 40 Mtpa LOM production schedule (Item 16) shows a “tail” of long term stockpile reclaim into the processing plant at the conclusion of mining. About 30% of this feed is from remaining high/low grade stocks at an average grade of 0.21%Cu, whilst 70% is from marginal grade (mineralised waste) at an average grade of 0.13%Cu[12]. Some of this 230 Mt of additional inventory is therefore marginal grade ore which would be fed into the plant in the final decades of the operation.

[12] Information on ore grade ranges is provided in Item 16.2.

Pit optimisation sensitivity analyses indicate that a long term copper price of $4.50/lb could sustain the inclusion of this marginal grade plant feed in the Mineral Reserve inventory. This is borne out by the economic analysis in Item 22.

The updated pit slope design specifications used for pit optimisation and design continue to be based on limited geotechnical drilling and mechanical testing data. A programme of further geotechnical drilling, testing and associated hydrogeological data collection has been recommended (FQM, November 2024). This is to be followed by comprehensive geotechnical slope stability analyses and design updates, as necessary.

ITEM 16 MINING METHODS

16.1 Mining method and description

The ore mineralisation grades, geometry and depth of the Taca Taca deposit make it suitable for conventional bulk open pit mining. This is an intuitive view rather than a conclusion drawn from an analysis of other methods such as underground mining or strip mining. A solely underground mining approach is expected to be highly unlikely to deliver the required annual ore processing tonnages with minimal risk, when compared with a conventional open pit mining approach. A strip-mining approach is unsuitable for this orebody due to the small lateral extent of the pit when compared with the final vertical depth. This is coupled with an inability to backfill previously mined voids with waste material, without compromising future ore supply.

There is a possibility that supplementary underground mining could be adopted as the open pit approaches its ultimate depth. This possibility may be assessed during the continuing Project engineering phase.

16.1.1 Open pit mining equipment

The Taca Taca pit would be mined using conventional open pit methods involving blasthole drills, diesel loaders and hydraulic excavators, electric shovels and off-highway haul trucks.

16.1.2 Drilling and blasting

Drilling and blasting activities would be carried out by the Company. Near-surface material may be mined essentially as ‘free-dig’ (i.e., not requiring blasting for excavation) material. Once operations commence, there will be better field identification of areas with potential drill and blast savings from free-dig practices. With increasing mining depth, production drilling and blasting would take place in rock conditions requiring a range of drilling/charging patterns and powder factors.

Relative to the current geological definition, indicative geological properties have been used to devise generic production and wall control blasting requirements (and cost estimates) listed as follows:

- rock properties:
  - density = 2.65 g/cm³
  - unconfined compressive strength = 150 MPa
  - Young’s Modulus = 53.8 GPa
  - Poisson’s ratio = 0.27
- a bench height of 15 m
- a production blasthole diameter of predominantly 270 mm in ore and 311 mm in waste, wall control blasthole diameter of 229 mm and presplit hole diameter of 165 mm
- a bulk explosive product with an average in-hole density of 1.20 g/cm³ and relative weight strength of 115%[13]
- a packaged explosive with a diameter of 30 mm and a density of 1.10 g/cm³
- a waste fragmentation target with an 80% passing size of 321 mm, which should be suitable for efficient excavation using large shovels and excavators with a bucket size greater than 30 m³

[13]. Relative weight strength compared to an equal mass of Ammonium Nitrate/Fuel Oil (ANFO) at 0.80 g/cm³ and an absolute energy of 2.30 MJ/kg.

- an ore fragmentation target with a 99.50% passing size of 442 mm, which should be suitable for both the primary crushers and for efficient excavation using large shovels and excavators with a bucket size greater than 30 m³

Table 16-1 provides an overview of the above concepts.

Table 16-1 Drilling and blasting parameters

Rock Type	Hard		
Scenario	**Trim**	**Waste**	**Ore**
Rock Mass Properties			
Rock density (g/cm³)	2.65	2.65	2.65
Compressive strength (MPa)	150.0	150.0	150.0
Young's Modulus (GPa)	53.8	53.8	53.8
Rock Mass Factor	5.78	5.78	5.78
Pattern Parameters			
Hole Diameter (mm)	229	311	270
Bench Height (m)	15	15	15
Burden (m)	6.7	8.5	6
Spacing (m)	7.7	9.8	6.9
Spacing factor	1.15	1.15	1.15
Pattern Type	Staggered	Staggered	Staggered
Subgrade (m)	0	3	3
Stemming (m)	4.7	6.3	5.5
Explosive Product Type	HANFO	HANFO	HANFO
Explosive Density (g/cm³)	1.1	1.2	1.2
Relative Weight Strength (%)	115	115	115
Relative Bulk Strength (%)	158.1	172.5	172.5
Powder Factor (kg/m³)	0.60	0.85	1.38
Energy Factor (MJ/m³)	0.60	0.85	1.38
Drill Productivity (tonne/m)	136.7	184.0	91.4
Fragmentation Parameters			
P_{20} (mm)	83	82	59
P_{50} (mm)	197	179	121
P_{80} (mm)	376	321	207
$P_{99.5}$ (mm)	933	733	442
% Passing 25.4 mm	5	4	6

It is anticipated that in the future, the geotechnical model will furnish further information to facilitate the evaluation of the potential to increase the face angles to greater than 65°, as envisaged in the current design. The implementation of a double bench will enable the presplit to be executed in double passes >30 m; this controlled blasting technique would be required initially along and in front of all batter and overall slope faces. Pre-splitting is a form of controlled blasting, and in this instance, would involve the drilling of a single row of blastholes in front of the wall profile. These holes would be lightly charged and fired in the same adjacent trim or production blast, but milliseconds before that blast.

The pre-split blastholes would be smaller diameter (165 mm) and would be charged with packaged explosive, allowing for the application of a decoupled charge, distributed along the length of the blasthole. The presplit row would be fired well in advance of the adjacent blasts to ensure that the rock at the pit wall is adequately protected.

16.1.3 Loading and hauling

A "pioneering" fleet is considered to better suit the initial pre-strip and the potentially confined operating space for effective mining during this period. Initially, the Project considers the acquisition of a 2,000 t/hour class hydraulic excavator and front-end loader, with a capacity to match 92 t to 120 t trucks. However, further refinements will be evaluated in the context of a contractor option, or the size of the owned fleet for pioneering will be reviewed again according to topography and the further use of the same equipment, such as its use in operations for support, rehandling, scaling and battering practices.

A larger "operational" fleet is considered, comprising rope shovels of 5,000 t/hour capacity matched to 290 t capacity trucks. This fleet would be assisted with 700 t class face shovels and front-end loaders.

The 2021 Technical Report had opted for 360 t capacity trucks from the outset, consistent with the ultra-class fleet scale in operation at other Company sites. In this latest update, the focus has been on a type of truck with a lower capacity, and one which is utilised extensively by companies operating in the proximity of the Project. It is anticipated that this will enhance synergy and ensure more efficient response times when it comes to the acquisition and distribution of spare parts.

16.1.4 Trolley-assisted haulage

Trolley assisted haul lanes have been designed for all mid and long-term haulage routes, and the road widths have been optimised to meet the minimum requirements for the selected truck class, always ensuring the availability of double lanes.

Regarding the waste dump, it is also anticipated that trolley routes will be available at each of the permanent access ramps situated in the south, the middle, and in the north of the dump.

As part of a dynamic event simulation by Theia Consulting (March 2025), the adoption of trolley-assisted haulage showed a 10% decrease in truck travel time relative to unassisted travel time for a loaded haul truck coming out of the pit.

In FQM practice, trolley ramps require triple lane width (up, down and drop-off lanes) for efficient operation, and to suit the haul trucks applicable to this Project, the design pit ramps are mostly not less than 42 m wide and not steeper than 1:10 gradient.

In detailed operational designs it may be possible to incorporate trolley routes in some instances within the smaller stage pit designs, even with curved haulage segments. This aspect is to be further evaluated as the Project engineering phase proceeds.

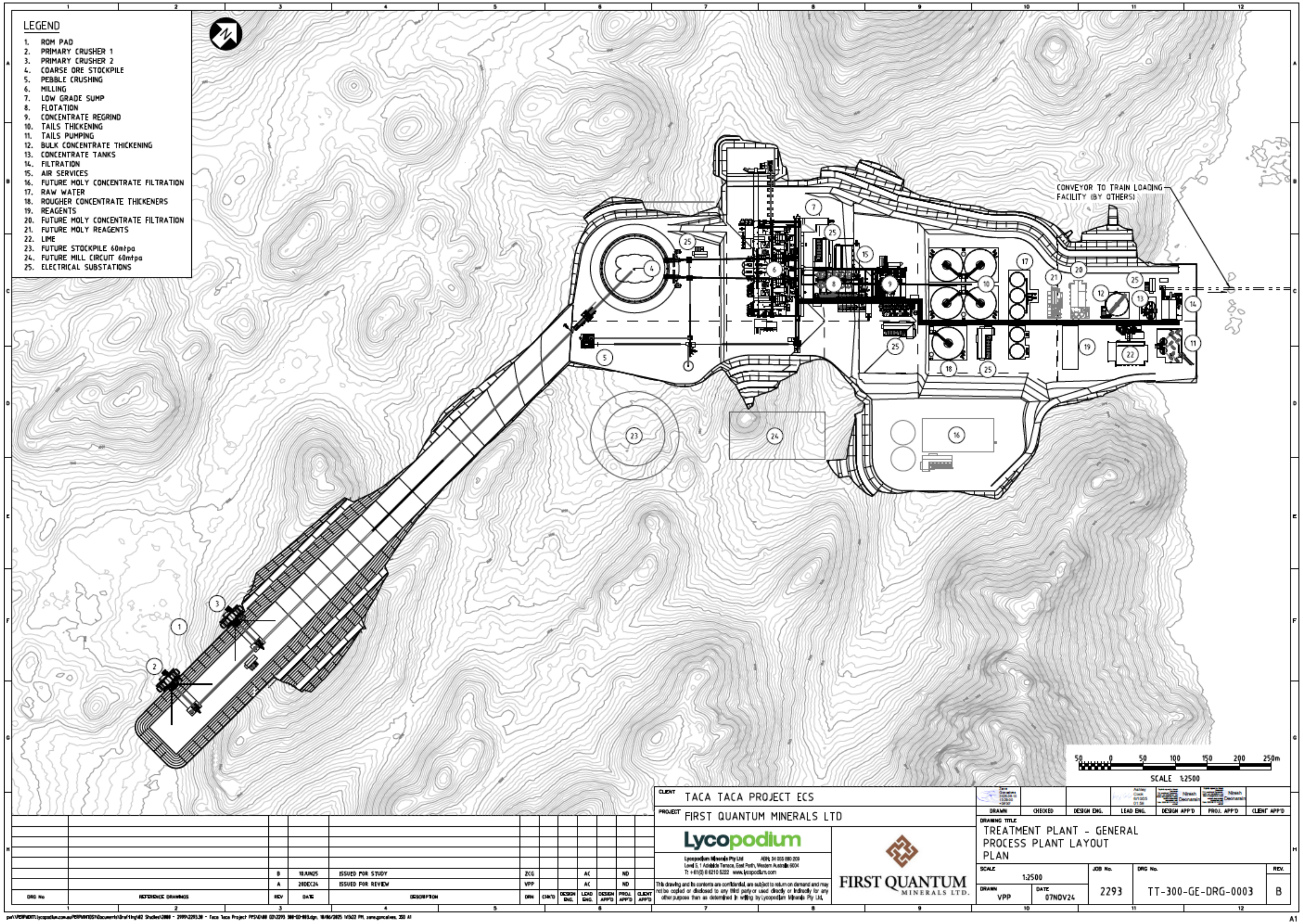
16.1.5 Ore crushing and conveying

Figure 16-1 shows the Stage 1 40 Mtpa plant layout including the ore crushing and conveying arrangement, the features of which are:

- primary crushers located on the north west side of an excavated slot, with a tipping bench situated in the pit at 3,511.5 mRL
- a discharge conveyor located within the slot extending up to the crushed ore stockpile, the location of which has changed to suit the revised process plant position
- an 8.4 ha area has been set aside adjacent to the crushers for surge stockpiling

A layout has not been produced for the Stage 2 60 Mtpa layout; Figure 16-1 shows where the expansion components could be in the future. Under these circumstances, the alignment of the crushed ore conveyor would need to be modified, as would the excavated slot.

Figure 16-1 Stage 1 40 Mtpa Process plant and primary crushing layout (source: Lycopodium, June 2025)



16.1.6 Waste dumping

Given the topography and general geography of the Project, as well as the distribution of waste and ore materials in the deposit, it is considered that the southeast location of the waste dump provides the optimal conditions for the execution of the Project.

As part of the enhancements to the Project and considering the constraints imposed by concession boundaries, a staged dump design and sequencing approach has been developed. This will allow more materials to be accommodated without compromising or threatening future resource development. Furthermore, in comparison with previous versions, the new design has the potential to offer distinct advantages including a reduction in the footprint on the Salar de Arizaro.

The mineralised waste material, which has a relatively low grade but is above the cutoff grade calculated in the economical pit design process (explained in Item 15.3), will also be placed in this waste dump. This material will initially be dumped separately, ready for rehandling towards the end of the mine's operational life.

Gold mineralised rock from the pit pre-strip horizons would also be stockpiled on the waste dump, available for future processing consideration.

Dump slope stability

In the selection of the preferred site for the waste dump, the following was taken into account:

- the ultimate overall slope angle of the dump is <30°, very flat and not atypical for a waste dump design
- the slopes are unlikely to have any pore water pressure in such an arid environment
- the bearing capacity of the salar surface indicates that subsidence could be expected at dump heights approaching 20 m to 50 m, depending on the conditions below the salar crust (GP, September 2019):
    - the ultimate dump height is in the order of 300 m
    - if the underlying sediments were to compact resulting in slope toe movement, then at the dump height and design overall slope angle involved, any slumping would likely be contained within the slope geometry
    - compaction and subsidence would likely be gradual and should there be any impact to the dump slope surface, it is unlikely to be sudden or especially hazardous
    - if the eastern, northern and southern walls of the dump were to slump, then apart from a visual impact, there would be no hazard to infrastructure
    - if the western wall of the dump was to slump, it is sufficiently distant from the pit to be not a hazard

The waste dump would ultimately be operated in accordance with industry best practice and, where possible, would be progressively battered to a long term stable angle as mining activities proceed.

Dozers would be used to ensure appropriate tip-head management and safe dumping practices. Access to the tip heads would be achieved using ramp systems constructed through the waste rock landform and providing access to multiple tip heads. This is to maintain tip heads at a maximum safe working height of 20 m and minimise horizontal haul distances across the dump. Additional safety controls if deemed necessary in design and operation on the dump, could include dumping short and dozing out to the edge.

During the engineering phase, the overall landform and placement of the PAF and NAF volumes within the waste dumps and TSF embankment would be refined using dump optimising software. The aim would be to minimise horizontal and vertical haul costs within pre-defined dump constraints such as stand-off distance from the open pit crest and maximum height.

Acid rock drainage (ARD) management

In the prevailing arid environment, the waste dump slopes are unlikely to be impacted by pore water pressure. Similarly, these conditions and the absence of waste dump runoff may inhibit the possibility of ARD conditions developing within the waste dump slopes.

ARD conditions are more likely to be generated at the base of the waste dump, where non-acid forming waste (NAF) waste lies in contact with brine in the underlying Salar. Recent work has been done to quantify the relative volumes of NAF and potentially acid-forming (PAF) waste, in the context of available volumes of NAF rock that could be placed as a waste dump base layer.

Further commentary on the revised NAF and PAF rock differentiation is provided in Item 16.3.1. The production scheduling of NAF and PAF waste volumes (implying the placement of NAF as a waste dump base layer) is listed in Table 16-8.

Mine water management

The major components for mine site water management, including management of waste dump run-off, will include diversion channels, collection and settlement ponds.

16.1.7 Ore stockpiling

As part of the simulations carried out for the Project, several decisions were made regarding the long-term ore stockpile:

- the mine-plan seeks to feed the best grades as direct-tip which reduces the amount of high-grade ore that is stored in the stockpile.
- this makes most of the stockpiled material low grade and reduces the dilution of grades in the stockpile.

The stockpile, located where shown in Figure 16-2, is intended for the long-term stockpiling of high- and low-grade ore, reaching 150 Mt in Year 17 and a maximum of 200 Mt in Year 35. The mineralised waste is intended to be stored within the footprint of the waste dump, in such a way that it can be reclaimed in the final nine years of processing. The maximum that the mineralised waste dump reaches before reclamation is 373.3 Mt.

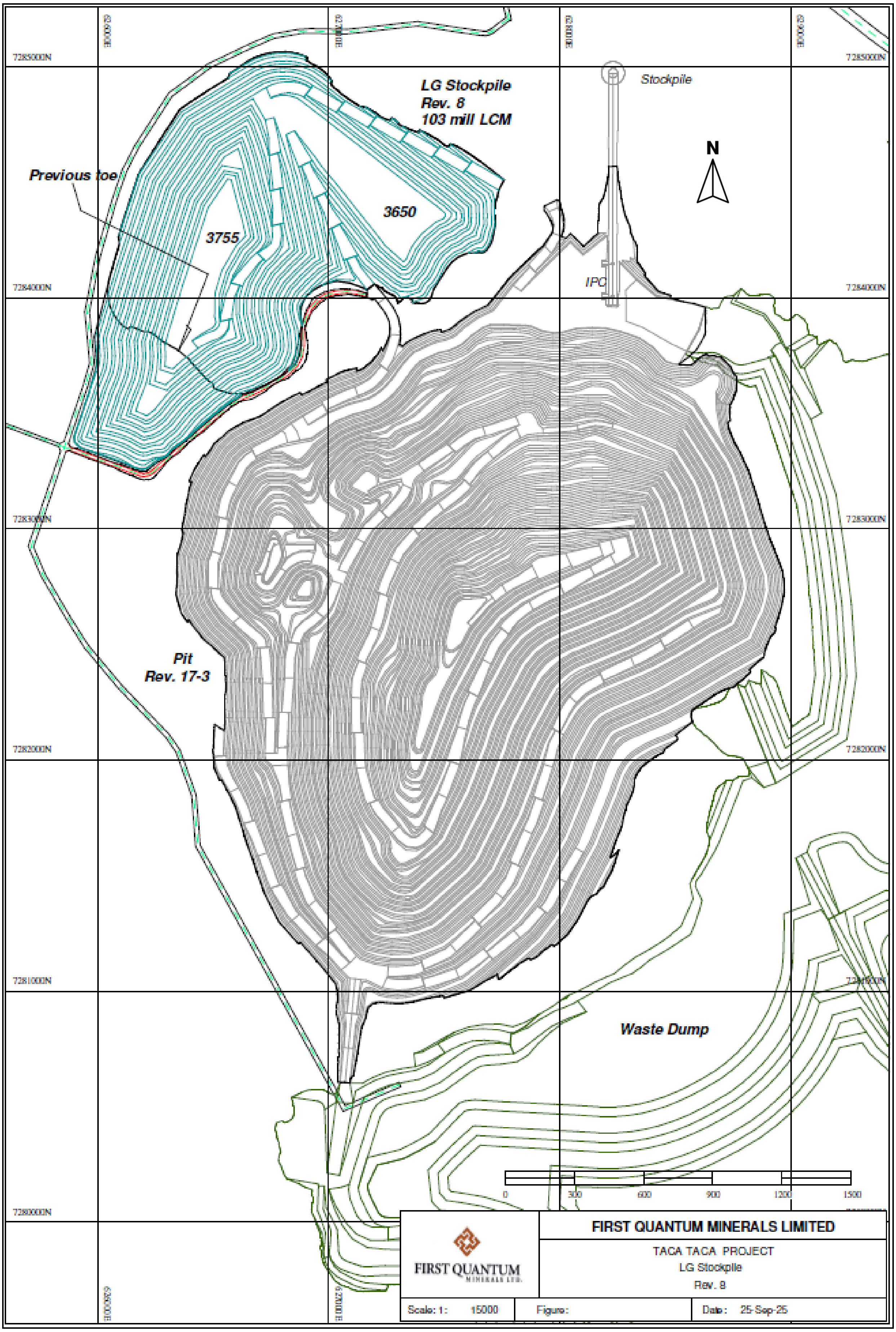
16.1.8 Pre-mining activities

As part of the construction phase, initial mining areas will have been stripped of thin topsoil, if any, and then excavated for waste construction materials. Preliminary mining of plant feed will have been undertaken to provide plant commissioning stockpiles.

Following the construction phase, new mining areas would routinely be opened up into wide benches, enabling typical pre-mining activities to commence. These activities, throughout the entire operations phase, will include close-spaced grade control drilling and sampling (for improved mining Resources and Reserves definition), geological mapping, construction of temporary access ramps, extension of services to drills and shovels, and digging of drainage ditches and water sumps.

The deposit has been found to contain a gold resource associated with the leach cap zone, which is likely to be encountered during the initial mine development works and subsequently during pre-strip mining. The perched gold mineralisation is to be stockpiled pending assessment of the viability for further processing. The additional storage capacity required is 56 Mt, to be located within the designated waste dump footprint.

Figure 16-2 Ore stockpile layout



16.1.9 Grade control

Conventional open pit grade control practices are envisaged, incorporating RC drilling and sampling on a suitably designed drilling pattern and over multiple bench horizons. Multi element sample assaying will be carried out on site. A grade control modelling process will be implemented as the basis for designing dig blocks. Supplemental grade control may be carried out using blasthole sampling, as required.

16.2 Mine planning parameters

16.2.1 General design parameters

The design of the open pit slopes has followed the geotechnical specification of the inter ramp slope parameters listed in Table 16-2, with general design parameters typically as follows:

- mining bench height of 15 m
- batter heights = 30 m
- haul ramp width of 42 m
- haul ramp gradient of 10%

Pit design sectors with a 15 m bench height in the leach zone will be assessed in the future stages of mine development; this will not change the overall angle but will provide more bench-to-bench control in localised areas exhibiting lower rock quality.

Table 16-2 Pit slope batter/berm design criteria

Slope Sector	Horizon	1	2	3	4			5	6	7
					4.1	4.2	4.3			
Bench face angle (degrees)	Leach	55	60	60	60	60	60	62	62	62
	Fresh	55	65	65	65	65	65	62	62	62
Catch bench width (m)	All	10.6	10.5	10.5	10.5	10.5	10.5	10.6	10.6	10.6
Catch bench vertical Interval (m)	All	30	30	30	30	30	30	30	30	30

For operational reasons, a decision was made to increase the number of haul ramps in sectors with a high stripping ratio for Stages 1 to 4, thereby ensuring at least two independent accesses to the pit and two independent exits to the waste dump.

Geotechnical berms or step-outs of 20 m width, at every 90 m in height, have been incorporated into the pit design to cater for zones of inferior rock quality.

16.2.2 Geotechnical parameters

Mine geotechnical engineering

An updated geotechnical study was completed during 2024 (FQM, November 2024), specifically to review the design parameters for the ultimate eastern pit wall adjacent to the Salar de Arizaro. The design domains around the entire pit perimeter were revised and updated design parameters specified for all domains. The new parameters are shown in Figure 16-3 and listed in Table 16-2 and Table 16-3.

The updated pit slope design specifications continue to be based on limited geotechnical drilling and mechanical testing data. A programme of further geotechnical drilling, testing and associated hydrogeological data collection has been recommended (FQM, November 2024). This is to be followed by comprehensive geotechnical slope stability analyses and design updates, as necessary.

Figure 16-3 Revised pit slope design parameters

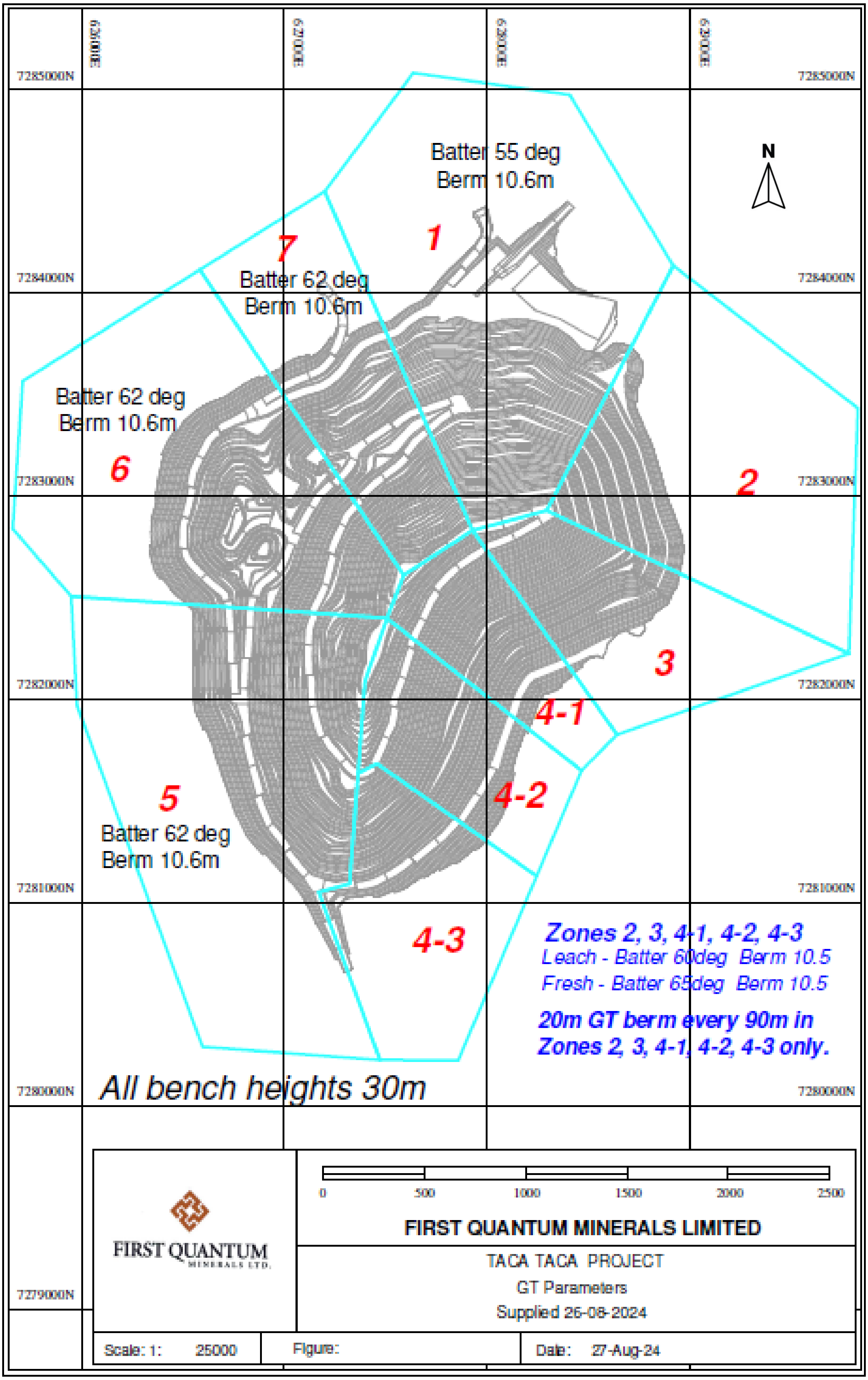


Table 16-3 Pit slope design criteria

Domain	RL range (m) (m)	O'all slope angle (deg)
1	< 3220	36.0
	3220 - 3340	39.0
	> 3340	38.0
2		47.5
3	< 3190	47.0
	3190 - 3370	48.0
	> 3370	45.5
4-1	< 2950	43.0
	2950 - 3250	48.0
	3250 - 3430	47.0
	> 3430	38.0
4-2	< 2860	57.0
	2860 - 3040	44.0
	3040 - 3280	49.0
	3280 - 3460	47.0
	> 3460	39.0
4-3	< 3400	47.5
	> 3400	39.0
5	< 2920	46.0
	2920 - 3070	40.0
	3070 - 3310	50.0
	3310 - 3430	41.0
	> 3430	50.0
6	< 2950	50.0
	2950 - 3250	45.0
	> 3250	45.0
7	< 2875	45.0
	2875 - 2980	37.0
	2980 - 3130	49.0
	3130 - 3250	25.5
	3250 - 3340	49.0
	3340 - 3490	40.5
	> 3490	28.0

Waste dump geotechnical engineering

Geotec Perforaciones (GP) carried out a geotechnical investigation of the salar surface that would underlie the proposed waste dump (GP, September 2019). This investigation involved the drilling of six vertical holes to a depth of 10 m over the footprint area of the dump, with Standard Penetration Testing (SPT) at each metre of drilling, and then followed by laboratory testing on samples taken from every hole, i.e.:

- natural moisture content
- plasticity index
- particle size analysis
- soil classification by the Unified System
- Proctor tests and Support Value determination
- chemical analysis

Geological logging of the drill hole cuttings showed a salar surface profile composed of a salt crust to 2.5 m depth, overlying horizons of sand of varying compactness, from loose to dense depending on the degree of crystallisation of the halite salts. The sands contained minimal fine material, and were typically of low plasticity, with a high natural moisture content. Table 16-4 shows that the bearing capacity was determined as a range from 3.9 kg/cm^2 (382 kPa) to 10.9 kg/cm^2 (1,068 kPa). These values indicate that there could be localised areas of compaction or heave of the Salar surface, potentially leading to slumping of the outer slope surface at the dump perimeter (GP, September 2019).

At the lower level of bearing capacity, compaction of the Salar surface could be expected at dump heights approaching 20 m. GP advised that such compaction would likely be gradual and should there be any impact to the dump slope surface, it is unlikely to be sudden or especially hazardous.

Table 16-4 Geotechnical analysis for the waste dump

	units	loose sands	compacted sands
bearing capacity	kg/cm^2	3.9	10.9
	kg/m^2	39,000	10,900
bearing pressure	kN/m^2 (kPa)	382.2	1068.2
dump. comp. density	kg/m^3	2,060	2,060
limiting dump height	m	18.9	52.9

16.2.3 Mining dilution and recovery

For pit optimisation and mine planning purposes, geological losses were built into a regularised mine planning model to account for the low level of ore continuity on the edge of the main ore zone. These losses could be considered as “planned mining dilution”. The re-blocking impact of this regularisation is indicated in Table 16-5, where the inventory comparison is reported when the two models are constrained by the same indicative ultimate pit design. The difference in contained metal is less than 1%.

In the Whittle optimisation inputs, “unplanned dilution” of 5% and a mining recovery factor of 95% were included to emulate practical mining losses. In the absence of operational reconciliation information, these selected factors are reasonable for the bulk mining of large orebodies.

Table 16-5 Impact of model re-blocking

Parameter	Units	Original Resource Model	Reblocked Resource Model	Planned dilution	
				dilution	metal loss
Block dimensions	m x m x m	7.5 x 7.5 x 7.5	7.5 x 7.5 x 15.0		
Tonnes	Mt	1,604.3	1,697.5	5.8%	
Grade	% Cu	0.51	0.48		
Contained metal	kt Cu	8,134.0	8,080.0		0.7%

16.2.4 Ore grade ranges

The ore grade ranges adopted for mine planning and production scheduling are as follows:

- high grade ore is greater than 0.30% Cu_{eq}
- low grade ore is between 0.22% Cu_{eq} and 0.30% Cu_{eq}
- mineralised waste (MW or marginal grade ore) is between marginal cut-off and 0.22% Cu_{eq}

16.2.5 Open pit inflow and pit slope depressurisation

In terms of water inflow to mine workings, localised packer testing at the Project site indicates that permeability across fractures often results in higher hydraulic conductivity values, whilst the recorded higher hydraulic heads suggest that groundwater movement within the mining environment will likely be dominated by discrete fracture zones rather than through the intact rock matrix (Ausenco, 2011). An evaluation of the effects of faulting and rock mass alteration on rock mass properties has not been completed.

Ausenco (2011) produced a hydrogeological model to assess the groundwater response to open pit mining drawdown. This modelling indicated the potential for the development of excess pore water pressure within the pit walls. Analysis by W&N (2106) indicated that pore water pressure impacts could be decreased, with the potential for steeper overall slope angles, by means of allowing natural drainage in the leach cap horizons but with an active depressurisation system in the underlying sulphide deposit.

The Ausenco analyses showed:

- natural inflow arising from excavation of the pit would range from 16 L/s to 33 L/s
- active dewatering (pit slope depressurisation) from pit bores and horizontal drains could yield 28 L/s to 54 L/s

16.3 Stage 1 40 Mtpa production schedule

With the completion of the ultimate and staged pit designs, a Stage 1 40 Mtpa life of mine (LOM) production schedule was completed using Mine Planner software. Scheduling assumptions included:

- minimum mining block size for stage 1,2 = 200 m x 200 m x 15 m (X, Y, Z); for the stage 3 and 4 = 400 m x 400 m x 15 m (X, Y, Z)
- mining bench height = 15 m
- in the initial periods, mining is to be carried out using terrace systems
- the average sinking rate is:
  - stage 1: average three benches per year, with a peak of five benches in Year 1 (owing to terraced and small benches)
  - stage 2: average of three benches per year, with peaks of five benches in Year 8 and six benches in Year 9
  - stage 3: average of two benches per year, with a peak of four benches in Year 2
  - Stage 4: average of two benches per year, with a peak of six at the end of the mine life (owing to small areas on remaining benches and seeking to avoid fixed costs for an additional year of mining).

In the production scheduling process, a modified approach was taken when compared with that adopted for pit optimisation purposes (Item 15.3.2). A mining dilution factor was applied relative to the depth of the model blocks in the planning model. In this instance, 5% dilution (at zero grade) was applied from the surface to the 3,475 mRL level; 2.5% dilution (at zero grade) between the 3,475 and 3,440 mRL levels; and 1.0% dilution (at zero grade) below 3,440 mRL. A mining recovery factor of 95% was also applied in this process.

Other scheduling constraints and strategies were as follows:

- the ore processed in Year 5 will be 24.9 Mt, ramping to 40 Mtpa from Year 6 onwards
- provide a relatively consistent Cu production level, whilst maximising the annual profile over the first eight years of processing so as to compensate for the high strip ratio initial mining
- in principle, the intent is to feed high grade Cu first (Cu_{eq}>=0.30%):
  - this helps to generate a better production profile but also requires a larger low-grade stockpile.
- mineralised waste (MW or marginal ore) is to be stockpiled on the waste dump:
  - where possible, this material will not be direct fed or actively reclaimed to the plant until after open pit mining has been depleted completely
  - as part of the mine sequence and as described above, the MW will be directed onto an area of the waste dump which does not permanently sterilise future Mineral Resources.

The schedule level of detail is annually for all periods. Additionally, as a measure to mitigate any operational/sequencing risk in the early years, it was decided to develop a detailed monthly plan covering the first five years. Features of the LOM production schedule as listed in Table 16-6 are as follows:

- Mining (i.e., starting with the pre-strip period) commences in mid-Year 1 whilst processing commences in Year 5. The Project life at 40 Mtpa (processing years) is 50 years.
- 422.1 Mt of waste is mined in the pre-strip period, during which time 10.5 Mt of ore is mined onto a stockpile for subsequent active and longer-term reclaim.
- The total material mined over the life of operations amounts to 5,615.5 Mt (1,737.0 Mbcm) of which:
- 1,990.1 Mt is ore with average grades of 0.42% Cu, 0.012% Mo and 0.09 g/t Au, and 2,903.4 Mt is waste
- The overall life of mine strip ratio (waste tonnes: ore tonnes) is 1.46 : 1.
- The direct feed ore to the plant is 1,268.1 Mt at an average grade of 0.56% Cu, whilst 348.6 Mt at an average grade of 0.26% Cu is ore reclaimed from active stockpiles, and 373.3 Mt at an average grade of 0.13% Cu is ore (marginal ore) reclaimed from longer term stockpiles (mostly after the mine depletion).
- The Inferred Mineral Resource that is mined as waste amounts to 124.5 Mt (i.e., about 4.3% of the total waste mined). This material is encountered in the mining schedule after Year 9 and following completion of mining the stage 1 pit.
- The crusher feed ramps up from Year 5 at 24.9 Mt, and thereafter to 40 Mtpa until Year 50.
- In terms of total plant feed (after mining dilution/recovery):
  - the weighted average copper grade is 0.73% Cu for the first eight years,
  - then 0.53% Cu to Year 17,
  - then 0.44% Cu to Year 37,
  - then 0.21% Cu to Year 41,
  - and finally 0.13% Cu for the remaining nine years of Project life when reclaiming from longer term stockpiles
- Before the final thirteen years of long term stockpile reclaim, the total plant feed is 1,465.8 Mt at an average grade of 0.52% Cu.
- The annual average copper metal production to Year 12 (after eight years of processing) is 239.5 kt, and ranging between 112.4 kt and 283.1 kt. Thereafter, the annual average is 162.1 kt, ranging between 121.20 kt and 207.4 kt (ignoring the final years of stockpile reclaim). In terms of life of Project totals:
  - 1,915.9 kt of copper is recovered in the first eight years,
  - then 4,712.2 kt of copper to Year 41,
  - and finally 683.6 kt of copper for the remaining thirteen years of Project life when reclaiming from longer term stockpiles
- In terms of total metal produced during the Project life, the production sequence generates 7,312 kt of copper, 108 kt of molybdenum, and 3,367 k(t)oz of gold.

16.3.1 Scheduled ore and waste distribution

The Stage 1 life of mine ore and waste mining schedule is listed in Table 16-7. The total mined waste includes 124.5 Mt of Inferred Resource and 55.9 Mt of auriferous mineralisation (Domain 102). The total mined ore includes secondary mineralisation (18% of the total), supergene mineralisation (60%) and primary mineralisation (22%). Except for years 12 to 14, the largest proportion of ore mined is supergene; year 12 corresponds to the year in which stage 1 mining is completed.

Table 16-6 Stage 1 40 Mtpa life of mine production schedule

Year	Stage	Mining						Processing				Metal Insitu			Metal Recovered			Average Recovery		
		Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)	Waste (Mt)	Total Mined (Mt)	Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)	Cu (kt)	Mo (kt)	Au (k(t)oz)	Cu (kt)	Mo (kt)	Au (k(t)oz)	Cu (%)	Mo (%)	Au (%)
1	Pre-strip					19.3	19.3													
2						94.3	94.3													
3		0.5	0.18	20.34	0.06	146.7	147.3													
4		9.9	0.28	34.41	0.06	161.7	171.6													
5	Production	52.3	0.37	52.72	0.07	143.0	195.2	24.9	0.61	71.60	0.09	151.7	1.8	75.2	132.3	0.7	45.3	87.2%	40.0%	60.3%
6		51.5	0.59	80.74	0.10	143.8	195.2	40.0	0.73	92.31	0.12	292.0	3.7	147.9	252.8	1.5	88.5	86.6%	40.0%	59.9%
7		45.5	0.77	110.81	0.14	150.3	195.8	40.1	0.78	110.56	0.14	312.9	4.4	181.8	271.5	1.8	109.6	86.8%	40.3%	60.3%
8		42.4	0.69	136.20	0.12	152.8	195.2	40.0	0.78	119.50	0.13	312.0	4.8	163.3	271.6	2.0	97.9	87.1%	40.8%	60.0%
9		52.7	0.67	182.71	0.14	142.5	195.2	40.0	0.80	178.74	0.15	320.0	7.1	187.8	283.1	3.0	113.8	88.5%	42.6%	60.6%
10		46.7	0.67	196.76	0.17	148.5	195.2	40.0	0.80	206.76	0.18	320.0	8.3	226.3	284.0	3.7	137.9	88.8%	44.2%	60.9%
11		56.2	0.56	118.68	0.11	139.5	195.8	40.1	0.71	147.23	0.13	284.8	5.9	166.4	248.3	2.7	102.1	87.2%	45.1%	61.4%
12		69.9	0.42	87.35	0.09	125.4	195.2	40.0	0.60	127.53	0.12	240.0	5.1	158.2	208.0	2.3	96.8	86.7%	45.8%	61.2%
13		57.8	0.40	79.91	0.08	137.4	195.2	40.0	0.50	102.33	0.10	200.0	4.1	128.6	171.6	2.0	79.3	85.8%	47.8%	61.6%
14		55.0	0.37	84.69	0.08	140.2	195.2	40.0	0.46	103.02	0.10	184.0	4.1	131.2	158.3	2.0	81.4	86.0%	49.2%	62.1%
15		57.8	0.39	114.85	0.10	137.9	195.8	40.1	0.49	132.91	0.11	196.5	5.3	144.4	171.0	2.3	88.0	87.0%	43.9%	60.9%
16		63.4	0.40	108.27	0.09	116.8	180.2	40.0	0.52	128.16	0.12	208.0	5.1	150.5	181.7	2.4	92.2	87.3%	46.1%	61.3%
17		74.0	0.43	118.59	0.11	82.0	156.0	40.0	0.59	136.44	0.13	236.0	5.5	168.5	207.1	2.4	103.2	87.8%	44.4%	61.3%
18		73.8	0.44	111.62	0.10	58.0	131.8	40.0	0.59	129.65	0.12	236.0	5.2	158.2	207.4	2.4	97.7	87.9%	47.1%	61.8%
19		57.3	0.42	110.48	0.11	59.8	117.1	40.1	0.59	132.26	0.14	236.6	5.3	183.1	207.3	2.5	112.4	87.6%	46.2%	61.4%
20		51.9	0.42	96.65	0.09	64.9	116.8	40.0	0.50	110.42	0.11	200.0	4.4	141.5	173.6	2.1	87.3	86.8%	48.1%	61.7%
21		48.5	0.42	98.30	0.09	68.3	116.8	40.0	0.50	109.82	0.10	200.0	4.4	128.6	173.1	2.1	79.9	86.5%	48.7%	62.1%
22		43.6	0.37	149.35	0.10	73.0	116.6	40.0	0.41	132.48	0.10	164.0	5.3	127.3	142.2	2.5	78.8	86.7%	47.7%	61.9%
23		55.5	0.40	130.07	0.09	43.3	98.8	40.1	0.47	141.05	0.11	188.5	5.7	138.0	163.4	2.6	84.7	86.7%	45.4%	61.4%
24		56.6	0.35	139.17	0.08	41.9	98.5	40.0	0.46	133.11	0.09	184.0	5.3	110.6	158.1	2.3	67.2	85.9%	43.5%	60.7%
25		60.9	0.34	106.43	0.07	37.6	98.5	40.0	0.45	110.43	0.08	180.0	4.4	102.9	153.9	1.8	62.2	85.5%	41.8%	60.5%
26		60.3	0.38	112.94	0.07	38.2	98.5	40.0	0.47	126.75	0.09	188.0	5.1	111.9	162.2	2.2	67.9	86.3%	42.8%	60.7%
27		62.6	0.31	135.46	0.07	36.2	98.8	40.1	0.42	148.16	0.08	168.5	5.9	108.3	146.6	2.6	66.2	87.0%	44.0%	61.1%
28		59.2	0.27	107.75	0.06	39.3	98.5	40.0	0.35	118.42	0.08	140.0	4.7	99.0	121.2	2.1	60.9	86.6%	45.3%	61.5%
29		68.4	0.31	110.86	0.06	30.1	98.5	40.0	0.43	106.09	0.07	172.0	4.2	88.7	150.6	2.1	55.8	87.6%	49.0%	62.9%
30		67.0	0.33	115.16	0.05	31.5	98.4	40.0	0.45	108.85	0.06	180.0	4.4	78.4	158.6	2.2	49.6	88.1%	50.7%	63.2%
31		56.0	0.32	129.17	0.07	19.4	75.4	40.1	0.39	134.14	0.08	156.4	5.4	104.5	137.7	2.6	65.3	88.1%	48.4%	62.5%
32		49.5	0.33	145.94	0.07	10.6	60.2	40.0	0.38	146.26	0.08	152.0	5.9	102.9	134.0	2.8	64.6	88.2%	47.5%	62.8%
33		47.5	0.36	132.79	0.07	12.7	60.2	40.0	0.40	128.82	0.08	160.0	5.2	101.6	140.8	2.4	63.3	88.0%	47.3%	62.3%
34		45.8	0.37	134.65	0.06	14.4	60.2	40.0	0.41	134.36	0.07	164.0	5.4	95.2	144.5	2.4	59.0	88.1%	45.6%	62.0%
35		47.0	0.37	146.59	0.07	13.3	60.3	40.1	0.42	146.18	0.08	168.5	5.9	99.3	148.7	2.7	61.2	88.3%	45.5%	61.7%
36		50.9	0.43	166.00	0.07	9.2	60.2	40.0	0.51	172.74	0.08	204.0	6.9	100.3	181.2	3.0	61.1	88.8%	43.6%	61.0%
37		53.1	0.42	170.87	0.07	7.1	60.2	40.0	0.50	175.16	0.08	200.0	7.0	108.0	177.1	2.9	65.6	88.6%	42.0%	60.7%
38		52.7	0.41	155.48	0.08	5.5	58.2	40.0	0.47	161.40	0.09	188.0	6.5	113.2	165.4	2.7	68.5	88.0%	42.0%	60.5%
39		27.8	0.44	166.93	0.07	2.8	30.6	40.1	0.40	156.62	0.08	160.4	6.3	96.7	140.6	2.7	59.2	87.7%	43.4%	61.2%
40		27.7	0.54	163.31	0.08	2.5	30.2	40.0	0.44	151.99	0.08	176.0	6.1	102.9	154.3	2.6	62.7	87.7%	43.1%	61.0%
41		30.9	0.56	162.92	0.07	1.5	32.4	40.0	0.48	154.74	0.08	192.0	6.2	96.5	169.2	2.7	59.1	88.1%	43.2%	61.3%
42								40.0	0.21	130.66	0.08	84.0	5.2	102.9	71.8	2.3	62.3	85.5%	43.3%	60.5%
43								40.1	0.21	130.41	0.08	84.2	5.2	100.6	72.2	2.3	61.3	85.7%	44.0%	60.9%
44								40.0	0.21	130.11	0.08	84.0	5.2	97.7	72.1	2.3	59.7	85.8%	44.8%	61.0%
45								40.0	0.21	130.11	0.08	84.0	5.2	97.7	72.1	2.3	59.7	85.8%	44.8%	61.0%
46								40.0	0.13	77.70	0.03	52.0	3.1	41.2	44.1	1.4	28.0	84.9%	43.6%	68.1%
47								40.1	0.13	74.91	0.03	52.1	3.0	38.7	44.2	1.3	26.4	84.8%	43.5%	68.2%
48								40.0	0.13	74.91	0.03	52.0	3.0	38.6	44.1	1.3	26.3	84.8%	43.5%	68.2%
49								40.0	0.13	74.91	0.03	52.0	3.0	38.6	44.1	1.3	26.3	84.8%	43.5%	68.2%
50								40.0	0.13	74.91	0.03	52.0	3.0	38.6	44.1	1.3	26.3	84.8%	43.5%	68.2%
51								40.1	0.13	74.91	0.03	52.1	3.0	38.7	44.2	1.3	26.4	84.8%	43.5%	68.2%
52								40.0	0.13	74.91	0.03	52.0	3.0	38.6	44.1	1.3	26.3	84.8%	43.5%	68.2%
53								40.0	0.13	74.91	0.03	52.0	3.0	38.6	44.1	1.3	26.3	84.8%	43.5%	68.2%
54								43.9	0.13	74.91	0.03	57.1	3.3	42.4	48.4	1.4	28.9	84.8%	43.5%	68.2%
Total		**1,990.1**	**0.42**	**122.8**	**0.09**	**2,903.4**	**4,893.5**	**1,990.1**	**0.42**	**122.8**	**0.09**	**8,426.4**	**244.4**	**5480.2**	**7,342.8**	**109.1**	**3,380.8**	**87.1%**	**44.7%**	**61.7%**

Table 16-7 Stage 1 40 Mtpa life of mine ore and waste mining schedule

Year	Stage	Waste mined (excl. Min Waste)				Secondary	Supergene	Primary	Ore	Total	Strip ratio
		Waste (Mt)	Inferred (Mt)	Domain 102 (Mt)	Subtotal (Mt)	Domain 304,305,310 (Mt)	Domain 307,308,309 (Mt)	Domain 306 (Mt)	Subtotal (Mt)	Mined (Mt)	
1		17.1		2.2	**19.3**					**19.3**	
2	**Pre-strip**	84.7		9.6	**94.3**					**94.3**	
3		138.9	0.1	7.9	**146.8**		0.5		**0.5**	**147.3**	**280.7**
4		152.2	3.8	9.5	**165.5**		9.9		**9.9**	**175.5**	**16.7**
5		137.3	4.6	5.7	**147.6**	2.4	49.9		**52.3**	**199.8**	**2.8**
6		138.1	1.8	5.6	**145.6**	4.5	46.9		**51.4**	**197.0**	**2.8**
7		141.6	0.7	8.6	**151.0**	2.5	42.1	0.8	**45.5**	**196.4**	**3.3**
8		150.2	1.4	2.6	**154.2**	0.6	40.3	1.5	**42.4**	**196.6**	**3.6**
9		140.6	1.1	1.9	**143.6**	0.2	45.7	6.8	**52.7**	**196.3**	**2.7**
10		146.2	3.7	2.2	**152.1**	0.9	33.3	12.5	**46.7**	**198.9**	**3.3**
11		135.6	7.9	0.1	**143.6**	21.3	25.7	9.3	**56.2**	**199.8**	**2.6**
12		120.8	7.6		**128.4**	32.1	28.9	8.8	**69.9**	**198.2**	**1.8**
13		135.6	3.7		**139.3**	28.2	16.6	13.1	**57.8**	**197.1**	**2.4**
14		139.5	6.8		**146.3**	21.5	19.6	13.9	**55.0**	**201.3**	**2.7**
15		136.6	4.7		**141.3**	11.5	36.8	9.6	**57.8**	**199.1**	**2.4**
16		115.8	7.7		**123.5**	15.3	31.6	16.4	**63.4**	**186.8**	**1.9**
17		78.3	6.5		**84.8**	18.6	39.7	15.6	**74.0**	**158.8**	**1.1**
18		50.1	4.7		**54.8**	15.5	43.9	14.4	**73.8**	**128.6**	**0.7**
19		52.2	5.2		**57.4**	17.9	24.3	15.1	**57.3**	**114.7**	**1.0**
20		61.2	5.8		**67.1**	15.7	17.7	18.5	**51.9**	**118.9**	**1.3**
21		61.5	5.5		**67.0**	13.8	17.9	16.7	**48.5**	**115.5**	**1.4**
22		68.2	4.1		**72.3**	9.3	21.3	13.1	**43.6**	**115.9**	**1.7**
23	**Production**	35.6	5.4		**41.0**	16.8	27.0	11.7	**55.5**	**96.5**	**0.7**
24		35.5	3.0		**38.4**	20.5	29.6	6.5	**56.6**	**95.0**	**0.7**
25		32.9	2.0		**34.9**	24.6	32.1	4.2	**60.9**	**95.8**	**0.6**
26		33.0	2.8		**35.8**	25.6	28.3	6.3	**60.3**	**96.1**	**0.6**
27		30.4	4.3		**34.7**	10.6	37.3	14.7	**62.6**	**97.3**	**0.6**
28		33.9	5.1		**39.0**	14.6	27.2	17.4	**59.2**	**98.2**	**0.7**
29		26.1	4.3		**30.4**	7.3	34.4	26.6	**68.4**	**98.8**	**0.4**
30		26.0	3.2		**29.3**	2.0	34.0	30.9	**67.0**	**96.3**	**0.4**
31		16.4	2.8		**19.2**	0.8	28.4	26.8	**56.0**	**75.2**	**0.3**
32		8.6	1.2		**9.9**	0.1	28.0	21.4	**49.5**	**59.4**	**0.2**
33		9.9	2.0		**11.9**	0.0	27.1	20.3	**47.5**	**59.4**	**0.3**
34		10.1	1.1		**11.2**	0.1	31.6	14.1	**45.8**	**57.0**	**0.2**
35		8.2			**8.2**	0.1	32.5	14.4	**47.0**	**55.2**	**0.2**
36		4.9			**4.9**		40.1	10.9	**50.9**	**55.8**	**0.1**
37		3.8			**3.8**		47.3	5.8	**53.1**	**56.9**	**0.1**
38		2.8			**2.8**		45.9	6.8	**52.7**	**55.4**	**0.1**
39		1.5			**1.5**		22.8	5.1	**27.8**	**29.3**	**0.1**
40		0.5			**0.5**		22.6	5.1	**27.7**	**28.3**	**0.02**
41		0.4			**0.4**		24.4	6.5	**30.9**	**31.3**	**0.01**
Total		**2,723.0**	**124.5**	**55.9**	**2,903.4**	**355.0**	**1,193.5**	**441.6**	**1,990.1**	**4,893.5**	**1.5**

Figure 16-4 shows the mined ore types split into the metallurgical domains carried through from the Mineral Resource model. The supergene mixed ore (Domain 307, 308, 309) is clearly shown for the period when direct feeding ore from the near surface of the stage 1 pit. The large proportion of feed from the secondary ore sources (Domains 304, 305 and 310) is also shown. These sources would be processed in all years of the operation, supplemented with primary ore feed (Domain 306).

The mined distribution of NAF and PAF waste is listed in Table 16-8. This updated distribution reflects the changed definition of NAF and PAF criteria as discussed in Section 16.1.6. In the 2021 Technical Report, the %S grade differentiator was low enough to result in a conservative estimate of the NAF volume being around 4% of the total waste rock mined. Under these circumstances, it would have been difficult to provide sufficient NAF as a waste dump under-layer on the surface of the Salar de Arizaro.

The updated criteria now results in around 32% of the mined waste rock being classified as NAF. This additional NAF material is now sufficient to cover 876 ha, which is the area of the revised waste dump design positioned on the salar. The total volume of NAF required to cover this area to a nominal depth of 10 m is 79.3 Mlcm. The placement of this NAF base layer is scheduled progressively over a period of 6.5 years from the outset of pre-strip mining.

The main priority of the mine plan was to ensure the correct placement of the waste dump, creating an average 10 m NAF only base layer. However, it was recognised that NAF type material was required for construction purposes at other facilities, such as the tailing’s storage and plant. During the creation of the mine sequence, an area in the north-western sector of the Project, within Pit Stage 4, was identified as a potential borrow pit to meet the Project's construction requirements.

Table 16-12 shows the balanced waste mining schedule for sufficient delivery of NAF rock for construction requirements and creating the waste dump base layer.

16.3.2 Stockpile movements

The Stage 1 life of mine total material movement schedule, inclusive of stockpile reclaim is listed in Table 16-9.

Annual stockpile movements are listed in Table 16-10, whilst the separately stockpiled auriferous mineralisation is listed in Table 16-11.

16.3.3 Processing schedule

The Stage 1 life of mine processing schedule is listed in Table 16-13. In both of Table 16-10 and Table 16-13, the italicised figures for stockpile reclaim include 362.1 Mt of deferred mineralised waste feed into the plant in the final nine years of Project life

Figure 16-4 Stage 1 40 Mtpa mined ore types and grades from within the ultimate pit design

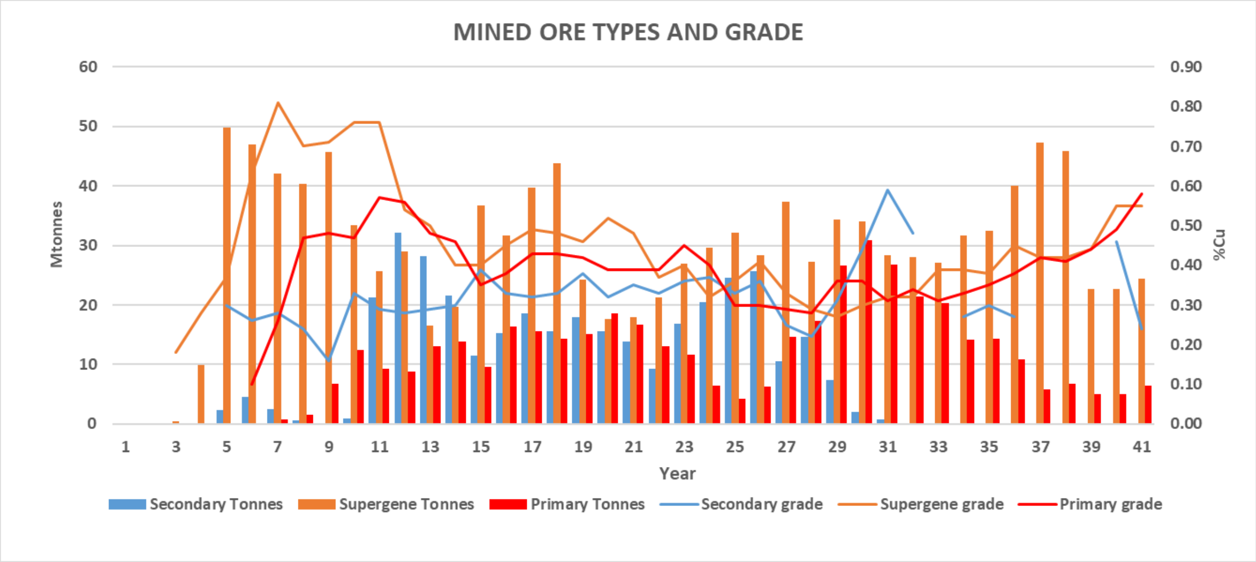


Table 16-8 Stage 1 40 Mtpa life of mine NAF and PAF waste mining schedule

Year	Stage	Waste Mined											
		NAF		PAF		Dom 102 = NAF		Inf Resource = PAF		Total NAF		Total PAF	
		(Mbcm)	(Mt)	(Mbcm)	(Mt)	(Mbcm)	(Mt)	(Mbcm)	(Mt)	(Mbcm)	(Mt)	(Mbcm)	(Mt)
1	Pre-strip	2.8	7.0	3.9	10.1	0.9	2.2	0.0	0.0	3.7	9.2	3.9	10.2
2		10.6	25.8	22.9	58.9	3.8	9.6	0.0	0.0	14.4	35.4	22.9	58.9
3		5.3	12.8	48.7	126.1	3.1	7.9	0.0	0.1	8.4	20.6	48.7	126.2
4		7.5	18.2	51.2	134.0	3.8	9.5	1.4	3.8	11.2	27.7	52.6	137.8
5	Production	9.4	23.9	43.2	113.4	2.2	5.7	1.7	4.6	11.7	29.5	44.9	118.0
6		13.9	35.2	39.8	102.9	2.2	5.6	0.7	1.8	16.2	40.8	40.5	104.7
7		14.9	37.4	40.6	104.2	3.5	8.6	0.3	0.7	18.4	46.1	40.9	104.9
8		19.7	49.6	39.6	100.6	1.1	2.6	0.5	1.4	20.8	52.2	40.1	102.0
9		18.0	45.2	37.6	95.4	0.7	1.9	0.4	1.1	18.8	47.1	38.0	96.5
10		20.1	49.3	38.8	97.0	0.9	2.2	1.4	3.7	21.0	51.5	40.2	100.7
11		24.4	61.3	29.2	74.3	0.0	0.1	3.0	7.9	24.5	61.4	32.3	82.2
12		19.3	48.9	27.9	71.8	0.02	0.05	2.9	7.6	19.3	48.9	30.8	79.5
13		23.0	58.4	29.8	77.2			1.4	3.7	23.0	58.4	31.2	80.9
14		18.7	47.5	35.2	92.0			2.6	6.8	18.7	47.5	37.7	98.8
15		23.3	58.1	30.2	78.5			1.8	4.7	23.3	58.1	32.0	83.2
16		18.0	45.2	27.5	70.6			2.9	7.7	18.0	45.2	30.4	78.3
17		14.2	36.3	16.1	42.0			2.5	6.5	14.2	36.3	18.6	48.5
18		11.7	30.2	7.6	19.9			1.8	4.7	11.7	30.2	9.4	24.6
19		13.2	34.2	6.8	18.0			2.0	5.2	13.2	34.2	8.8	23.2
20		15.0	38.4	8.8	22.8			2.2	5.8	15.0	38.4	11.0	28.7
21		15.5	40.0	8.3	21.5			2.0	5.5	15.5	40.0	10.3	27.0
22		13.7	35.1	12.8	33.2			1.5	4.1	13.7	35.1	14.3	37.2
23		5.6	14.8	8.0	20.8			2.0	5.4	5.6	14.8	9.9	26.2
24		3.6	9.2	10.0	26.3			1.1	3.0	3.6	9.2	11.1	29.2
25		1.2	3.1	11.3	29.8			0.7	2.0	1.2	3.1	12.0	31.8
26		0.3	0.9	12.1	32.1			1.0	2.8	0.3	0.9	13.1	34.9
27		0.2	0.5	11.2	29.9			1.6	4.3	0.2	0.5	12.8	34.2
28		0.3	0.7	12.4	33.1			1.9	5.1	0.3	0.7	14.3	38.3
29		0.3	0.9	9.3	25.1			1.6	4.3	0.3	0.9	10.9	29.5
30		0.8	2.1	8.8	23.9			1.2	3.2	0.8	2.1	10.0	27.2
31		0.7	1.9	5.4	14.5			1.0	2.8	0.7	1.9	6.5	17.2
32		0.1	0.3	3.2	8.4			0.5	1.2	0.1	0.3	3.6	9.6
33		0.1	0.4	3.6	9.6			0.7	2.0	0.1	0.4	4.3	11.5
34		0.2	0.6	3.6	9.5			0.4	1.1	0.2	0.6	4.0	10.6
35		0.1	0.4	3.0	7.8					0.1	0.4	3.0	7.8
36		0.0	0.1	1.8	4.9					0.0	0.1	1.8	4.9
37		0.1	0.1	1.4	3.7					0.1	0.1	1.4	3.7
38		0.0	0.0	1.1	2.8					0.0	0.0	1.1	2.8
39		0.0	0.0	0.6	1.5					0.0	0.0	0.6	1.5
40		0.0	0.0	0.2	0.5					0.0	0.0	0.2	0.5
41		0.1	0.2	0.1	0.2					0.1	0.2	0.1	0.2
42													
43													
44													
45													
46													
47													
48													
49													
50													
51													
52													
53													
54													
Total		**346.0**	**874.0**	**713.6**	**1,848.9**	**22.2**	**56.0**	**46.7**	**124.5**	**368.2**	**930.0**	**760.3**	**1,973.4**

Table 16-9 Stage 1 40 Mtpa life of mine schedule by mining stage and total material movement

Year	Stage	Phase 1 Mined			Phase 2 Mined			Phase 3 Mined			Phase 4 Mined			Mineralised	Domain 102	Inferred	Total Mined			Reclaim	Total Movement		
		Ore (Mt)	Waste (Mt)	Total (Mt)	Ore (Mt)	Waste (Mt)	Total (Mt)	Ore (Mt)	Waste (Mt)	Total (Mt)	Ore (Mt)	Waste (Mt)	Total (Mt)	Waste (Mt)	(Waste) (Mt)	(Waste) (Mt)	Ore +MW (Mt)	Waste (Mt)	Total (Mt)	Ore (Mt)	Ore +MW (Mt)	Waste (Mt)	Total (Mt)
1			17.1	17.1											2.2		0.0	19.3	19.3		0.0	19.3	19.3
2	Pre-strip		72.0	72.0					0.1	0.1		12.6	12.6	0.0	9.6	0.01	0.0	94.2	94.2		0.0	94.2	94.2
3		0.3	128.8	129.1		0.1	0.1		2.7	2.7		2.8	2.8	0.2	7.9	0.002	0.5	142.3	142.8		0.5	142.3	142.8
4		7.1	119.0	126.1		4.6	4.6		0.7	0.7		7.0	7.0	2.9	9.5		9.9	140.8	150.8		9.9	140.8	150.8
5		37.9	97.2	135.2		25.4	25.4		0.0	0.0		6.5	6.5	14.3	5.7		52.3	134.9	187.1	7.9	60.1	134.9	195.0
6		40.7	66.0	106.7		33.5	33.5		2.7	2.7		0.0	0.0	10.7	5.6		51.5	107.8	159.3	3.5	54.9	107.8	162.8
7		36.7	20.5	57.2	3.1	69.4	72.6		0.3	0.3		1.8	1.8	5.6	8.6		45.5	100.7	146.2	8.3	53.8	100.7	154.5
8		23.7	1.1	24.8	15.1	119.0	134.1		8.0	8.0		1.1	1.1	3.7	2.6		42.4	131.9	174.3	6.9	49.3	131.9	181.2
9		17.3	0.2	17.6	32.3	139.9	172.2		29.4	29.4		3.1	3.1	3.1	1.9		52.7	174.5	227.2	2.2	54.9	174.5	229.4
10		14.0	0.04	14.0	30.4	108.0	138.4	0.7	115.5	116.3		12.4	12.4	1.6	2.2	0.1	46.7	238.1	284.9	1.1	47.8	238.1	286.0
11		0.6	0.01	0.6	29.5	18.3	47.8	17.4	97.5	114.9		37.7	37.7	8.7	0.1	3.8	56.2	157.5	213.7		56.2	157.5	213.7
12					26.8	0.36	27.2	27.3	83.9	111.2	0.1	36.7	36.8	15.6	0.05	4.6	69.9	125.6	195.4		69.9	125.6	195.4
13					21.8	0.16	21.9	22.1	87.6	109.7	0.2	48.0	48.2	13.7		1.8	57.8	137.5	195.3	3.8	61.6	137.5	199.1
14					15.4	0.06	15.4	24.6	93.3	117.9	0.0	46.2	46.2	15.1		0.7	55.0	140.2	195.3	3.7	58.7	140.2	199.0
15					12.1	0.03	12.1	30.9	39.3	70.1	2.0	97.2	99.2	12.9		1.4	57.8	137.9	195.7	1.9	59.7	137.9	197.6
16					17.4	0.05	17.4	28.5	18.0	46.5	6.4	97.7	104.1	11.1		1.1	63.4	116.8	180.2	6.4	69.8	116.8	186.6
17					17.8	0.04	17.8	37.2	12.4	49.6	8.8	65.9	74.7	10.2		3.7	74.0	82.0	156.0	0.1	74.1	82.0	156.1
18					10.4	0.04	10.4	45.2	7.4	52.5	8.3	42.7	51.0	9.9		7.9	73.8	58.0	131.8	5.3	79.1	58.0	137.1
19					4.4	0.05	4.4	34.1	4.8	38.9	11.1	47.4	58.5	7.6		7.6	57.3	59.9	117.2	6.7	64.0	59.9	123.8
20								34.2	2.4	36.5	9.6	58.9	68.5	8.1		3.7	51.9	64.9	116.8	8.3	60.2	64.9	125.1
21								30.3	3.1	33.5	10.6	58.4	69.0	7.5		6.8	48.5	68.3	116.8	8.3	56.7	68.3	125.1
22								24.1	1.9	26.1	12.0	66.3	78.3	7.5		4.7	43.6	73.0	116.6	8.3	51.9	73.0	124.9
23								29.5	1.7	31.2	17.0	33.9	50.8	9.0		7.7	55.5	43.3	98.8	8.3	63.8	43.3	107.1
24								14.6	0.8	15.3	29.8	34.7	64.6	12.2		6.5	56.6	41.9	98.5	7.9	64.5	41.9	106.4
25								5.4	0.1	5.5	39.7	32.8	72.5	15.8		4.7	60.9	37.6	98.5	4.6	65.5	37.6	103.1
26								7.7	0.2	7.9	39.2	32.8	72.1	13.4		5.2	60.3	38.2	98.5	8.3	68.6	38.2	106.8
27								7.8	0.04	7.8	37.6	30.3	67.9	17.2		5.8	62.6	36.2	98.8	8.3	70.9	36.2	107.1
28								2.4		2.4	35.9	33.9	69.8	20.9		5.5	59.2	39.3	98.5	8.3	67.5	39.3	106.8
29	Production										48.8	26.1	74.8	19.6		4.1	68.4	30.1	98.5	2.6	71.0	30.1	101.2
30											50.3	26.0	76.4	16.6		5.4	67.0	31.5	98.4		67.0	31.5	98.4
31											43.6	16.4	60.0	12.4		3.0	56.0	19.4	75.4	4.1	60.1	19.4	79.5
32											38.3	8.6	47.0	11.2		2.0	49.5	10.6	60.2	4.1	53.7	10.6	64.3
33											39.0	9.9	48.9	8.5		2.8	47.5	12.7	60.2	6.7	54.1	12.7	66.9
34											36.9	10.1	47.0	8.9		4.3	45.8	14.4	60.2	5.7	51.5	14.4	65.9
35											38.1	8.2	46.3	8.9		5.1	47.0	13.3	60.3	4.8	51.8	13.3	65.1
36											44.0	4.9	48.9	7.0		4.3	50.9	9.2	60.2		50.9	9.2	60.2
37											46.9	3.8	50.7	6.2		3.2	53.1	7.1	60.2		53.1	7.1	60.2
38									0.02	0.02	45.0	2.8	47.8	7.7		2.8	52.7	5.5	58.2		52.7	5.5	58.2
39											24.0	1.5	25.5	3.8		1.2	27.8	2.8	30.6	16.1	43.9	2.8	46.7
40											25.4	0.5	25.9	2.3		2.0	27.7	2.5	30.2	14.6	42.3	2.5	44.8
41											29.2	0.4	29.6	1.7		1.1	30.9	1.5	32.4	10.8	41.7	1.5	43.2
42																				40.0	40.0		40.0
43																				40.1	40.1		40.1
44																				40.0	40.0		40.0
45																				40.0	40.0		40.0
46																				40.0	40.0		40.0
47																				40.1	40.1		40.1
48																				40.0	40.0		40.0
49																				40.0	40.0		40.0
50																				40.0	40.0		40.0
51																				40.1	40.1		40.1
52																				40.0	40.0		40.0
53																				40.0	40.0		40.0
54																				43.9	43.9		43.9
Total		178.3	522.0	700.4	236.5	519.0	755.5	424.1	613.8	1,037.9	777.9	1,068.0	1,845.9	373.3	56.0	124.5	1,990.1	2,903.4	4,893.5	722.0	2,712.1	2,903.4	5,615.5

Table 16-10 Stage 1 40 Mtpa life of mine schedule of stockpile movements

Year	Stage	Stockpile - In						Stockpile - Out						Stockpile - Balance					
		HG+LG		Min Waste		Total		HG+LG		Min Waste		Total		HG+LG		Min Waste		Total	
		Ore (Mt)	Cu (%)	Ore (Mt)	Cu (%)	Ore (Mt)	Cu (%)	Ore (Mt)	Cu (%)	Ore (Mt)	Cu (%)	Ore (Mt)	Cu (%)	Ore (Mt)	Cu (%)	Ore (Mt)	Cu (%)	Ore (Mt)	Cu (%)
1		0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00
2	**Pre-strip**	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00	0.0	0.00
3		0.3	0.25	0.2	0.13	0.5	0.19	0.0	0.00	0.0	0.00	0.0	0.00	0.3	0.25	0.2	0.13	0.5	0.19
4		7.1	0.35	2.9	0.14	9.9	0.29	0.0	0.00	0.0	0.00	0.0	0.00	7.4	0.35	3.1	0.14	10.5	0.28
5		20.9	0.30	14.3	0.14	35.3	0.24	7.9	0.51	0.0	0.00	7.9	0.51	20.4	0.24	17.4	0.14	37.9	0.19
6		4.2	0.21	10.7	0.14	14.9	0.16	3.5	0.32	0.0	0.00	3.5	0.32	21.1	0.22	28.2	0.14	49.3	0.17
7		8.1	0.52	5.6	0.14	13.7	0.36	0.02	0.22	8.3	0.14	8.3	0.14	29.2	0.30	25.5	0.14	54.7	0.23
8		5.6	0.17	3.7	0.12	9.3	0.15	3.9	0.82	2.9	0.14	6.9	0.53	30.8	0.21	26.3	0.14	57.1	0.18
9		11.8	0.33	3.1	0.11	14.9	0.28	2.2	0.21	0.0	0.00	2.2	0.21	40.5	0.25	29.3	0.13	69.8	0.20
10		6.3	0.20	1.6	0.12	7.8	0.18	1.1	1.61	0.0	0.00	1.1	1.61	45.7	0.21	30.9	0.13	76.6	0.18
11		7.4	0.22	8.7	0.15	16.1	0.18	0.0	0.00	0.0	0.00	0.0	0.00	53.1	0.21	39.6	0.14	92.7	0.18
12		14.3	0.23	15.6	0.15	29.9	0.19	0.0	0.00	0.0	0.00	0.0	0.00	67.4	0.21	55.2	0.14	122.5	0.18
13		7.9	0.23	13.7	0.15	21.6	0.18	3.8	0.21	0.0	0.00	3.8	0.21	71.5	0.22	68.9	0.14	140.4	0.18
14		3.7	0.19	15.1	0.14	18.7	0.15	3.7	0.22	0.0	0.00	3.7	0.22	71.5	0.21	83.9	0.14	155.4	0.18
15		6.7	0.19	12.9	0.14	19.6	0.16	1.9	0.21	0.0	0.00	1.9	0.21	76.3	0.21	96.8	0.14	173.1	0.17
16		18.7	0.25	11.1	0.14	29.8	0.21	6.4	0.21	0.0	0.00	6.4	0.21	88.5	0.22	108.0	0.14	196.5	0.18
17		23.9	0.30	10.2	0.14	34.1	0.25	0.1	0.78	0.0	0.00	0.1	0.78	112.3	0.24	118.1	0.14	230.5	0.19
18		29.2	0.38	9.9	0.14	39.1	0.32	5.3	0.62	0.0	0.00	5.3	0.62	136.2	0.25	128.1	0.14	264.3	0.20
19		16.2	0.26	7.6	0.13	23.9	0.22	6.7	0.67	0.0	0.00	6.7	0.67	145.7	0.23	135.7	0.14	281.4	0.19
20		12.1	0.23	8.1	0.14	20.2	0.19	8.3	0.29	0.0	0.00	8.3	0.29	149.5	0.23	143.8	0.14	293.3	0.19
21		9.2	0.21	7.5	0.14	16.7	0.18	8.3	0.39	0.0	0.00	8.3	0.39	150.5	0.22	151.3	0.14	301.8	0.18
22		4.4	0.15	7.5	0.11	11.9	0.12	8.3	0.22	0.0	0.00	8.3	0.22	146.6	0.22	158.8	0.14	305.4	0.18
23		14.7	0.27	9.0	0.13	23.7	0.22	8.3	0.22	0.0	0.00	8.3	0.22	153.0	0.22	167.8	0.14	320.8	0.18
24		12.3	0.19	12.2	0.12	24.5	0.16	7.9	0.29	0.0	0.00	7.9	0.29	157.4	0.22	180.0	0.14	337.4	0.17
25		9.8	0.20	15.8	0.12	25.5	0.15	4.6	0.22	0.0	0.00	4.6	0.22	162.5	0.22	195.8	0.14	358.3	0.17
26		15.2	0.27	13.4	0.13	28.6	0.20	8.3	0.21	0.0	0.00	8.3	0.21	169.4	0.22	209.2	0.14	378.6	0.17
27		13.6	0.19	17.2	0.12	30.8	0.15	8.3	0.29	0.0	0.00	8.3	0.29	174.7	0.22	226.4	0.13	401.1	0.17
28		6.5	0.20	20.9	0.13	27.5	0.15	8.3	0.21	0.0	0.00	8.3	0.21	173.0	0.22	247.3	0.13	420.3	0.17
29	**Production**	11.4	0.20	19.6	0.13	31.0	0.16	2.6	0.21	0.0	0.00	2.6	0.21	181.7	0.21	266.9	0.13	448.7	0.17
30		10.3	0.20	16.6	0.12	27.0	0.15	0.0	0.00	0.0	0.00	0.0	0.00	192.1	0.21	283.6	0.13	475.6	0.17
31		7.6	0.20	12.4	0.12	20.0	0.15	4.1	0.21	0.0	0.00	4.1	0.21	195.5	0.21	296.0	0.13	491.5	0.16
32		2.5	0.19	11.2	0.12	13.7	0.13	4.1	0.21	0.0	0.00	4.1	0.21	193.9	0.21	307.2	0.13	501.0	0.16
33		5.7	0.21	8.5	0.12	14.1	0.16	6.7	0.21	0.0	0.00	6.7	0.21	192.8	0.21	315.7	0.13	508.5	0.16
34		2.6	0.21	8.9	0.12	11.5	0.14	5.7	0.21	0.0	0.00	5.7	0.21	189.7	0.21	324.6	0.13	514.3	0.16
35		2.8	0.21	8.9	0.12	11.7	0.14	4.8	0.21	0.0	0.00	4.8	0.21	187.7	0.21	333.4	0.13	521.2	0.16
36		4.0	0.21	7.0	0.12	10.9	0.15	0.0	0.00	0.0	0.00	0.0	0.00	191.7	0.21	340.4	0.13	532.1	0.16
37		6.9	0.22	6.2	0.12	13.1	0.17	0.0	0.00	0.0	0.00	0.0	0.00	198.6	0.21	346.6	0.13	545.2	0.16
38		5.0	0.41	7.7	0.13	12.7	0.24	0.0	0.00	0.0	0.00	0.0	0.00	203.6	0.22	354.3	0.13	557.9	0.16
39				3.8	0.13	3.8	0.13	16.1	0.27	0.0	0.00	16.1	0.27	187.5	0.21	358.1	0.13	545.6	0.16
40				2.3	0.13	2.3	0.13	14.6	0.21	0.0	0.00	14.6	0.21	172.9	0.21	360.4	0.13	533.3	0.16
41				1.7	0.13	1.7	0.13	10.8	0.21	0.0	0.00	10.8	0.21	162.1	0.21	362.1	0.13	524.2	0.16
42								40.0	0.21	0.0	0.00	40.0	0.21	122.1	0.21	362.1	0.13	484.2	0.15
43								40.1	0.21	0.0	0.00	40.1	0.21	82.0	0.21	362.1	0.13	444.1	0.15
44								40.0	0.21	0.0	0.00	40.0	0.21	42.0	0.21	362.1	0.13	404.1	0.14
45								40.0	0.21	0.0	0.00	40.0	0.21	2.0	0.21	362.1	0.13	364.1	0.13
46								2.0	0.21	*38.0*	*0.13*	40.0	0.13	0.0	0.00	324.1	0.13	324.1	0.13
47										*40.1*	*0.13*	40.1	0.13			284.0	0.13	284.0	0.13
48										*40.0*	*0.13*	40.0	0.13			244.0	0.13	244.0	0.13
49										*40.0*	*0.13*	40.0	0.13			204.0	0.13	204.0	0.13
50										*40.0*	*0.13*	40.0	0.13			164.0	0.13	164.0	0.13
51										*40.1*	*0.13*	40.1	0.13			123.9	0.13	123.9	0.13
52										*40.0*	*0.13*	40.0	0.13			83.9	0.13	83.9	0.13
53										*40.0*	*0.13*	40.0	0.13			43.9	0.13	43.9	0.13
54										*43.9*	*0.13*	43.9	0.13			0.0	0.00	0.0	0.00
Total		**348.6**	**0.26**	**373.3**	**0.13**	**722.0**	**0.19**	**348.6**	**0.26**	**373.3**	**0.13**	**722.0**	**0.19**						

Table 16-11 Stage 1 40 Mtpa stockpile schedule, auriferous mineralisation

Year	Stage	Stockpile - Balance		
		Auriferous		
		Inv. (Mt)	Cu (%)	Au (g/t)
1	Pre-strip	2.2	0.17	0.43
2		11.8	0.06	0.46
3		19.7	0.05	0.44
4		29.2	0.08	0.42
5	Production	34.9	0.08	0.43
6		40.5	0.07	0.42
7		49.1	0.06	0.42
8		51.7	0.06	0.41
9		53.6	0.06	0.41
10		55.8	0.06	0.41
11		55.9	0.06	0.41
12		56.0	0.06	0.41
Total		**56.0**	**0.056**	**0.410**

Table 16-12 Stage 1 40 Mtpa NAF construction volumes and waste dump base layer schedule

Suggested reschedule to varying height dump base laye	Cut Volume '000 lcm	Fill Volume '000 lcm	Pre-strip/construction Year 1 '000 lcm	Year 2 '000 lcm	Year 3 '000 lcm	Year 4 '000 lcm	Production > Year 5 '000 lcm	Year 6 '000 lcm	Year 7 '000 lcm	Year 8 '000 lcm	Year 9 '000 lcm	Year 10 '000 lcm	Total Volume '000 lcm
Construction rock excavated													
PAF waste excavated	503,305		5,308	30,937	65,778	71,035	60,623	54,693	55,203	54,099	51,367	54,262	**503,305**
NAF waste excavated	194,920		4,928	19,373	11,335	15,180	15,781	21,821	24,829	28,061	25,317	28,294	**194,920**
Power line access road (Neil's design)	500			500									**500**
Total NAF rock excavated	**195,420**		**4,928**	**19,873**	**11,335**	**15,180**	**15,781**	**21,821**	**24,829**	**28,061**	**25,317**	**28,294**	**195,420**
NAF waste to dump, varying height base layer	*79,340*		*4,928*	*16,105*	*8,980*	*14,303*	*15,781*	*17,821*	*1,422*				***79,340***
NAF base layer height (m)			*10.0*	*8.3*	*7.9*	*9.4*	*10.0*	*14.0*					
Balance of NAF waste available	**116,080**		**0**	**3,768**	**2,355**	**877**	**0**	**4,000**	**23,407**	**28,061**	**25,317**	**28,294**	**116,080**
Construction rock required													
In concrete		100		50	50								**100**
Airfield on salar		740		740									**740**
Borefield and airport access roads		250		250									**250**
Camp and switchyard		150		75	75								**150**
Site roads		600		600									**600**
MSA		663		663									**663**
Explosives facilities		100			100								**100**
TSF access road		130			130								**130**
TSF embankment (Neil's design)		4,000						4,000					**4,000**
Power line access road (Neil's design)		390		390									**390**
Other (off-site)		3,877		1,000	2,000	877							**3,877**
Total NAF rock required		**11,000**	**0**	**3,768**	**2,355**	**877**	**0**	**4,000**	**0**	**0**	**0**	**0**	**11,000**
Balance of NAF rock available for construction			**0**	**0**	**0**	**0**	**0**	**0**	**23,407**	**28,061**	**25,317**	**28,294**	**105,080**

Table 16-13 Stage 1 40 Mtpa life of mine processing schedule

Year	Stage	Pit to Mill (Direct Feed)				Stockpile Reclaim				Processing			
		Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)	Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)	Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)
1	Pre-strip												
2													
3													
4													
5	Production	17.0	0.66	86.29	0.10	7.9	0.50	39.93	0.08	**24.9**	**0.61**	**71.60**	**0.09**
6		36.5	0.77	97.09	0.12	3.5	0.31	41.99	0.07	**40.0**	**0.73**	**92.31**	**0.11**
7		31.8	0.95	133.79	0.17	8.3	0.14	21.59	0.04	**40.1**	**0.78**	**110.56**	**0.14**
8		33.1	0.84	130.99	0.13	6.9	0.52	64.01	0.11	**40.0**	**0.78**	**119.50**	**0.13**
9		37.8	0.83	184.62	0.15	2.2	0.21	77.23	0.08	**40.0**	**0.80**	**178.74**	**0.15**
10		38.9	0.77	203.20	0.17	1.1	1.57	335.17	0.31	**40.0**	**0.79**	**206.76**	**0.18**
11		40.1	0.71	147.23	0.13	0.0	0.00	0.00	0.00	**40.1**	**0.71**	**147.23**	**0.13**
12		40.0	0.60	127.53	0.12	0.0	0.00	0.00	0.00	**40.0**	**0.60**	**127.53**	**0.12**
13		36.2	0.53	102.50	0.10	3.8	0.21	100.69	0.10	**40.0**	**0.50**	**102.32**	**0.10**
14		36.3	0.49	103.47	0.10	3.7	0.21	98.62	0.09	**40.0**	**0.46**	**103.02**	**0.10**
15		38.2	0.51	134.48	0.11	1.9	0.21	101.14	0.09	**40.1**	**0.50**	**132.91**	**0.11**
16		33.6	0.58	132.39	0.12	6.4	0.21	105.98	0.10	**40.0**	**0.52**	**128.16**	**0.12**
17		39.9	0.59	136.42	0.13	0.1	0.77	147.54	0.09	**40.0**	**0.59**	**136.43**	**0.13**
18		34.7	0.59	120.06	0.12	5.3	0.62	192.59	0.17	**40.0**	**0.59**	**129.65**	**0.12**
19		33.4	0.57	124.85	0.14	6.7	0.67	169.32	0.17	**40.1**	**0.59**	**132.26**	**0.14**
20		31.7	0.56	108.35	0.11	8.3	0.29	118.37	0.11	**40.0**	**0.50**	**110.42**	**0.11**
21		31.7	0.54	104.15	0.10	8.3	0.39	131.58	0.11	**40.0**	**0.51**	**109.82**	**0.10**
22		31.7	0.46	140.73	0.11	8.3	0.22	100.85	0.07	**40.0**	**0.41**	**132.48**	**0.10**
23		31.8	0.54	150.76	0.11	8.3	0.21	103.84	0.09	**40.1**	**0.47**	**141.05**	**0.11**
24		32.1	0.50	131.59	0.09	7.9	0.29	139.28	0.09	**40.0**	**0.46**	**133.11**	**0.09**
25		35.4	0.48	109.28	0.08	4.6	0.21	119.23	0.08	**40.0**	**0.45**	**110.43**	**0.08**
26		31.7	0.54	130.55	0.09	8.3	0.21	112.23	0.09	**40.0**	**0.47**	**126.75**	**0.09**
27		31.8	0.46	153.55	0.09	8.3	0.29	127.51	0.08	**40.1**	**0.42**	**148.16**	**0.08**
28		31.7	0.38	119.71	0.07	8.3	0.21	113.48	0.09	**40.0**	**0.34**	**118.42**	**0.08**
29		37.4	0.44	105.45	0.07	2.6	0.21	115.10	0.09	**40.0**	**0.42**	**106.08**	**0.07**
30		40.0	0.45	108.85	0.06	0.0	0.00	0.00	0.00	**40.0**	**0.45**	**108.85**	**0.06**
31		36.0	0.41	136.06	0.08	4.1	0.21	117.32	0.09	**40.1**	**0.39**	**134.14**	**0.08**
32		35.9	0.40	149.58	0.08	4.1	0.21	117.42	0.09	**40.0**	**0.38**	**146.25**	**0.08**
33		33.3	0.44	131.10	0.08	6.7	0.21	117.52	0.09	**40.0**	**0.40**	**128.82**	**0.08**
34		34.3	0.45	136.57	0.07	5.7	0.21	121.05	0.09	**40.0**	**0.42**	**134.36**	**0.07**
35		35.3	0.45	149.41	0.08	4.8	0.21	122.55	0.08	**40.1**	**0.42**	**146.18**	**0.08**
36		40.0	0.51	172.74	0.08	0.0	0.00	0.00	0.00	**40.0**	**0.51**	**172.74**	**0.08**
37		40.0	0.50	175.16	0.08	0.0	0.00	0.00	0.00	**40.0**	**0.50**	**175.16**	**0.08**
38		40.0	0.47	161.40	0.09	0.0	0.00	0.00	0.00	**40.0**	**0.47**	**161.40**	**0.09**
39		24.0	0.49	172.23	0.07	16.1	0.27	133.31	0.08	**40.1**	**0.40**	**156.62**	**0.08**
40		25.4	0.58	164.26	0.08	14.6	0.21	130.66	0.08	**40.0**	**0.44**	**151.99**	**0.08**
41		29.2	0.58	163.60	0.07	10.8	0.21	130.66	0.08	**40.0**	**0.48**	**154.74**	**0.08**
42		0.0	0.00	0.00	0.00	40.0	0.21	130.66	0.08	**40.0**	**0.21**	**130.66**	**0.08**
43		0.0	0.00	0.00	0.00	40.1	0.21	130.41	0.08	**40.1**	**0.21**	**130.41**	**0.08**
44		0.0	0.00	0.00	0.00	40.0	0.21	130.11	0.08	**40.0**	**0.21**	**130.11**	**0.08**
45		0.0	0.00	0.00	0.00	40.0	0.21	130.11	0.08	**40.0**	**0.21**	**130.11**	**0.08**
46		0.0	0.00	0.00	0.00	40.0	0.13	77.70	0.03	**40.0**	**0.13**	**77.70**	**0.03**
47		0.0	0.00	0.00	0.00	40.1	0.13	74.91	0.03	**40.1**	**0.13**	**74.91**	**0.03**
48		0.0	0.00	0.00	0.00	40.0	0.13	74.91	0.03	**40.0**	**0.13**	**74.91**	**0.03**
49		0.0	0.00	0.00	0.00	40.0	0.13	74.91	0.03	**40.0**	**0.13**	**74.91**	**0.03**
50		0.0	0.00	0.00	0.00	40.0	0.13	74.91	0.03	**40.0**	**0.13**	**74.91**	**0.03**
51		0.0	0.00	0.00	0.00	40.1	0.13	74.91	0.03	**40.1**	**0.13**	**74.91**	**0.03**
52		0.0	0.00	0.00	0.00	40.0	0.13	74.91	0.03	**40.0**	**0.13**	**74.91**	**0.03**
53		0.0	0.00	0.00	0.00	40.0	0.13	74.91	0.03	**40.0**	**0.13**	**74.91**	**0.03**
54		0.0	0.00	0.00	0.00	43.9	0.13	74.91	0.03	**43.9**	**0.13**	**74.91**	**0.03**
Total		**1,268.1**	**0.56**	**136.89**	**0.10**	**722.0**	**0.19**	**98.05**	**0.06**	**1,990.1**	**0.42**	**122.80**	**0.09**

Plant feed profile and recovered copper profile

Figure 16-5 shows the plant feed profile, with a short ramp-up in the first year of processing. Also shown is the direct feed (i.e., direct from the pit) and stockpile rehandle contributions to the plant feed profile. The ratio is approximately 64% to 36%. Figure 16-5 also highlights the overall feed grade trend attributable to the selection of the pit stages and mining/reclaim sequence.

Figure 16-5 Stage 1 40 Mtpa chart of scheduled plant feed profile

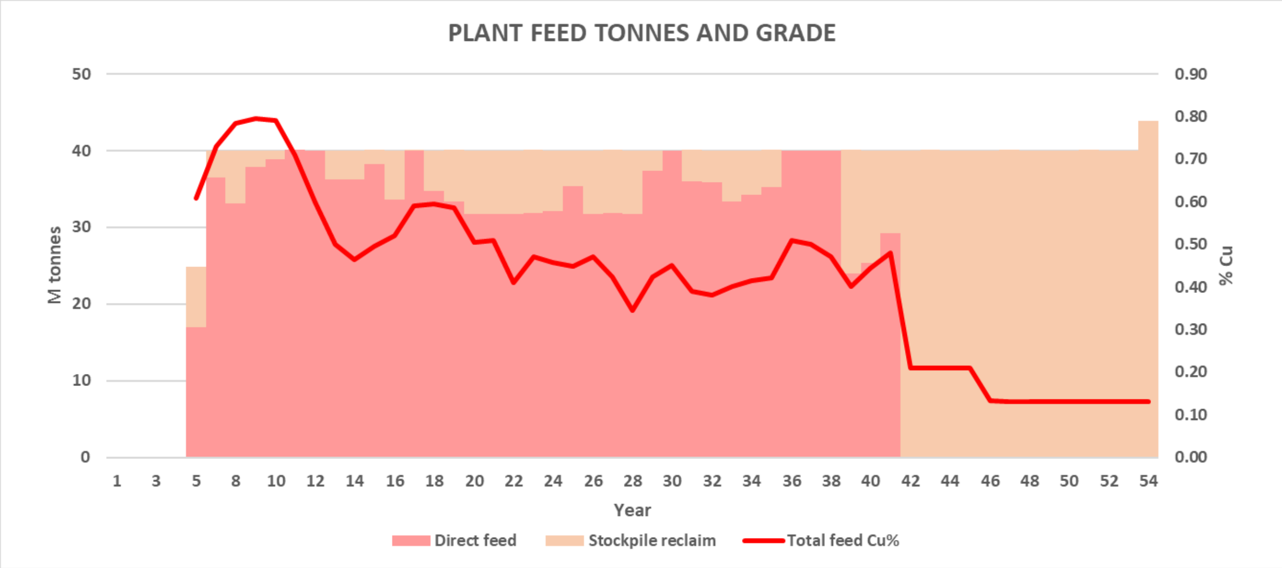


Figure 16-6 shows the annual recovered copper tonnages relative to the recovery rates carried in the model. Figure 16-7 shows the cumulative recovered copper trend; the average annual copper recovery for the first ten years of processing is 245.9 kt.

Figure 16-8 and Figure 16-9, show respectively, the recovered molybdenum metal and recovered gold (i.e., recovered into concentrate) profiles over the life of the Project.

Figure 16-6 Stage 1 40 Mtpa chart of scheduled recovered copper

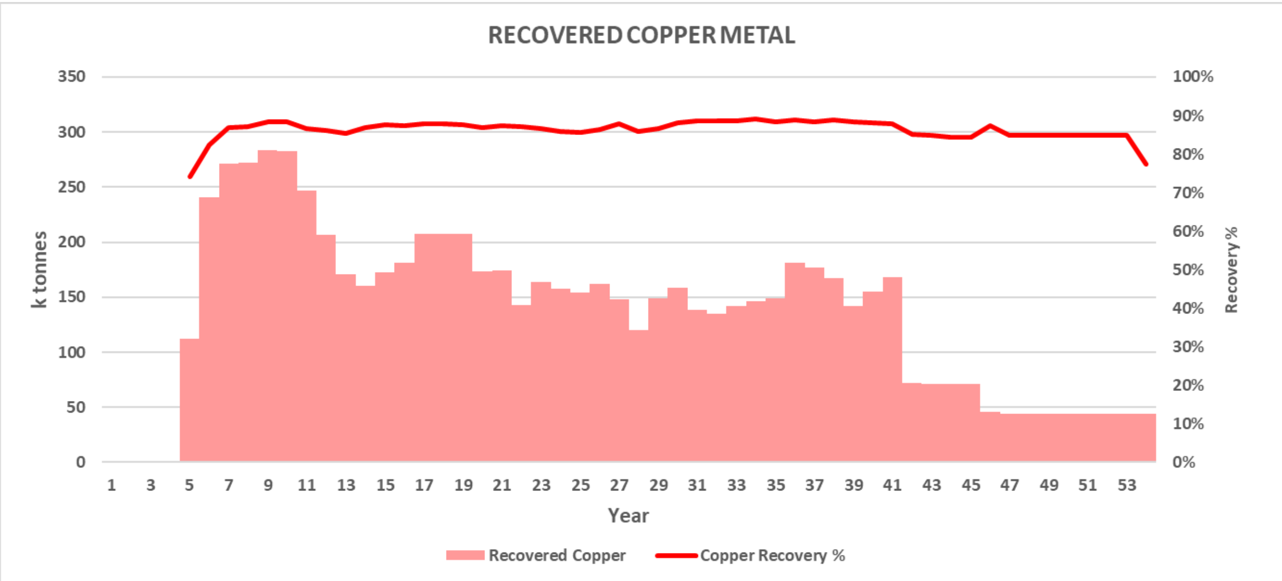


Figure 16-7 Stage 1 40 Mtpa chart of cumulative recovered copper

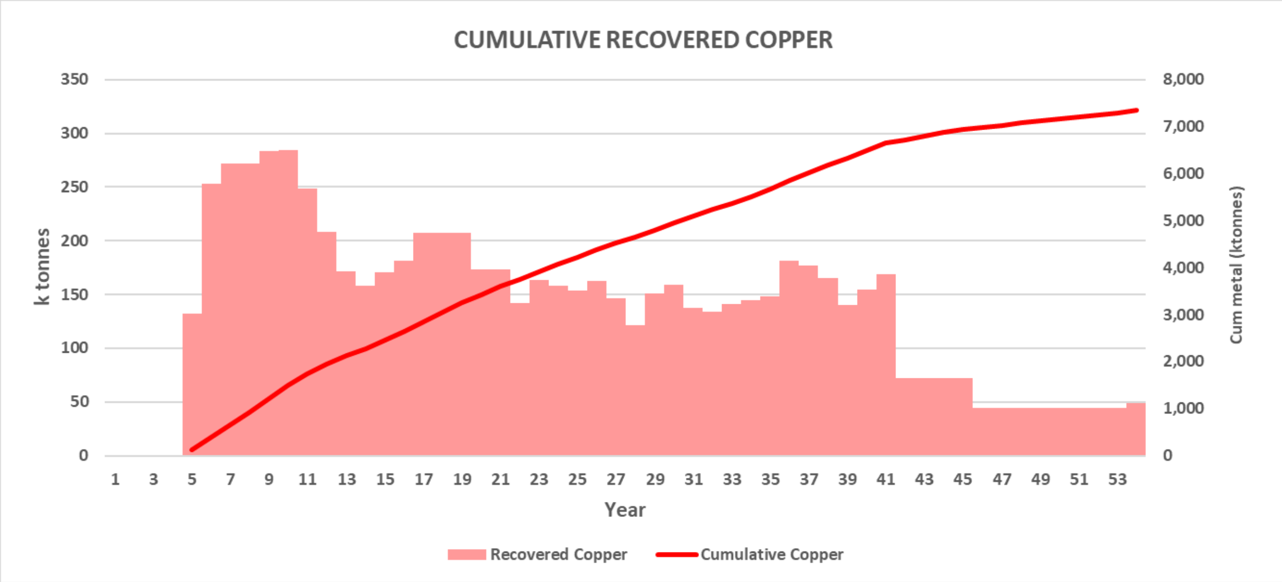


Figure 16-8 Stage 1 40 Mtpa chart of scheduled recovered molybdenum

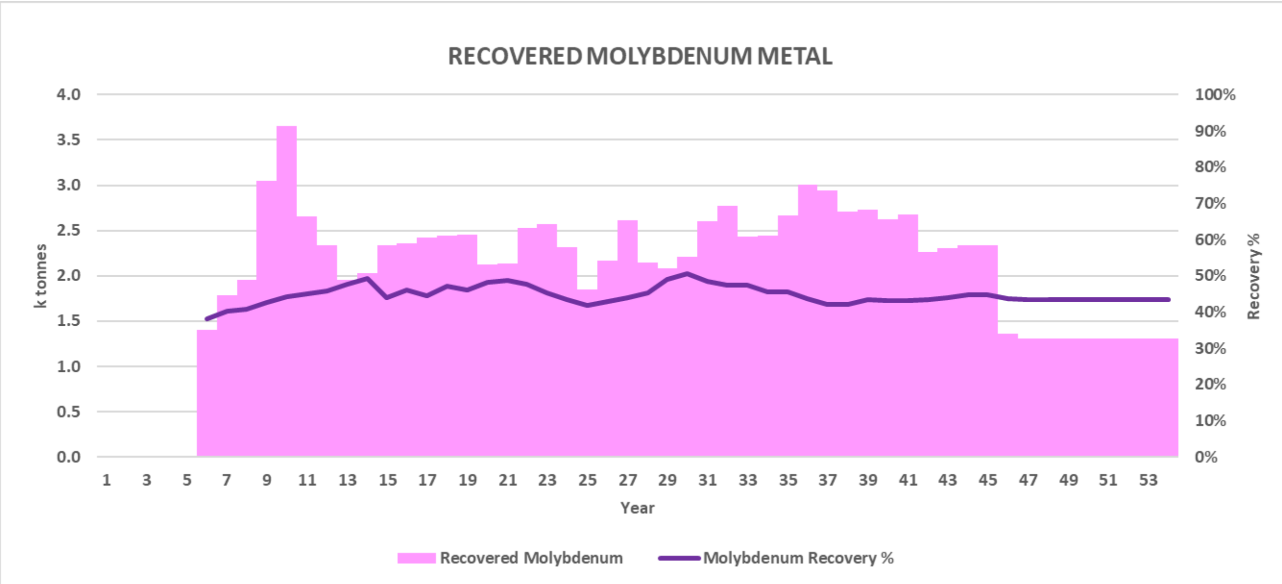
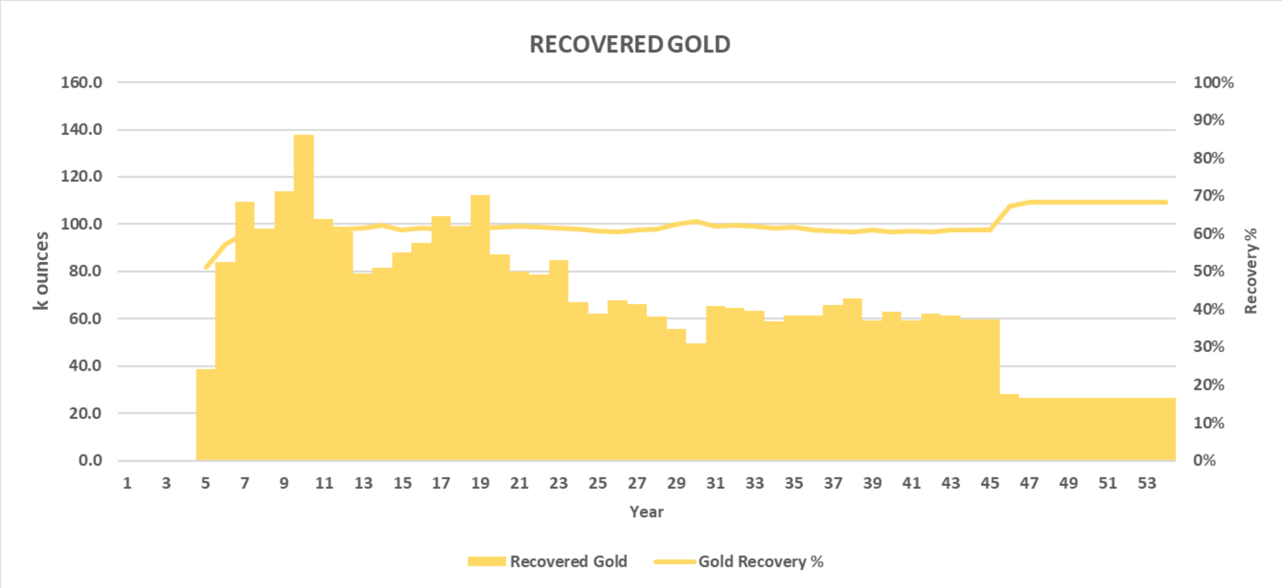


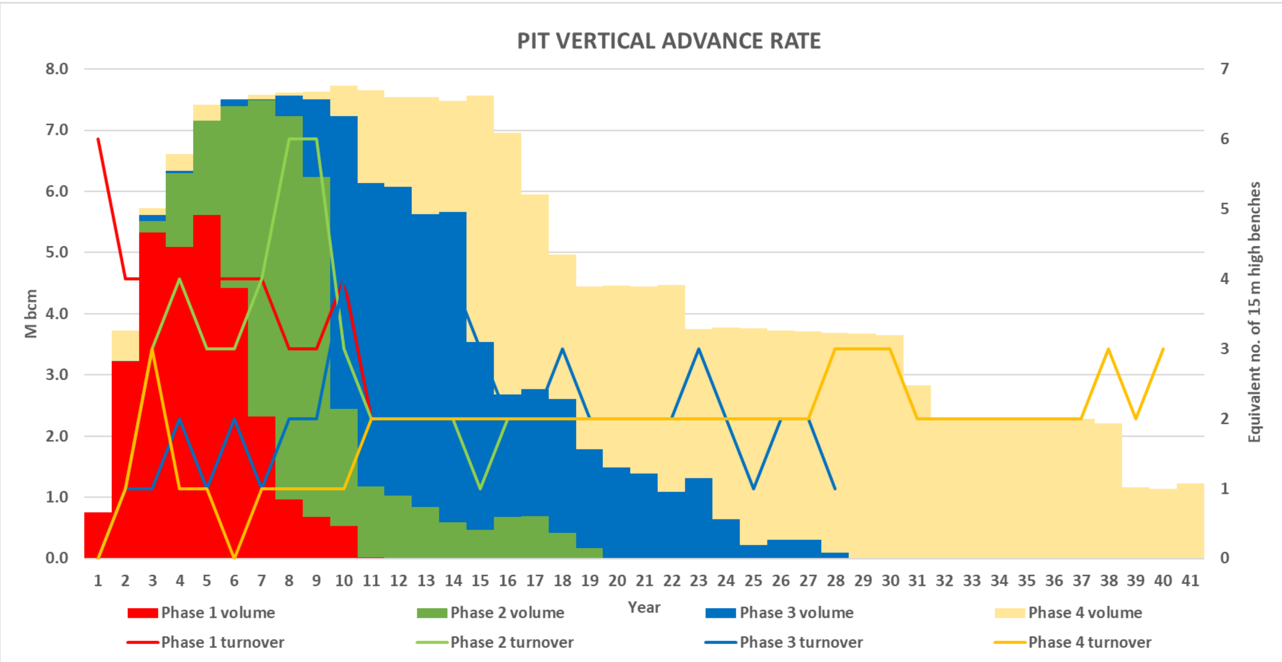
Figure 16-9 Stage 1 40 Mtpa chart of scheduled recovered gold



Pit vertical advance rates

Figure 16-10 is a chart showing the annual pit vertical advance rates in terms of bench turnover in each pit stage. In the instance, the bench turnover is calculated as the equivalent number of 15 m high benches; in reality, there can be multiple benches being mined in any one year that are not excavated to full depth.

Figure 16-10 Stage 1 40 Mtpa pit vertical advance (bench turnover) rates



16.4 Stage 2 60 Mtpa production schedule

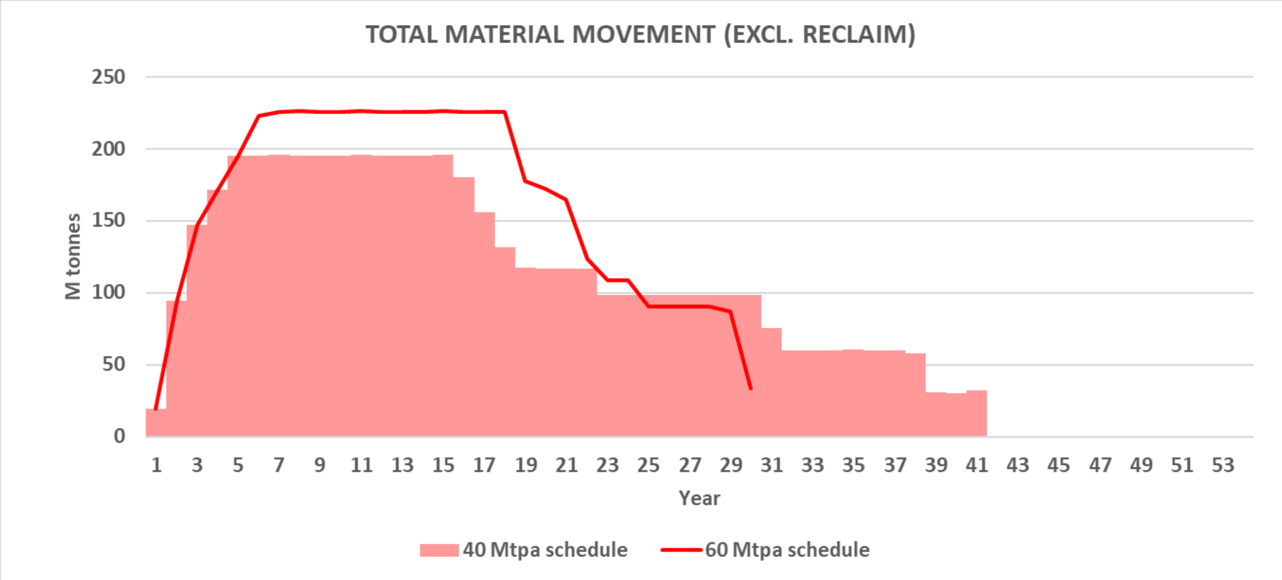
A preliminary Stage 2 60 Mtpa production schedule was produced, featuring an expansion from Stage 1 at Year 9 (i.e., five years after the 40 Mtpa commissioning).

The following commentary provides comparative information respective to the Stage 1 40 Mtpa production schedule.

16.4.1 Total mining material movement

Figure 16-11 is a comparison chart showing annual total material movement, defined as the total mined ore and waste, including mineral waste but excluding stockpile reclaim.

Figure 16-11 Comparison chart of annual total mined material movement



16.4.2 Process plant throughput and grade profile

The plant processing duration for Stage 1 and Stage 2 is 50 years and 35 years, respectively (Figure 16-12).

Figure 16-13 is a comparative chart of the LOM plant feed copper grade profile. In both scenarios, and by virtue of the optimised and staged pit designs, the better feed grades are scheduled during the first decade of processing. In terms of total plant feed at 60 Mtpa and relative to the 40 Mtpa profile (after mining dilution/recovery):

- the weighted average copper grade reduces to 0.66% Cu for the first eight years from initial commissioning,
- then 0.51% Cu to Year 17,
- then 0.46% Cu to Year 22,
- then 0.44% Cu to Year 30,
- and finally to 0.16% Cu for the remaining years of Project life when reclaiming from longer term stockpiles

Figure 16-12 Comparison chart of annual process plant feed tonnes

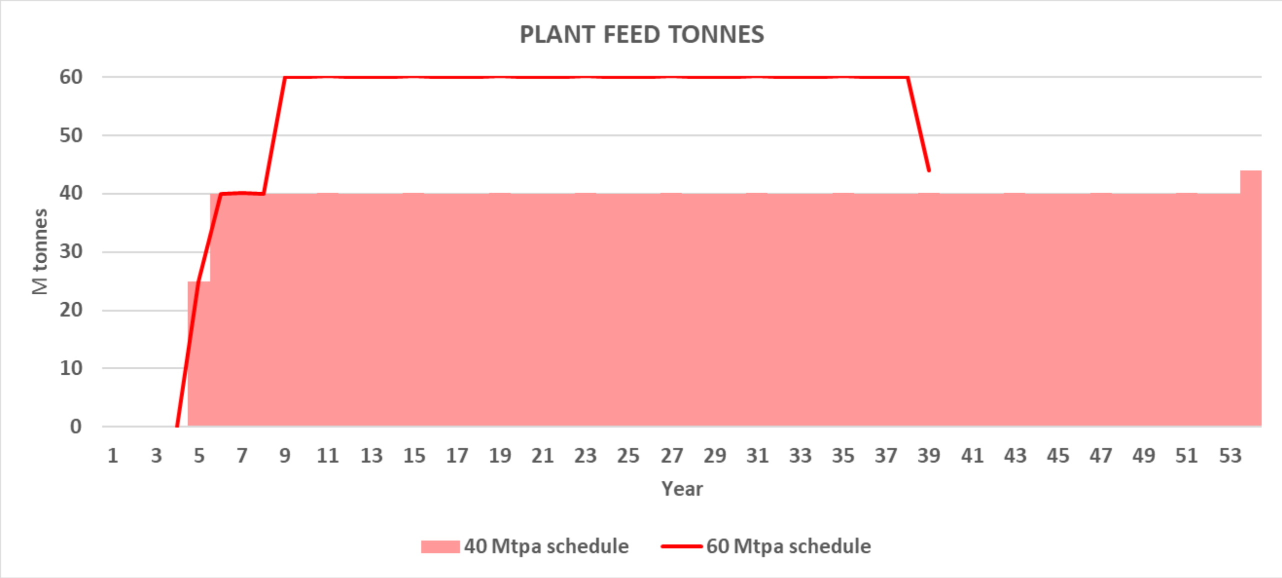
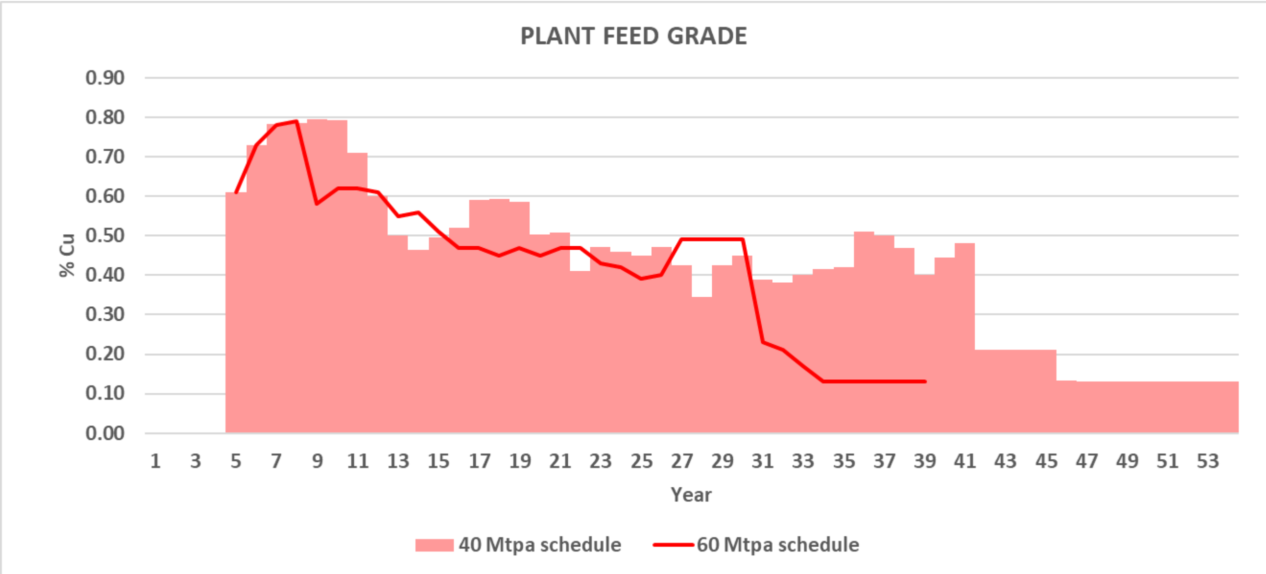


Figure 16-13 Comparison chart of annual process plant feed Cu grade



16.4.3 Recovered copper metal profile

Figure 16-14 and Figure 16-15 show the comparative charts of annual recovered copper and the cumulative annual recovered copper, respectively.

Figure 16-14 shows that the there is a marked increase in annual recovered copper metal to over 320 ktpa during the first ten years of the 60 Mtpa profile. This is a function of the ramp-up profile shown in Figure 16-12. It can be appreciated that the drop in the 60 Mtpa feed grade from Year 9 is compensated for by the increase in processing rate, resulting in the peak of recovered metal over the ensuing five years.

In Figure 16-15, showing the cumulative annual 40 Mtpa Stage 1 and 60 Mtpa Stage 2 metal production, the cumulative total is 7,340 kt reached in Years 54 and 39, respectively.

Figure 16-14 Comparison chart of annual recovered copper

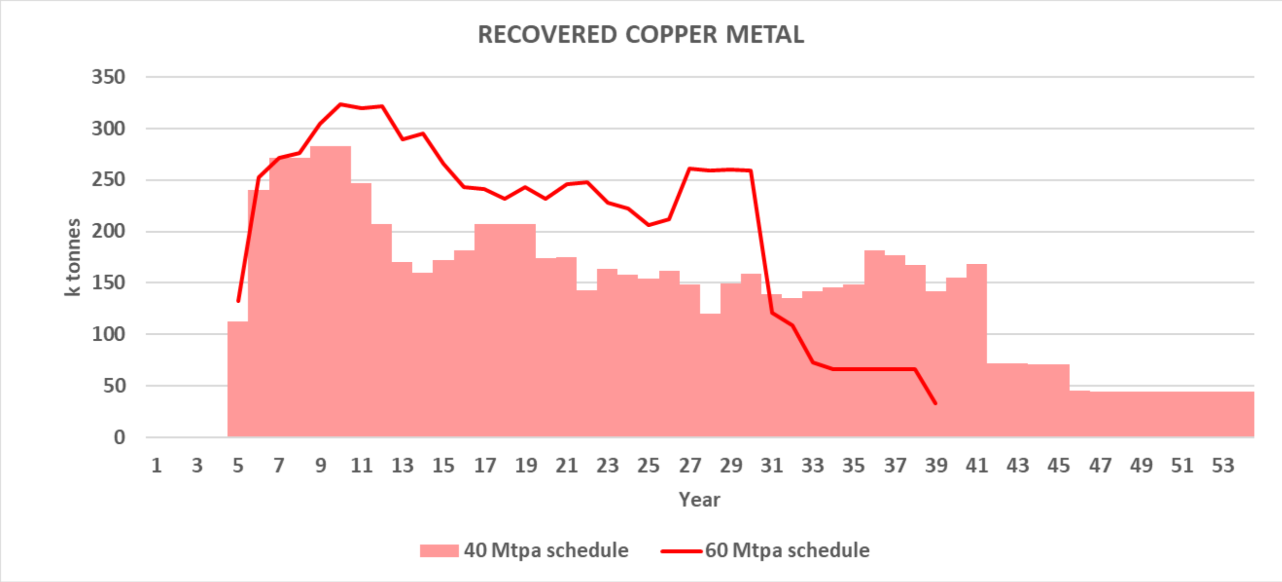
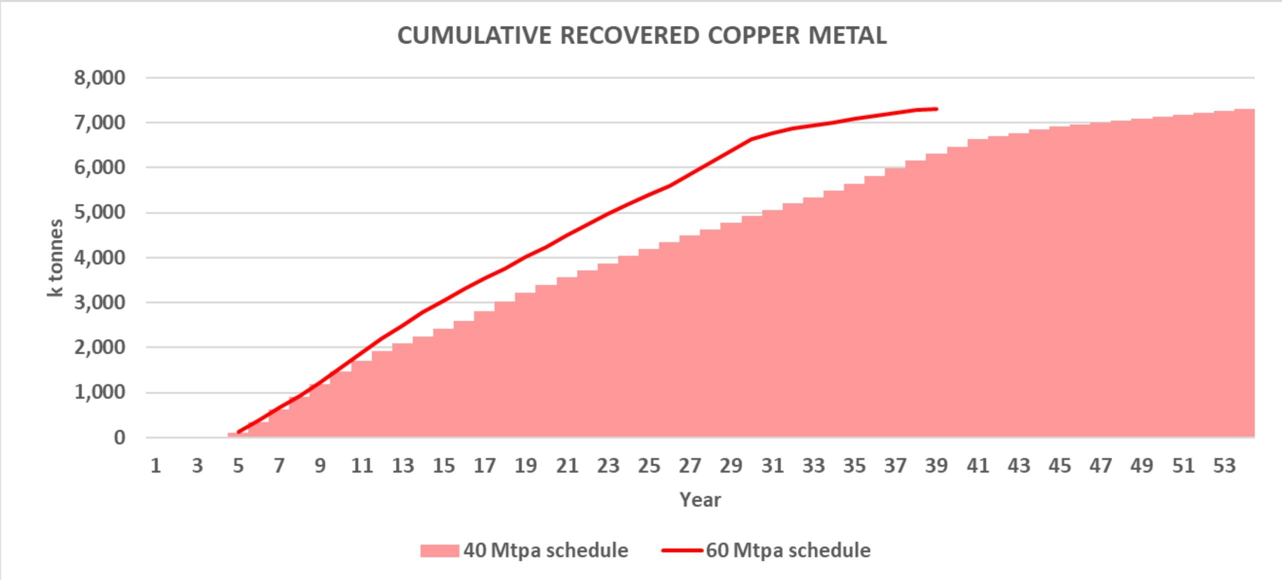


Figure 16-15 Comparison chart of cumulative annual recovered copper



16.4.4 Discussion on the 60 Mtpa scenario

The above scenario comparison charts indicate that a logical time to consider a possible expansion to Stage 2 60 Mtpa processing, with a corresponding mining material movement increase, would be from Year 9 (i.e., five years after Stage 1 commissioning).

Maintaining the 60 Mtpa throughput rate from Year 9, whilst increasing the total annual mined ore and waste accordingly, could achieve higher peaks of recovered metal to compensate for an apparent drop in plant feed grades from this time.

16.5 Mining equipment requirements

16.5.1 Equipment productivity assumptions

An estimate of the mining fleet requirements, specifically new and replacement items over the life of operations has been completed for the Stage 1 40 Mtpa production schedule.

Basic productivity information has been adopted as listed in Table 16-14, followed by truck haulage considerations as listed in Table 16-15, and service meter unit (SMU) engine hours as listed in Table 16-16.

Table 16-14 Basic equipment productivity assumptions

Nominal Fleet		Units	Rate
4100XPC rope shovels			
	Availability	%	85
	Utilisation	%	85
	Productivity	t/op hr	4,300 - 5,000
PC8000 face shovel			
	productivity	t/op hr	3,000 - 3,500
PC3400 excavator			
	productivity	t/op hr	2,300 - 2,700
WE2350 FEL			
	productivity	t/op hr	2,000
930E trucks			
	Availability	%	85
	Utilisation	%	85
	Productive time	%	70
	Productivity	t/op hr	500

Table 16-15 Truck haulage assumptions

Sector	Gradient (%)	Loaded speed (kph)	Unloaded speed (kph)
Downhill	-10	25	35
Flat inside pit	0	20	24
Flat outside pit	0	45	45
Ramp; no TA	10	13	15
Ramp; with TA	10	22	25

16.5.2 Estimated equipment requirements

Table 16-17 to Table 16-20 list the requirements for mining equipment to suit a Stag1 1 40 Mtpa operation.

Some specific comments on these requirements are as follows:

- The fleet will comprise a rope shovel from the outset of mining, with additional units supplemented in the immediately following years.
- These primary digging units will be supplemented with face shovels and a supporting front end loader.
- Although the front end loader requirement is listed up Year 30, it is assumed that stockpile reclaim duties thereafter could be fulfilled by one or several rope shovels as each approaches the end of its useful operating life.

Table 16-16 Service Meter Unit (SMU) engine hour assumptions

Nominal Fleet		Hours
Drills		
	Large	78
	Small	78
Shovels		
	Rope	150
	Face	100
Excavators		75
Front-end Loaders		72
Trucks		
	Large	120
	Small	90
Tracked Dozers		
	Large	70
Wheel Dozers		
	WD900	60
Graders		
	18m	60
Water Carts		
	120	60

Table 16-17 Stage 1 40 Mtpa estimated drilling fleet requirements

Year	Stage	Scheduled drill metres	Drilling Fleet								
			Large Rig			Small Rig			Pre-split Rig		
			Required	New	Replace	Required	New	Replace	Required	New	Replace
1	Pre-strip	122,554	1	1		1	1		1	1	
2		602,709	2	1		1			2	1	
3		928,984	4	2		2	1		4	2	
4		1,113,933	4			3	1		4		
5	Production	1,436,911	4			5	2		4		
6		1,448,061	4			5			4		2
7		1,432,292	4			5			4		2
8		1,425,209	4			6	1		4		
9		1,477,727	4			6			4		
10		1,464,854	4			6			4		2
11		1,499,534	4			6			4		2
12		1,544,640	4			6			4		
13		1,487,107	4		4	6			4		
14		1,463,921	4			6		1	4		2
15		1,487,624	4			6		1	4		2
16		1,415,641	4			6		1	4		
17		1,303,771	3			6		2	4		
18		1,139,765	2			6			3		1
19		981,693	2			5			3		2
20		957,264	2			5			3		
21		940,941	2			5			3		
22		926,139	1			5			3		1
23		865,420	1			5			3		2
24		875,564	1			5			3		
25		893,617	1			5			3		
26		884,474	1			5		1	3		1
27		890,294	1			5			3		2
28		869,579				5		1	3		
29		907,431				5		1	3		
30		895,898				5	1		3		1
31		715,340				5			2		1
32		598,595				4			2		
33		584,319				4			2		
34		579,166				4			2		1
35		586,374				4			2		1
36		605,849				4			2		
37		615,427				4			2		
38		601,513				4			2		
39		317,352				3			1		
40		312,659				3			1		
41		341,263				3					
42											
43											
44											
45											
46											
47											
48											
49											
50											
51											
52											
53											
54											
	Total	**39,541,408**	**4**	**4**	**4**	**6**	**7**	**8**	**4**	**4**	**25**

Table 16-18 Stage 1 40 Mtpa estimated digging fleet requirements

Year	Stage	Ore+waste Mined (Mt)	Ore Reclaimed (Mt)	Digging Fleet								
				Rope shovel			Face shovel			FEL		
				Required	New	Replace	Required	New	Replace	Required	New	Replace
1	**Pre-strip**	19.3		1	1					1	1	
2		94.3		2	2		1	1		1		
3		147.3		4	1		1			1		
4		171.6		5	1		1			1		
5	**Production**	195.2	7.9	5			2	1		2	1	
6		195.2	3.5	5			2			2		
7		195.8	8.3	5			2			2		
8		195.2	6.9	5			2			2		
9		195.2	2.2	5			2			2		
10		195.2	1.1	5			2			2		
11		195.8		5			2			2		
12		195.2		5			2			2		
13		195.2	3.8	5			2			2		1
14		195.2	3.7	5			2			2		
15		195.8	1.9	5			2		1	2		
16		180.2	6.4	5			2			2		
17		156.0	0.1	4			2			2		
18		131.8	5.3	4			1			2		
19		117.1	6.7	3			1			2		
20		116.8	8.3	3			1			2		
21		116.8	8.3	3			1			2		
22		116.6	8.3	3			1			2		
23		98.8	8.3	3						2		1
24		98.5	7.9	3		1				2		
25		98.5	4.6	3						2		
26		98.5	8.3	3						2		
27		98.8	8.3	3						2		
28		98.5	8.3	3						2		
29		98.5	2.6	3		1				2		
30		98.4		3						2		
31		75.4	4.1	3						1		
32		60.2	4.1	2						1		
33		60.2	6.7	2						1		
34		60.2	5.7	2						1		
35		60.3	4.8	2						1		
36		60.2		2								
37		60.2		2								
38		58.2		2								
39		30.6	16.1	2								
40		30.2	14.6	2								
41		32.4	10.8	2								
42			40.0	2								
43			40.1	2								
44			40.0	2								
45			40.0	2								
46			40.0	2		1						
47			40.1	2								
48			40.0	2								
49			40.0	2								
50			40.0	2								
51			40.1	2								
52			40.0	2								
53			40.0	2								
54			43.9	2								
	Total	**4,893.5**	**722.0**	5	**5**	**3**	2	**2**	**1**	2	**2**	**2**

Table 16-19 Stage 1 40 Mtpa estimated truck fleet requirements

Year	Stage	Ore+waste Mined (Mt)	Ore Reclaimed (Mt)	Trucking Fleet					
				Truck 290t			Truck 92-120t		
				Required	New	Replace	Required	New	Replace
1	Pre-strip	19.3		9	9		6	6	
2		94.3		25	16		8	2	
3		147.3		32	7		8		
4		171.6		36	4		8		
5	Production	195.2	7.9	39	3		8		
6		195.2	3.5	39			8		
7		195.8	8.3	39			8		
8		195.2	6.9	40	1		8		
9		195.2	2.2	48	8		8		
10		195.2	1.1	48			8		
11		195.8		48			10	2	
12		195.2		48			10		
13		195.2	3.8	48			10		
14		195.2	3.7	47			9		
15		195.8	1.9	47			9		
16		180.2	6.4	45			9		7
17		156.0	0.1	40			9		
18		131.8	5.3	35			9		
19		117.1	6.7	33			9		
20		116.8	8.3	33			9		
21		116.8	8.3	33		10	9		
22		116.6	8.3	33			9		
23		98.8	8.3	29		4	9		
24		98.5	7.9	29		2	9		
25		98.5	4.6	29		1	9		1
26		98.5	8.3	30	1	3	9		
27		98.8	8.3	33	3		9		1
28		98.5	8.3	34	1		9		
29		98.5	2.6	36	2		9		
30		98.4		36			9		
31		75.4	4.1	30			4		2
32		60.2	4.1	25			4		
33		60.2	6.7	25			4		
34		60.2	5.7	25			4		
35		60.3	4.8	25			4		
36		60.2		27			4		
37		60.2		28	1		4		
38		58.2		28			4		
39		30.6	16.1	19					
40		30.2	14.6	19					
41		32.4	10.8	19					
42			40.0	8					
43			40.1	8					
44			40.0	8					
45			40.0	8		4			
46			40.0	8		1			
47			40.1	8					
48			40.0	8					
49			40.0	8					
50			40.0	8					
51			40.1	8					
52			40.0	8					
53			40.0	8					
54			43.9	8					
	Total	**4,893.5**	**722.0**	**48**	**56**	**25**	10	**10**	**11**

Table 16-20 Stage 1 40 Mtpa estimated ancillary fleet requirements

Year	Stage	Ancillary Fleet											
		Dozers			Wheel Dozers			Graders			Water Carts		
		Required	New	Replace	Required	New	Replace	Required	New	Replace	Required	New	Replace
1	Pre-strip	1	1		1	1		1	1		2	2	
2		5	4		2	1		3	2		3	1	
3		8	3		4	2		4	1		5	2	
4		9	1		4			5	1		6	1	
5	Production	11	2		5	1		6	1		7	1	
6		11			5			6			7		
7		11			5			6			7		
8		11			5			6			7		
9		11			5		1	6			7		
10		11		2	5		1	6		1	7		3
11		11		2	5		2	6		2	7		2
12		11		3	5			6		1	7		1
13		11		2	5		1	6		1	7		1
14		11			5			6			7		
15		11			5			6			7		
16		10			4			5			6		
17		9			4			5			5		
18		8		1	3			4			4		
19		7		1	3		2	4			4		2
20		7			3			4		2	4		
21		7			3			4		1	4		1
22		7		3	3		1	4			4		
23		6		1	3			3			4		1
24		6			3			3			4		
25		6			3			3			4		
26		6		1	3			3			4		
27		6		1	3		2	3		1	4		2
28		6			3			3		0	4		
29		6			3			3		1	4		1
30		6			3			3			4		
31		5			2			3			3		
32		4		2	2			2			3		
33		4			2			2			3		
34		4		1	2			2			3		
35		4			2		2	2			3		2
36		4		1	2			2		1	3		
37		4			2			2			3		
38		4			2			2		1	3		
39		3			1			2			2		
40		3			1			2			2		
41		3			1			2			2		
42		3		1	1			2			2		
43		3			1			2		1	2		2
44		3		1	1			2			2		
45		3			1			2			2		
46		3			1		1	2			2		
47		3		1	1			2		1	2		
48		3			1			2			2		
49		3			1			2			2		
50		3			1			2			2		
51		3			1			2			2		
52		3			1			2			2		
53		3			1			2			2		
54		3			1			2			2		
	Total	11	11	24	5	5	13	6	6	14	7	7	18

16.6 Mine (and infrastructure) water requirements

Table 16-21 lists the estimated water requirements (and balance) for the mine, camp, site services, and for road maintenance and construction purposes. The basis for these estimates is as follows:

- fresh water for production drilling – 750 L/day/drill for six drills = 4.5 kL/day
- fresh water for equipment wash-down = 10 kL/day assumed
- saline water for dust suppression on mine roads:
  - water cart capacity = 120 kL/load
  - average water cart cycle time = 0.63 hrs
  - average water cart loads per year = 65,830
  - average water consumption = 21.6 ML/day
- potable water for the camp (including the plant and administration areas):
  - average of 2,430 persons on site/day during the three years of construction, consuming 250 kL/day
  - average of 1,630 persons on site/day for the 49 years of production, consuming 130 kL/day
- fresh water for site services:
  - equipment washdown in the plant and other non-mine areas = 10 kL/day assumed
  - concentrate storage and rail loadout dust suppression = 10 kL/day assumed
  - concrete batching = 3 kL/day assumed
- saline water for site road maintenance (other than in the mine):
  - 2 x on-highway truck loads per day at 20 kL/load = 80 kL/day
  - road construction, including laying of sub-base = 9 kL/day assumed

Table 16-21 Stage 1 40 Mtpa demand and balance for mine (and infrastructure) water consumption

Water demand	Average, 40 Mtpa				Peak, 40 Mtpa			
	ML/annum	kL/day	m^3/h	L/s	ML/annum	kL/day	m^3/h	L/s
Mining operations								
fresh water for drilling	1.6	4.5	0.2	0.1	4.9	13.5	0.6	0.2
fresh water for equipment washdown	3.7	10.0	0.4	0.1	11.0	30.0	1.3	0.3
brine for dust suppression	1,089.6	2,985.1	124.4	34.6	2,179.1	5,970.2	248.8	69.1
Subtotal	**1,094.9**	**2,999.6**	**125.0**	**34.7**	**2,195.0**	**6,013.7**	**250.6**	**69.6**
Camp, plant and administration								
potable water	75.6	207.0	8.6	2.4	189.8	520.0	21.7	6.0
Other								
fresh water for site services, construction etc	7.3	20.0	0.8	0.2	21.9	60.0	2.5	0.7
brine for road maintenance	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**36.5**	**100.0**	**4.2**	**1.2**	**94.9**	**260.0**	**10.8**	**3.0**
TOTAL	**1,206.9**	**3,306.7**	**137.8**	**38.3**	**2,479.7**	**6,793.7**	**283.1**	**78.6**
Water loss	**Average, 40 Mtpa**				**Peak, 40 Mtpa**			
	ML/annum	**kL/day**	**m^3/h**	**L/s**	**ML/annum**	**kL/day**	**m^3/h**	**L/s**
Mining operations								
water into the ground	130.7	358.2	14.9	4.1	261.5	716.4	29.9	8.3
evaporation	988.0	2,706.9	112.8	31.3	1,990.6	5,453.8	227.2	63.1
Subtotal	**1,118.8**	**3,065.1**	**127.7**	**35.5**	**2,252.1**	**6,170.2**	**257.1**	**71.4**
Camp, plant and administration								
sewage treatment	76.2	208.7	8.7	2.4	189.8	520.0	21.7	6.0
evaporation	12.6	34.5	1.4	0.4	37.8	103.5	4.3	1.2
Subtotal	**88.8**	**243.2**	**10.1**	**2.8**	**227.6**	**623.5**	**26.0**	**7.2**
TOTAL	**1,207.5**	**3,308.3**	**137.8**	**38.3**	**2,479.7**	**6,793.7**	**283.1**	**78.6**

16.7 Mining consumables

16.7.1 Diesel fuel

Diesel fuel and lubricants for the mining fleet would be delivered by a combination of road and rail tankers to the main storage tanks adjacent to the MSA, which would be sized to provide for two weeks of consumption. Average hourly and annual fuel consumption rates are listed in Table 16-22. Estimated life of mine and annual average fuel consumption for the 40 Mtpa Stage 1 is listed in Table 16-23.

Table 16-22 Estimated diesel fuel consumption rates for the mining fleet

Equipment	Diesel fuel (litres/hour)	Fleet average (hrs/annum)
Pre-Split drill	30.0	13,004
Face shovel	557.8	10,927
Rehandle loader (FEL)	160.1	5,099
Large dump trucks	173.1	170,764
Small dump trucks	131.0	47,349
Dozer	84.5	40,498
Grader	28.0	21,590
Water cart	95.0	26,067
Wheel dozer	84.5	19,242

Table 16-23 Estimated life of mine diesel fuel consumption for the mining fleet

Equipment	LOM Total		LOM Average
	kL	ML	kL/annum
Production drills	15,605	15.6	390.1
Trucks	1,688,965	1,689.0	31,277.1
FELs	44,080	44.1	1,259.4
Ancillaries	438,963	439.0	8,128.9
Support equipment	104,500	104.5	1,935.2
MMUs	21,639	21.6	527.8
Explosives	72,131	72.1	1,759.3
	Total	**2,385.9**	**45,277.9**

16.7.2 Tyres

Tyres would be delivered to the MSA tyre storage, handling and fitting facility. The estimated total average equipment operating hours per annum, the typical tyre life duration and the consumption rates for mining tyres are listed in Table 16-24.

Table 16-24 Estimated lifespan and consumption/annum for mining tyres

Equipment	Fleet average (hrs/annum)	Tyre life (hrs)	Tyres/set (#)	No. of tyres (#/annum)
Large dump trucks	170,764	5,000	6	205
Small dump trucks	47,349	5,500	6	52
Rehandle loader (FEL)	5,099	9,000	4	3
Grader	21,590	3,000	6	44
Water cart	26,067	4,000	6	40
Wheel dozer	19,242	4,000	4	20

The tyre life assumptions in Table 16-24 are conservative when compared against actual operational data from the Company’s Zambian operations. Performance in these operations has been consistently strong across the haul truck fleets, exceeding OEM reference expectations in comparable operating conditions. Despite the challenging seasonal wet conditions experienced in Zambia, this performance reflects effective haul road management, loading practices, and tyre maintenance regimes, resulting in reduced tyre consumption and improved operating cost stability.

16.7.3 Explosives

The design production drilling and blasting patterns for ore and waste rock are likely to be different, and this is reflected in the respective powder factors tabled in Chapter 9. These factors have been used to estimate the explosives consumption listed in Table 16-25.

Table 16-25 Stage 1 40 Mtpa estimated explosives consumption/annum

Year	Volume Mined (Mm^3)		Explosives (t)	Explosives (t)	Total Explosives
	ore	Waste	Ore	Waste	(t)
1 - 4	3.9	148.3	5,450	126,025	131,475
5 - 7	56.3	157.4	77,698	133,775	211,473
8 - 10	53.5	164.9	73,882	140,207	214,089
11 - 13	69.4	151.7	95,783	128,981	224,764
14 - 16	66.5	149.0	91,759	126,688	218,447
17 - 19	77.4	75.4	106,783	64,118	170,902
20 - 22	54.3	77.8	74,977	66,148	141,125
23 - 25	65.3	46.3	90,093	39,394	129,487
26 - 28	68.7	42.9	94,815	36,486	131,300
29 - 31	72.2	30.6	99,655	25,974	125,629
32 - 34	53.9	14.2	74,353	12,091	86,444
35 - 37	57.0	11.2	78,645	9,500	88,145
38 - 40	40.8	4.1	56,353	3,469	59,822
41	11.7	0.6	16,109	470	16,579
Total	**751.0**	**1,074.5**	**1,036,354**	**913,325**	**1,949,679**

ITEM 17 RECOVERY METHODS

17.1 Preferred processing route

Initial testwork programmes undertaken by Lumina looked at both flotation and acid leaching for recovery of copper from supergene and primary ores and from a zone of oxide copper in the leach cap.

Acid leaching gave low copper recoveries, even at fine grind sizes of 80% passing 75 µm, and this work was not continued. All subsequent testwork looked at flotation circuits for recovery of copper, gold and molybdenum into bulk flotation concentrates, followed by separation of the Cu and Mo into individual high grade concentrates for despatch to smelters.

Recent metallurgical testwork conducted by FQM at ALS laboratories confirmed that reasonable flotation recoveries could be achieved for the various mining domains comprising the plant feed.

Flotation tails from some ore domains could contain significant values of acid soluble copper, which might be recovered by tank leaching and solvent extraction/electrowinning (SX/EW) to produce copper cathodes. However, the low flotation tailings grades achieved during the most recent testwork programme suggest that such a process option would not be economically viable.

Leaching of flotation tails has not been tested in the laboratory and is not included in the current process designs.

Approximately 56 Mt of gold bearing material at 0.41 g/t Au (essentially leached of copper) has been identified in the oxide cap and will be mined during the pre-strip phase of mine development. Cyanide leaching testwork indicates about 40% recovery from this material at a crush size of 12 mm (1/2 inch) and 90% at 80% passing 75 µm.

A gold recovery circuit is not included in the current processing plans. However, this material would be stockpiled separately from waste material and subject to extensive future testwork to define whether to treat it at a later stage in the Project life.

The process selection thus comprises a conventional copper concentrator for producing a bulk Cu-Mo concentrate, followed by separation of that concentrate into separate Cu and Mo concentrates.

17.2 Process design basis and design criteria summary

The key process design criteria are provided in Table 17-1. Two sets of grade and recovery numbers are presented in this table; one set describes the peak production years for Cu and Mo grades (Year 10, i.e. five years after the commencement of processing) and the second column provides the average numbers for the life of mine (LOM). Year on year recoveries and grades may vary from the average.

The process plant facilities have been designed for an annual throughput of up to 120,000 tpd, or 40 Mtpa.

Rougher concentrates would be thickened prior to regrind, for density control. Separation of Mo and Cu sulphides would be accomplished by depressing chalcopyrite and floating molybdenum in the molybdenum flotation circuit. Flotation concentrates from this separation would comprise the molybdenum sulphides, and flotation tailings would comprise the copper sulphides.

Flotation concentrates would be dewatered prior to being sent off site for further processing. Copper concentrates would be shipped in bulk by train to a coastal port in Chile, and molybdenum concentrates would be filtered, dried and bagged for transport.

Table 17-1 Summary of key process design criteria

Parameter	Units	Peak Years	LOM
Annual Treatment rate	Mtpa	40	
	dry tpd	120,000	
Crusher Utlization	%	70	
Crusher Throughput	dry tph	7,200	
Mill & Flotation Utilization	%	91.3	
Mill & Flotation Throughput	dry tph	5,000	
Ore Head Grades	% Cu	0.80	0.42
	ppm Mo	203	123
	ppm Au	0.174	0.086
Recoveries to Final Concentrates	Cu %	88.5	87.1
	Mo %	44	44.3
	Au %	61	61.4
Annual Production	Cu tonnes	283,200	143,450
	Mo Tonnes	3,573	2,120
	Au kg	4,246	2,053
Copper Concentrate	% Cu	27.4	25.7
Dry Concentrate Produced	dry tph	129	72
Concentrate Produced (at 10% moisture)	wet tpa	1,148,400	620,000
Molybdenum Concentrate	% Mo	47	47
Concentrate Produced (at 8% moisture)	tpa	8,263	5,120
Primary Grind Size	P_{80} µm	180	
Regrind Size	P_{80} µm	20 to 30	
Mass Recovery to Rougher Cons (nominal)	%	10	
	dry tph	500	
Crusher Work Index (CWi)	kWh/t	8.0	
JK Parameter A*b (for SAG Mill sizing)	kWh/t	68.8	
Rod Mill Work index (RWi)	kWh/t	14.5	
Bond Ball Mill Work index (BWi)	kWh/t	19.0	
Abrasion Index (Ai)	g	0.18	

The flowsheet comprises the following unit processes:

- ore delivery by truck to two gyratory crushers
- primary crushing of ROM ore to a P_{80} size of 112 mm
- stockpiling of crushed ore on an eight hour live capacity stockpile
- SAG and ball milling of crushed ore, with size classification by means of hydrocyclones
- pebble crushing on scats generated from the SAG mills, with crushed pebbles returned to the mill feed conveyors
- rougher and scavenger flotation of cyclone overflow slurry
- thickening of rougher flotation tails to a high density for water recovery
- re-pulping of thickened tailings with brine
- pumping of re-pulped tailings to the tailings storage facility (TSF)
- dewatering of rougher concentrates, prior to regrind, for density control
- regrind of dewatered rougher concentrates in high intensity grinding (HIG) mills
- cleaner flotation of the rougher concentrates to improve the copper grade
- cleaner scavenger tails pumped to final tailings
- Cu – Mo separation of the bulk cleaner concentrates in a molybdenum differential flotation circuit
- dewatering of copper concentrates by thickening and filtration, followed by bulk transportation to off-site smelters

- dewatering of molybdenum concentrates by thickening, filtration and drying, followed by bagging and transportation to off-site smelters
- reagent make-up and dosage systems to support the milling and flotation operations
- water reticulation systems
- compressed air systems to support instrumentation and for automatic valve activation
- low pressure air systems provided by blowers for the flotation cells

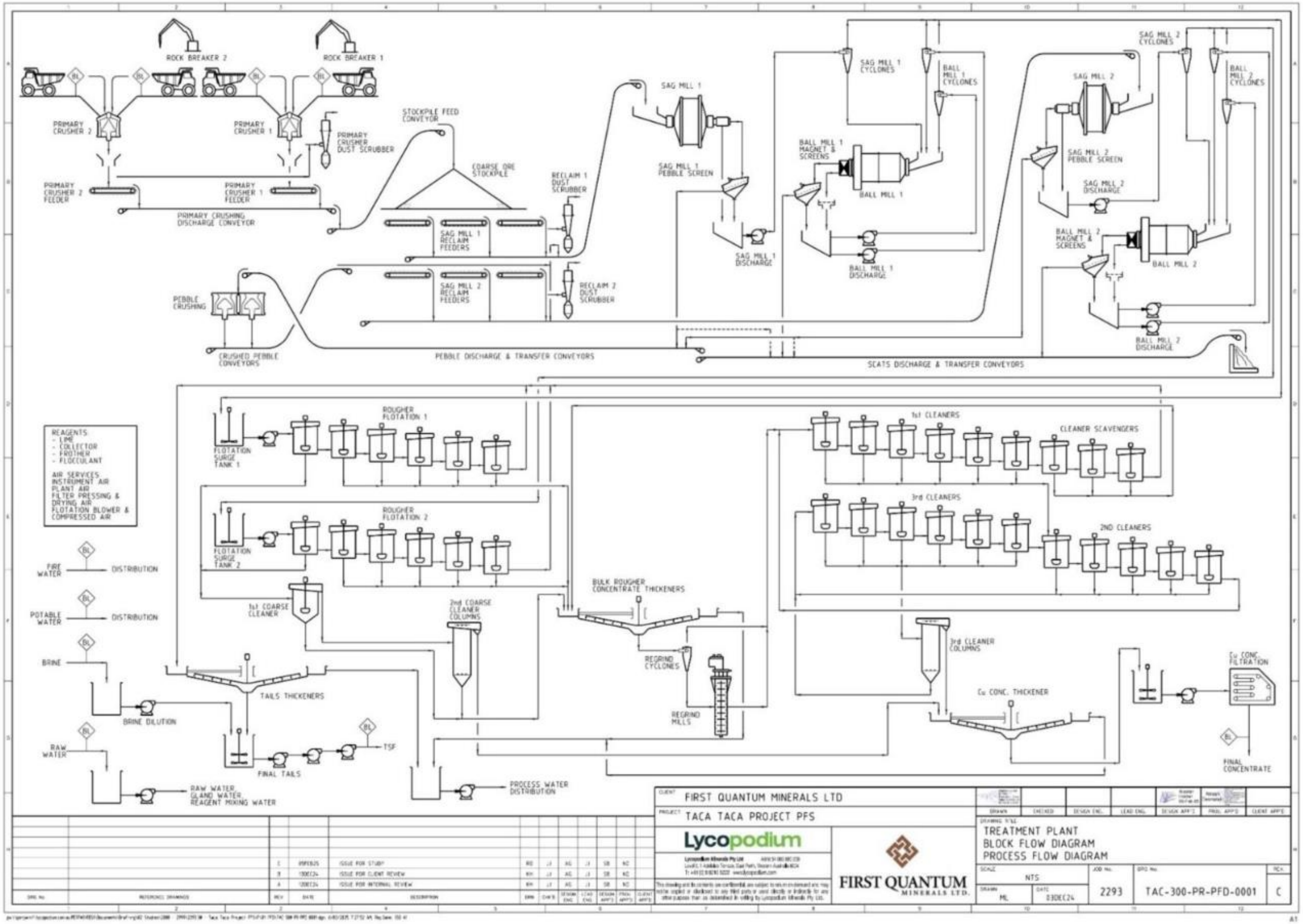
The circuit is designed for an annual Stage 1 throughput rate of 40 Mtpa, with a plant availability of 91.3% (8,000 hrs per year). Hourly throughput rates would be 5,000 tph, which gives a daily throughput (full 24 hours operation) of 120,000 tonnes.

Over the Stage 1 life of mine, an average of 620,000 of wet (10% moisture) tonnes of copper concentrate is expected to be generated annually at an average grade of 25.7% Cu, along with an average of 5,100 wet tonnes of molybdenum concentrate at a grade of 47% Mo, and an average 2,050 kg of gold. At peak production in years 7 to 10 of the operations (i.e., after commissioning in year 5), these figures would increase to an average of 1,179,100 tpa of Cu concentrate at 26.1% Cu, an average of 6,160 tpa of Mo concentrate at 47% Mo, and an average 3,570 kg of gold.

The molybdenum circuit may not be constructed at the same time as the main copper concentrator but may be delayed for a several months to reduce the complexity of the Project and minimise the initial capital outlay.

A detailed process description is provided below, whilst a pictorial flowsheet is presented in Figure 17-1.

Figure 17-1 Flowchart for flotation/concentrator operation



17.2.1 Ore supply and primary crushing

Run-of-mine ore (ROM) will be delivered to the crusher dump pockets by 290 t capacity mine trucks. The live capacity of individual primary crusher dump pockets will be one and a half truckloads (435 t), sufficient for six minutes of operation for a single crusher circuit.

Each dump pocket will allow truck dumping from two dump positions oriented 180 degrees apart to maximise capacity and to facilitate even wear on the primary crusher concaves, shields and liners.

The dumping of ore would be regulated by a crusher operator using the SCADA system, who would activate green ‘Dump’ and red ‘No Dump’ traffic lights to control the rate of dumping into the crusher while maintaining the proper levels in the crusher bowl and primary crusher surge bin below the crusher.

Each crusher dump pocket would be fully enclosed for dust control in the windy climate. Trucks would pass through a flexible plastic curtain, and a dust suppression system would provide a high pressure, mechanically atomised water spray in the dump pocket area and would wet down the ore to control dust in the area. The system would automatically switch-on as truck dumping commences.

A hydraulically operated rock breaker will be installed on the wall of each dump pocket to handle oversized material and any material bridging the crusher feed cavities.

Two primary Metso MK-III 60 - 89 gyratory crushers (or equivalent) would be installed, operating with a nominal open side setting of 165 mm. These crushers would be located on surface adjacent to the pit rim. The crushing circuit would operate 17 hours daily, although 24 hour operation will be possible for periods.

The gyratory crusher product would discharge directly into a surge pocket immediately below each crusher. The ore level in each surge pocket would be controlled to ensure the crushed ore does not fill and contact the crusher underside, or that each crusher discharge feeder is not exposed to falling rock.

Each primary crusher would be equipped with a 50 tonne overhead crane for maintenance purposes, and a rock breaker for dealing with oversize rocks.

High pressure compressed air (850 kPa) would be generated by a local compressor station with air filtered through an in-line filter and stored in an air receiver to ensure a continuous air pressure in the system.

The gyratory crushers will be supplied as a package that includes crusher and spider lubrication systems, lube oil cooling units, hydraulic crusher setting systems, eccentric removal equipment, and all other ancillary equipment for crusher operation and maintenance.

For safety reasons, the primary crushing tunnel will have two entrances and exits and will be ventilated via fans. A dedicated raw water tank and pumps will deliver crushing water services for dust suppression and housekeeping requirements.

17.2.2 Stockpile reclaim

There is space provided on the tipping bench level of the primary crushers for short-term surge stockpiling of ore. The available space allows for reclaim of up to 100 ktonnes.

A long term ore stockpile is also required to balance the optimal mined ore profile against the plant feed profile. During the time of pit operations, there will be a preference for direct tipping of mined ore into the primary crushers. At times, this will be supplemented by stockpile reclaim varying annually between 23 ktonnes and 8.3 Mtonnes. After the cessation of mining there is a period when 40 Mtpa of stockpiled ore is reclaimed into the crushers. The stockpile is located to the west of the pit, adjacent to the workshops and related facilities, and reaches a maximum capacity of 204 Mt in Year 38.

A mineralised waste stockpile (low grade plant feed) is also generated over the life of mining operations. This longer-term feed source is reclaimed into the plant at up to 40 Mtpa between Years 46 and 54. The mined mineralised waste will be stored within an area of the waste dump from which it can be readily reclaimed. The size of the stockpile reaches a maximum of 362 Mt by the time that open pit mining is completed.

Surface caprock containing gold mineralisation is to be stockpiled for possible future processing. This stockpile is also located on the waste dump and reaches a maximum capacity of 56 Mt at Year 12.

17.2.3 Ore transfer to the concentrator

Each crushing circuit will be equipped with a variable speed crusher discharge apron feeder, its speed controlled by the ore level in the crusher surge pocket. Each crusher discharge feeder will discharge directly onto the single crushing discharge conveyor, then onto the overland stockpile feed conveyor, which will transport the ore to the coarse ore stockpile.

The crusher discharge conveyor will also be equipped with a self-cleaning magnet to remove any tramp metal in the ore. The self-cleaning magnet will be an electromagnet belt unit discharging into a tramp metal bunker located beneath the conveyor. The magnet would be followed by a metal detector enabling the belt to be tripped in the event it detects pieces of metal potentially large enough to plug chutes or cause damage to downstream equipment. Weightometers will be installed on the primary crushing discharge conveyor for tonnage accounting.

All conveyors would be partially covered and provided with water sprays for dust control.

Control of the crusher and conveyor circuits would be by programmable logic controllers (PLCs), which would control the start-up and stopping sequences of the conveyors and the speed and electrical load sharing of the drive motors. The motor variable speed drives would operate together and if one drive fails, the system would trip.

17.2.4 Crushed ore stockpile

Crushed ore would be delivered to a conical stockpile by a single stockpile feed conveyor.

The stockpile would have a live capacity of 8 hours (at 40 Mtpa) between the crusher circuits and the concentrator, or approximately 40,000 tonnes. The total capacity of the stockpile (about 120,000 tonnes) could be utilised in the event of ore delivery or crusher problems, by bull-dozing the dead load into the stockpile discharge chutes.

The stockpile itself would be covered with metal cladding for dust control. Several openings would be provided at the base of the cover to allow access for dozing of the pile when/as required.

Due to the high wind speeds at the Project site, all conveyors would be covered and equipped with water sprays at the feed points to suppress dust.

Water cannons would be provided at strategic points for housekeeping. Spillage and washdown would be collected in drive-in sumps located throughout the circuit. Coarse material would settle in the sumps and fines would overflow into vertical-spindle sump pumps. These pumps would deliver spillage to rotoscoops which would discharge grit onto one of the conveyors in the area, producing an overflow that would be pumped to the milling circuit.

Coarse material would be removed from the drive-in sumps by front-end loader and then deposited onto sloped concrete pads for drainage back to the sumps, prior to being returned to the conveyor circuit via feeders and chutes.

All conveyors (and feeders) within the crushing and milling circuits would be equipped with standard operational and safety devices:

- audible alarms to warn of impending conveyor start-up
- safety pull wires along each side of the conveyor
- belt misalignment switches to warn of the conveyor running off centre
- low-speed detection switches to detect potential slipping of the drive pulley
- belt rip detectors
- tilt switch plugged-chute detectors installed in the chute to prevent possible damage due to blockages
- drift switches on each side of the conveyor at the head and tail ends to detect belt misalignment
- belt scrapers at the head pulleys
- belt plough at the tail pulleys

In addition, conveyor belt scales would be installed on selected conveyors to monitor and record the plant ore throughput.

Start-up sirens and pull wire switches would be hard wired. Tilt switches, drift switches and under-speed switches would be interlocked through the process control system.

17.2.5 Grinding circuit

The grinding circuit would consist of two trains each of 20 Mtpa capacity operating to process a total of 40 Mtpa of ore. Each circuit would be designed to treat 2,500 tph of material from a feed size of 80% passing 180 mm to product size of 80% passing about 180 µm. A single circuit is described below, whilst Figure 17-2 shows the proposed arrangement of the comminution circuit, including crushing and grinding.

Mill feed conveyors

Crushed ore would be recovered from the stockpile by the use of variable speed apron feeders. A separate stockpile tunnel would be installed for each SAG mill feed. Feed chutes located in the tunnels underneath the stockpile would be equipped with dust collection and dust suppression facilities on ore transfer chutes, while ventilation fans would be fitted to ensure adequate ventilation of the tunnels.

For safety reasons, the tunnels would be equipped with two entrances and exits. Hosing water services would be provided in the tunnels for housekeeping requirements.

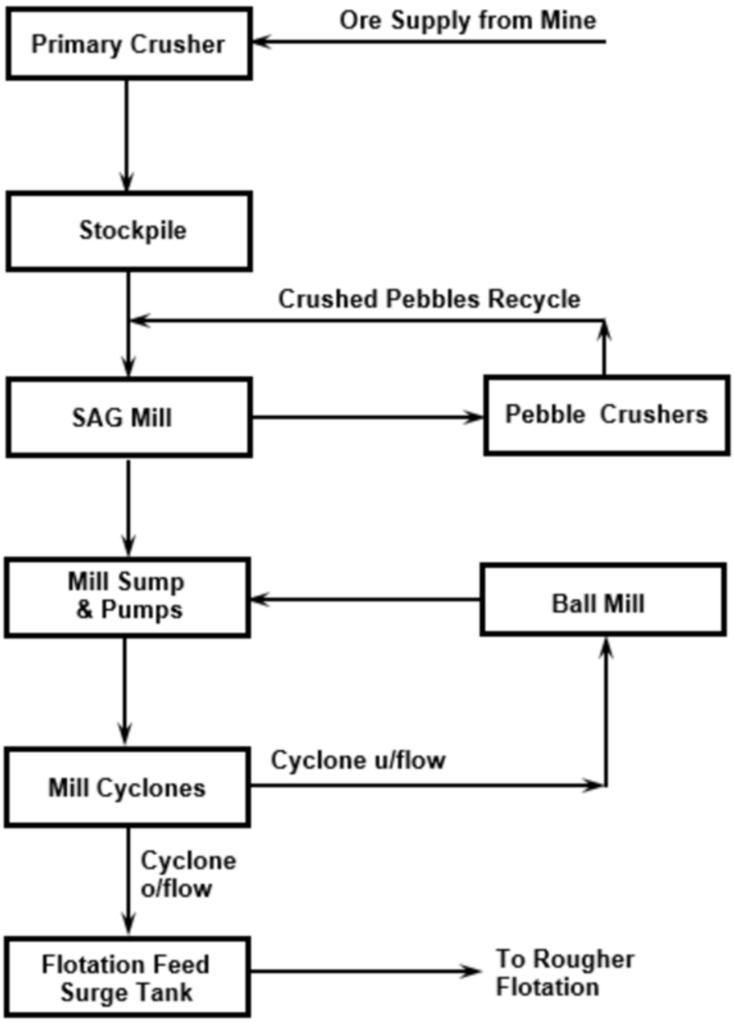
Three apron feeders would be installed in each tunnel; at least two of which would normally be operating for each milling train to give a net mill feed of 2,500 tph. However, the actual feed rate to the mill would be 10 to 20% higher to cater for the recycled SAG mill scats treated through the pebble crusher, and due to the ore moisture content. The feeders would be selected to ensure a consistent ore feed to the mill and to minimise segregation. The feeders would discharge onto the SAG mill feed conveyor, feeding the SAG mill directly. The SAG mill feed conveyor would be equipped with a weightometer, and a variable speed drive. It would also be covered to control dust emissions.

SAG milling

The SAG mill would have an internal shell diameter of 12.19 m (40 ft) and an effective grinding length (EGL) of 8.2 m (26.9 ft) and would be equipped with a 28 MW gearless (wrap-around) drive, providing full variable speed capability for control purposes.

Mill feed rate would be controlled through the plant control system (PCS), using the weightometer on the mill feed conveyor to control on a set feed rate, a set mill weight, or a set mill power. Water addition to the feed end of the mill would be controlled as a set ratio of the mill feed rate to give a fixed slurry density inside the mill.

Figure 17-2 Block diagram for comminution circuit



The SAG mill would be equipped with 50 or 60 mm slotted discharge grates, a discharge trunnion, short trommel screen and a vibrating discharge screen. The trommel screen and vibrating screen would initially have 11 x 40 mm slotted apertures to provide drainage of slurry and prevent large particles entering the mill discharge sump and causing pump damage or line blockages.

Undersize slurry from the SAG mill trommel and discharge screen would gravitate to a mill discharge hopper, which would also receive slurry discharging from the ball mill.

Lime slurry will be added at a controlled rate to each SAG mill feed chute for downstream pH control.

Pebble crushing and recycle

Oversize material from both the SAG mill discharge trommel and the vibrating screen (all minus 50 mm and larger than 11 mm in size) would be collected on a single pebble conveyor and transferred to a pebble crusher feed bin.

Two Metso MP 1250 pebble crushers would be installed for pebble crushing, operating at a close side setting (CSS) of less than 16 mm. The two pebble crushers would have a capacity of about 1,600 tph at these settings, or about 30% of the mill feed rate. If pebble generation rates from the two mills are lower, the crusher gap could be reduced for more efficient crushing and full utilization of the machines.

Crushed pebbles would be conveyed to either or both mill feed conveyor belts, for recycle directly to the SAG Mills.

Circuit designs would allow pebbles to be returned to the milling circuit uncrushed or stockpiled separately to scats for future processing. A pebble reload bin would be provided to return stockpiled pebbles to the circuit, using a front end loader.

Self-cleaning magnets and a metal detector installed on the pebble conveyors would protect the pebble crushers from any steel that may discharge from the SAG mill. A weightometer on this belt would measure the pebble tonnage for metallurgical accounting purposes.

The pebble crushers would be serviced by an overhead crane, and each would be provided with a dedicated lubrication system.

Ball milling

A single ball mill would be installed for each SAG mill. These mills would be identical to the ball mills installed at the Company's Sentinel mine in Zambia, being 8.53 m (28 ft) inside shell diameter by 13.25 m long (43.5 ft EGL) and equipped with a 22 MW gearless drive.

The ball mill would be fed by recirculated mill cyclone underflow. Cyclone underflow density is expected to be about 75% solids, and hence dilution water facilities would be provided at the mill feed end to reduce this to 70% solids for efficient milling.

The gearless drive would provide full variable speed capability, used to control mill power draw.

The ball mill would discharge via a trunnion magnet. Steel scats would be separated and washed and conveyed to a scats stockpile. A common discharge hopper would be provided for each SAG and ball mill combination.

Mill classification circuit

As noted above, a single mill discharge hopper would be installed for each pair of SAG and ball mills.

Additional process water would be added to the mill discharge hopper for cyclone feed density control and/or hopper level control. Mill discharge density is expected to be about 75% solids from the SAG mill and 70% from the ball mill, and this would be diluted down to about 55% solids for efficient classification.

Two variable speed cyclone feed pumps would be installed in duty/standby configuration, feeding a single cyclone cluster. Pump speed would be used to control cyclone feed pressure and hopper level with water addition used to control feed density. Depending on the number of cyclones required for the volumetric throughput, consideration may be given, in the detailed design, to the installation of two duty cyclone feed pumps and two duty cyclone clusters.

Cyclone operation would be controlled by constant density control, or constant flow control, monitored/controlled by a density meter and a flow meter on the cyclone feed line.

The number of operating cyclones would be determined by the volume of slurry flowing to the cyclone cluster, and the cyclone feed pressure. As the flow increases and the cyclone feed pressure builds, additional cyclones would be put online to maintain a constant feed pressure.

The underflow from the cluster of ball mill cyclones would be directed to the ball mill feed chute, with the ability to recycle some of the underflow to the SAG mill. In practice it is expected that about 20% of the cyclone underflow would be directed to the SAG mill and 80% directed to the ball mill.

It is anticipated that the transfer size between the SAG and ball mills would be about 0.6 – 0.8 mm, and the required product size from the ball mill cyclones would be 80% finer than 180 µm.

Cyclone overflow slurry would flow by launder through a cyclone overflow sampling box from which a sample would be recovered for size analysis in a particle size analyser. The analyser would also produce a sub-sample for head grade analysis; for the on-stream analyser (OSA) and a two-hourly composite for the analytical laboratory. Slurry from the cyclone overflow box would be discharged by gravity to a surge tank ahead of the rougher flotation section. A cyclone overflow launder and a surge tank would be installed for each milling circuit, so their individual products could be measured separately.

Steel ball addition

New balls (140 or 125 mm balls) would be added to the SAG mills on a daily basis. This would be via a ball feed bunker and conveying system over the top of the SAG mill feed chute.

65 or 80 mm balls would be added to the ball mills on a daily basis via a separate ball feed bunker and the common conveying system over the top of the ball mill feed chute. Balls would be fed from the bunker at a controlled rate using a ball indexer. A camera and PLC would be used to accurately measure the balls added to each mill on a daily basis.

Mill cooling and lubrication systems

The SAG and ball mills would be equipped with gearless drives, providing variable speed and inching capabilities. Each mill would be supplied with lubrication systems comprising high pressure pumps on the trunnion bearings for mill start-up, and low pressure oil conditioning pumps.

The lubrication systems will be housed in an enclosure adjacent to each mill, and craneage will be supplied for maintenance and lubricant drum handling.

Cooling of the mill motors and the lubrication system would be by a refrigerative system providing chilled water. This water would be operated in closed circuit through a chiller, with the excess heat removed from the refrigerant in a second cooling water circuit. The chiller water would be high quality water, with reagents added to prevent scale build-up. The circuit would be topped up with potable water from a steady-head tank.

The heat load on the cooling system for each mill circuit will be approximately 3 MW (6 MW for the two trains). This heat would be removed by circulating fresh water through the chiller heat exchangers and through a set of evaporative cooling towers. Water make-up to the cooling towers would be from the potable water plant.

Mill area services

The grinding area would be serviced by strategically placed water cannons for hosing and vertical spindle spillage pumps. A trench would run the entire length of the milling building at the discharge end of the mills, leading to a drive in sump at the NW end of the milling, to cope with large volumes of spillage. Excess spillage from the rougher flotation circuit would also flow to this sump. Excess water and fines from the drive-through sump would be pumped to the mill discharge hoppers and settled solids removed by front end loader. The sumps would be equipped with level switches to start and stop the pumps.

Dedicated liner handling machines would also be provided for the SAG and ball mills to assist with mill relining. When not in use, the machines would be stored on the mill operating platform at the feed end of the mills.

The milling area would be serviced by two cranes:

- a 10 t capacity overhead crane servicing the cyclones and the gravity recovery area
- a 110 t capacity semi-portable crane servicing the mills and discharge area, with the capability to handle the mill motor segments as well as the largest mill construction lift

17.2.6 Flotation circuit

The proposed flotation circuit would comprise a standard Cu-Mo flotation flowsheet designs used throughout the industry and in Latin America, including:

- flotation of a bulk rougher flotation concentrate comprising mixed copper and molybdenum sulphides in rougher-scavenger flotation
- dewatering of the rougher concentrates for density control prior to regrind.
- regrind of the clean concentrate to assist in upgrading the concentrates (pyrite liberation from copper sulphide minerals) and for Mo separation
- cleaning of the bulk concentrate in a single cleaner circuit comprising two stages of conventional cells followed by Jameson or Concorde cells and flotation columns.
- Dewatering of the cleaner scavenger tails for water recovery prior to discharge to final tailings
- Cu-Mo separation in a five stage cleaning circuit, by depressing the copper minerals, and floating a molybdenum sulphide concentrate containing minimal copper
- concentrate from the molybdenum flotation circuit would comprise the final Mo concentrate, and the final Cu concentrate is the tailings stream
- dewatering of both concentrates in a thickener and filter
- bulk transport of concentrates by train to the nearest port for export to off-site smelters
- drying and bagging of filtered Mo concentrates prior to shipment

Figure 17-3 shows the proposed block flowsheet for the flotation (and tailings) circuit.

Bulk (copper and molybdenum) rougher and cleaner flotation

Cyclone overflow slurry from each milling circuit would be collected in surge tanks (one per milling train) and distributed between two rougher flotation trains, each comprising an agitated surge tank, a single bank of rougher flotation cells, and a tailings hopper and pumps. Concentrate hoppers and pumps would be installed as common units for the two trains.

Concentrates from both rougher-scavenger trains would be thickened for density control ahead of the regrind circuit. Dewatered concentrates would be reground to 80% passing 20 to 30 µm to fully liberate the fine copper minerals prior to cleaner flotation.

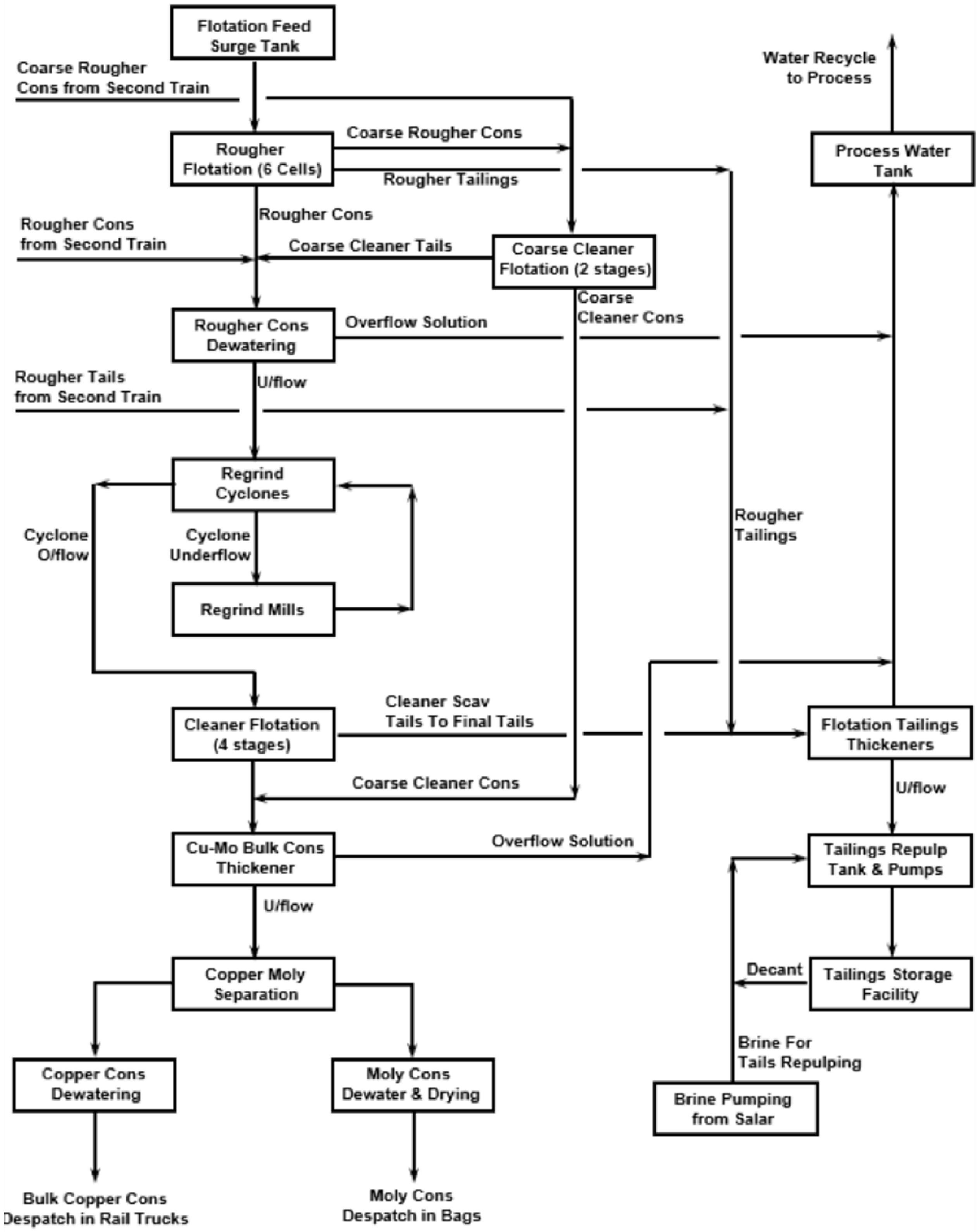
A single four-stage cleaner flotation circuit would produce a bulk Cu-Mo concentrate to be sent for Mo separation in a separate circuit.

An overhead crane would span the two rows of rougher-scavenger cells and run the full length of the rougher-scavenger circuit. A separate crane would service equipment in the cleaner circuit.

Flotation feed surge and conditioning tanks

Cyclone overflow slurry at 80% passing 180 µm and a density of 30% to 35% solids would gravitate from each mill train through a launder and cross stream sampler to a dedicated rougher flotation feed surge tank. These tanks would provide surge capacity (of about 5 minutes when operated half full) between the milling and flotation circuits to ensure a steady feed of material to the flotation cells.

Figure 17-3 Block flowsheet for flotation and tailings



Slurry would be pumped from each surge tank to the respective rougher flotation circuit, each headed by an agitated conditioning tank. Reagents would be added to each conditioning tank including collector, frother, lime for pH control, and possibly a depressant to help decrease the recovery of gangue minerals. The conditioned slurry would flow by gravity into the first of the rougher flotation cells.

Rougher flotation

Each train would consist of a single bank of six 500 m^3 flotation cells. Each flotation cell would have an agitator mechanism driven by an electric motor. The cells in each line would be arranged as individual cells separated by intermediate boxes containing dart valves to control the flow from the bank for level control. The cells would be stepped to allow gravity flow from the feed to the tails discharge.

Concentrate would overflow each flotation cell into a launder encircling the top of the cell, and then into the main concentrate launders. Each flotation cell concentrate launder would be provided with spray water (from the plant process water reticulation system) to break down the froth and assist in slurry flow down the launders.

Concentrates from the two trains would be combined in concentrate hoppers located at the tail end of the flotation trains, to allow easy access to the concentrate pumps for maintenance. Two concentrate hoppers, each equipped with a single pump, and operating in duty/standby configuration would be installed for each concentrate duty.

The concentrate from the first cell in both trains would be sampled and pumped to the copper coarse cleaner surge tank.

Concentrate from the remaining five cells from both trains would be sampled and pumped to the bulk rougher concentrate thickeners ahead of the regrind mills. The concentrate from the second cell in each train will be able to feed either concentrate launder depending on grade. Spray water will be installed on all concentrate launders to break down froth, and aid slurry flow.

Flotation reagents, including frother, depressant (if needed) and a promoter for Mo flotation would be added to each flotation cell as required, using dedicated dosing pumps. Final reagent selection and dosage rates will be determined during the preliminary engineering phase of the Project.

Tailings from the last cell in each train would report to the scavenger flotation tails hopper and be pumped to the feed distributor ahead of the flotation tailings thickeners.

Spillage and washdown in the area would be collected in strategically located sumps and pumped by vertical spindle pumps to an appropriate location within the rougher circuit. Excess spillage would be directed by sloped trenches to the drive in sump, described under Mill Area Services.

Low pressure air (at 65 kPag) would be added to each cell from a set of low pressure blowers provided for all the conventional flotation cells.

Rougher concentrate dewatering

Rougher concentrate would be dewatered by thickening prior to regrind. Two 50 m diameter conventional thickeners would be installed, to avoid the addition of flocculant at this point in the circuit, which could adversely affect subsequent flotation operation. However, settling rates without flocculant are significantly slower than with flocculant, leading to large thickener area requirements. This aspect of process design will be revisited during the preliminary engineering phase of the Project.

The thickener would be equipped with a froth weir to retain floating concentrate particles within the thickener. Process water may be applied as sprays to this froth to assist in its break-up.

Thickener underflow density and bed level would be controlled by varying the thickener underflow pump speed. Thickener overflow would be brine water, and this would be returned to the process water circuit for use in mill dilution.

Thickener underflow at about 50% solids would be pumped by one of two horizontal centrifugal pumps to the regrind mill cyclone feed sump for regrind prior to cleaner flotation.

Bulk concentrate regrind mill

Mineralogical examination has indicated that the Taca Taca mineralisation is fine grained, and a regrind of the rougher concentrates to 80% passing 20 to 30 µm would be required to achieve good concentrate grades at high copper recoveries.

Drawing upon designs for the Company's Cobre Panamá project, it is estimated that approximately 12 MW of regrind capacity would be required for Taca Taca. This would be provided by three 4 MW high intensity grinding (HIG) mills.

Thickened concentrate would be pumped from the dewatering thickeners into the cyclone feed hopper from where it would be pumped to hydrocyclones cutting at 20 to 30 µm. Cyclone overflow would be finished product and would gravitate to a concentrate surge tank ahead of the cleaner flotation circuit.

Cyclone underflow would gravitate to a single regrind mill feed tank alongside process water for mill feed density control and then pumped through three Metso HIG5000 (or similar) high intensity grinding mills operating in a duty / duty / standby configuration. The third regrind mill and spare cyclones installed on each cluster will be used when treating high-grade material.

The grinding chamber would contain grinding media (5 mm ceramic beads) and rotating discs providing momentum to stir the charge against a series of stationary counter discs. The particles would be ground by attrition between the beads from an estimated P_{80} of 160 µm to a P_{80} of 20 µm.

Due to the vertical arrangement of the mill, classification would occur simultaneously throughout the grinding process with larger particles remaining longer at the periphery, whilst smaller particles would move upwards. There would be no external classification of mill product, and no recirculation of mill discharge to the cyclones and back into the mill.

Slurry overflowing the mills would join the feed cyclone overflow slurry, be sampled, and would gravitate to the regrind product hopper and then be pumped to the cleaner flotation circuit.

Copper coarse cleaner flotation

The combined rougher first concentrate from both trains containing fast-floating sulphides would be processed through a single coarse cleaner circuit comprising two stages of cleaning before reporting to the final concentrate.

Feed would report to the coarse cleaner surge tank prior to being diluted to target density and pumped to the first coarse cleaner high-intensity flotation cell, a Metso Concorde CD5500 cell (or equivalent). A variable portion of the cell tails would be recirculated to the coarse cleaner surge tank to provide a constant feed flow rate to the cell.

The first coarse cleaner concentrate would be sampled and pumped to the second coarse cleaner columns, which would comprise two parallel 4.5 m diameter by 14 m high flotation columns. The single feed stream would be evenly distributed, allowing the columns to operate in a duty / duty configuration. Column concentrate would be sampled and pumped to the bulk concentrate thickener or to the copper concentrate thickener if the molybdenum circuit is offline.

The first and second coarse cleaner tails would be pumped to the bulk rougher concentrate thickener and regrind circuit and reprocessed with the lower grade concentrates recovered from the last five rougher and scavenger flotation cells.

Copper cleaner flotation

Reground concentrates would be processed through a single cleaner circuit comprising three cleaning stages: two of conventional cells, followed by a stage of columns.

The circuit would be operated in counter-current fashion, whereby the feed to each stage would comprise concentrate from the previous stage and tailings from the subsequent flotation stage.

Feed to the first cleaner conditioning tank would comprise regrind product and recycled tailings from the second cleaner cells. The conditioning tank would have a conditioning time of one minute before overflowing to the first cleaner cell one feed box. The first stage cleaner and cleaner scavenger circuit would consist of nine 70 m^3 tank cells in a six and three cell configuration.

Concentrate from the 1st cleaner cells will form the feed to the 2nd cleaner flotation circuit, along with the 3rd cleaner column tails. Concentrate from the first cleaner scavenger cells would be recycled to the bulk rougher concentrate thickener for additional regrind processing. The first cleaner scavenger tailings would be pumped to the tails thickeners via the rougher tails hopper.

The second cleaner circuit would have five 70 m^3 tank cells in series.

The third cleaner column circuit would consist of two parallel 4.5 m diameter by 14 m high flotation columns. The single feed stream would be evenly distributed, allowing the columns to operate in a duty / duty configuration. Provision would be made to bypass the 3rd cleaner column circuit if not required. Column concentrate would be pumped to the bulk concentrate thickener after sampling or to the copper concentrate thickener if the molybdenum circuit is offline. Column tails would be combined and recycled to the second cleaner feed.

Each flotation tank cell would have an agitator mechanism driven by an electric motor. Internal dart valves would separate each cell to control slurry flow and level. The cells would be stepped to allow gravity flow from feed to tails discharge.

Reagent additions of frother, collector, lime and possibly a depressant could be made to each cleaner flotation stage via the feed boxes. Reagent addition would be via dedicated peristaltic dosing pumps from the respective reagent day tanks.

Water sprays would be installed above the fourth cleaner columns for concentrate washing, and in each concentrate launder for froth suppression. Water would also be added to each concentrate hopper for density control and line flushing.

Air would be added to each cell from a set of low pressure blowers dedicated to the cleaner circuit.

A series of samplers (pressure and gravity) would split samples from critical streams. The samples would be assayed by the on-stream analysis unit and then returned to the circuit.

Floor sump pumps will handle minor spillage and washdown in the area, returning spillage to the respective cleaner bank. Major spillage will be directed via concrete bunds to the high-grade drive-in sump and returned to the circuit where appropriate.

Cleaner circuit thickener

With the addition of spray water to the concentrate launders and hoppers within the cleaner circuit, slurry densities would be diluted, and this could affect the residence times in the various stages of the cleaner circuit. Thickening of cleaner concentrates at some point in the cleaner circuit would be used to control slurry densities within this circuit, providing optimum slurry residence times in the flotation cells. The location of this thickener within the circuit, and the target density to be achieved would be defined during detailed engineering.

To maintain densities, thickening of a partial stream of concentrate would be required, and a single conventional thickener would be installed. It is important to avoid the addition of flocculant at this point in the circuit, which could adversely affect subsequent flotation operation.

Overflow from the thickener would be recycled to the process water tank; underflow would be pumped by one of two installed cleaner thickener underflow pumps to a density control tank. Unthickened concentrate and water would be added to this tank to provide a controlled slurry density for feed to further stages of cleaner flotation.

17.2.7 Copper – molybdenum separation

The early testwork performed by Lumina indicated that it was possible to separate out a high grade molybdenum concentrate from the bulk Mo –Cu concentrate at acceptable Mo recoveries (45% Mo recovery into a 47% Mo concentrate). However, limited testwork has been performed to assist in defining the specifics for a molybdenum recovery circuit. The process description below describes a circuit which is common in the industry.

The separation of copper and molybdenum is usually achieved by flotation, whereby copper sulphide minerals are depressed by removal of the xanthate collector through the addition of NaHS, and molybdenite is floated using fuel oil as a collector, and pine oil as a frother. Final tails from the circuit (rougher scavenger tails) would be copper concentrates containing between 27 and 25% Cu and minimal molybdenum, and the final concentrates (after five stages of cleaning) would be molybdenite assaying about 47% Mo with low levels of copper.

As noted previously, it is possible that the detailed design and construction of the molybdenum recovery circuit would be delayed for several years to minimise the initial technical complexity of the Project and the capital cost of the processing facilities.

Cu-Mo concentrate thickener

The separation of copper and molybdenum sulphides involves the chemical destruction of the xanthate collector with NaHS. To reduce the consumption of both xanthate and NaHS, a dewatering stage would be installed between the cleaner flotation circuit and the Cu-Mo separation circuit, which would allow recycle of most of the water in the cleaner concentrate slurry, together with the contained flotation reagents to the regrind circuit. Dewatered concentrates would then contain less xanthate solution, leading to reduced NaHS consumption.

Dewatering would take place in conventional thickeners without the use of flocculant, because of the subsequent flotation stages. Two 38 m diameter thickeners would be installed for this duty.

The thickeners would be equipped with froth weirs to retain floating concentrate particles within the thickener. Process water could be applied as sprays to this froth to assist in its break-up.

Thickener underflow at about 65% solids would be pumped by one of two installed horizontal centrifugal pumps to the Cu-Mo conditioning tanks. The variable speed underflow pumps would be controlled to maintain the underflow density and the bed depth in the thickener.

Mo float conditioning tanks

A series of two mechanically agitated conditioning tanks would be used to prepare the slurry for Mo separation from chalcopyrite. Slurry would overflow one tank into the next, and from the last tank into the first Mo rougher flotation cell.

The first conditioning tank would be used to adjust the slurry pH to between 8 and 9 through the addition of dilute sulphuric acid or carbon dioxide, and to adjust the slurry density to approximately 30% solids for flotation.

The second tank would be used for conditioning the molybdenite particles with fuel oil as collector, and for stripping xanthates adsorbed onto the chalcopyrite particles by the addition of NaHS. After the xanthate collector has been removed from the copper mineral surfaces, Mo should float, and the copper minerals depressed in the flotation circuit.

Mo flotation is often carried out in sealed cells to retain the NaHS in circuit, and often under nitrogen to reduce the loss of NaHS through oxidation. No decision has been made in this regard for Taca Taca; all options will be considered in the detail design. However, at the Company's Cobre Panamá Project, the decision was made to use sealed, self-aspirating flotation cells (i.e., Wemco cells) without nitrogen purging and this could be the primary selection for Taca Taca.

Mo rougher and scavenger flotation

Roughing and scavenging Mo flotation would be achieved in two parallel banks of seven conventional flotation cells, being 42.5 m^3 U-shaped cells. The combined concentrate from the first four cells (rougher concentrate) of each bank would be pumped to the head of the first cleaner circuit and concentrate from the last three cells (scavenger concentrate) would be returned to the rougher feed or to the first cleaner scavenger cells, depending on grade.

Scavenger tailings would comprise copper sulphides with low levels of residual molybdenum and would be the final copper product. Tailings would be collected, and entrained NaHS removed prior to being pumped to the copper concentrate thickener for dewatering.

Spray water would be installed on all concentrate launders to break down froth and aid slurry flow.

Fuel oil, pine oil frother and NaHS would be added at various points within this circuit to maintain flotation kinetics of molybdenite and depression of the coper sulphide minerals.

Off gas from the sealed flotation cells and respective slurry hoppers will report to the Mo caustic gas scrubbing system to neutralise hydrogen sulphide gas.

Mo cleaning circuit

Mo cleaner flotation would comprise five stages of conventional cleaner flotation, followed by two columns operating in series. The circuit would be operated in counter-current mode, with Mo concentrate being advanced to the next stage for further cleaning, and the tails being returned to the previous stage upstream for scavenging of residual Mo. Thus, second cleaner tails would be returned to the first cleaners, and the first cleaner scavenger tails would be pumped to the copper concentrate thickener along with the rougher scavenger tails. Similarly, first cleaner concentrate would pass to the second cleaners, until ending up as final concentrate from the fifth stage of cleaning.

The circuit will comprise the following flotation cells:

- 1st stage cleaners and cleaner scavengers 6 x 14.2 m^3 U-shaped cells
- 2nd stage cleaners 6 x 2.8 m^3 U-shaped cells
- 3rd stage cleaners 6 x 2.8 m^3 U-shaped cells
- 4th stage cleaners 4 x 2.8 m^3 U-shaped cells
- 5th stage cleaners 4 x 2.8 m^3 U-shaped cells
- Column cleaners 1 column 2.1 m diam., and 1 of 1.6 m diam.; both 12.5 m tall

The second column concentrate would comprise the final molybdenum concentrate, which would gravitate to the moly multi-stream analyser and then be pumped to the molybdenum concentrate thickener.

A dewatering thickener would be installed between the third and fourth Mo cleaners to control the slurry density within the circuit. Dilute concentrate from the third cleaners would feed the thickener, and a denser thickener underflow would be returned to the fourth stage cleaners. Thickener overflow solution would combine with overflow from the Mo final concentrate thickener overflow and would be returned to the circuit as spray water. These water streams cannot be returned to the process water circuit due to the presence of residual amounts of NaHS.

Reagents (fuel oil collector, pine oil, NaHS, depressant, sodium silicate dispersant and carbon dioxide) would be added to the first four cleaner stages using dedicated pumps.

Spray water will be installed on all concentrate launders to break down froth, and aid slurry flow.

Medium-pressure air will be added to each column from the copper column compressors.

Spillage and washdown in the area will be handled by floor sump pumps, which will return spillage to the respective areas.

Off gas from the sealed flotation cells and respective slurry hoppers will report to the Mo caustic gas scrubbing system to neutralise hydrogen sulphide gas.

17.2.8 On-stream analysis

An on-stream analysis (OSA) system would be installed to provide up-to-date assay information for the critical flow streams in the plant.

Samples would be collected by pressure pipe samplers installed on pump discharge lines and gravity samplers, where required. Excess samples from the OSA would be grouped together in three or four streams and returned to appropriate locations in the flotation circuit.

Additionally, metallurgical samples would be collected for accounting purposes. These samples would be obtained via cross-cut samplers on feed, final concentrate and final tailings streams to provide metallurgical balances across the circuit.

17.2.9 Copper concentrate handling

Final copper concentrate would comprise the tailings streams from the molybdenum circuit rougher and first cleaner scavenger flotation cells. This concentrate would be dewatered in a concentrate thickener and three pressure filters prior to dispatch for export.

Over the life of mine, copper concentrate production is expected to be about 80 wet tph at an average concentrate grade of 25.7% Cu, and 10% moisture, for a throughput of 40 Mtpa, an average head grade of 0.42% Cu and 86.7% Cu recovery. The circuit would be designed for the maximum production rates, which are expected to be 143 tph of wet concentrate in a year (with a Cu head grade of 0.80% Cu).

Copper concentrate thickening

A single high-rate thickener is proposed to partially de-water the copper concentrate prior to filtration, and to collect filtrate, spillage and wash down in the area. Feed to the thickener would be the Mo flotation rougher-scavenger tails, and the cleaner scavenger tails pumped from the Mo float circuit.

If the Mo flotation circuit is not operated, the final copper concentrate would be produced by the bulk concentrate thickener installed ahead of the molybdenum circuit.

Settling rates for fine (80% passing 20 to 30 µm) concentrate have been measured by Outotec at 0.2 $t/m^2/h$ for a high rate thickener using flocculant; a thickening of 30 m diameter is anticipated to be required for the highest production year.

Thickener overflow may contain fine concentrate particles and would be collected in a small overflow tank and returned to the copper cleaner flotation circuit for use as spray water in the concentrate launders as well as reuse in the concentrate area. Thickener underflow at about 65% solids would be pumped by one of two installed horizontal centrifugal pumps to the copper filter feed surge tanks.

Dilute flocculant would be added to the thickener feed stream to assist in settling the concentrates. The thickener would be equipped with a froth weir to retain floating concentrate particles within the thickener. Process water could be applied as sprays to this froth to assist in its break-up.

Thickener underflow density would be controlled by varying the thickener underflow pump speed. Thickener bed level would be controlled to maintain a clear thickener overflow by varying flocculant dose.

Filter feed surge tanks

Feed to the copper filtration circuit would be from the copper concentrate thickener (if the Mo recovery circuit is operating) or from the bulk concentrate thickener (if the Mo circuit is in bypass mode).

The copper filtration circuit would be located at the concentrate loadout facility, adjacent to the rail spur.

Thickened concentrate would be pumped by one of two installed pumps to the concentrate loadout facility and would be stored in a two filter feed surge tanks which together would provide a live residence time of about 24 hours ahead of the concentrate filters, so as to provide a constant feed rate and density to the filters and to allow for maintenance requirements without affecting the flotation circuits.

One of three installed horizontal-centrifugal pumps would pump thickened slurry from the surge tanks to the two pressure filters. The pumps would have a pressure relief line back to the tank to cater for the batch filtration operation without switching the pumps off and on. One pump would be dedicated to each filter, with the third being a common standby.

Concentrate filtration

Three automatic Larox type pressure filters would be provided to handle the maximum copper concentrate production rate; at a utilisation of 76% this would be 193 tph (dry) of concentrate. The filters would operate in batch mode; the filtration cycle would comprise six stages lasting a total of approximately nine minutes.

The filtrate and cloth wash water discharging from the filters would flow to the filtrate tank and be pumped to the concentrate thickener feed box in the main plant area. The filters would be fully automated via a PLC system which would schedule the operation of the two filters to ensure the operating cycles of the filters are staggered.

The hydraulic system for the pressure filters would comprise a hydraulic fluid tank, a low-pressure pump system, and a high-pressure system. The filters and associated ancillaries would be housed in a dedicated building and would be supplied as a complete vendor package.

Concentrate storage

The filter cake discharge conveyors would discharge filter cake to a concentrate conveyor feeding the concentrate stockpile via a shuttling conveyor. The concentrate conveyor would be equipped with a weigh scale to totalise the concentrate production for metallurgical accounting purposes.

Filtered concentrate would be stored in a concentrate shed located adjacent to the rail loadout facility, which would provide a total of 45,000 tonnes of concentrate to be stored on site to act as a surge for transport purposes. This storage capacity would be equivalent to two weeks concentrate production during the peak years, and nearly four weeks production assuming the average LOM production rates.

17.2.10 Molybdenum concentrate handling

Molybdenum concentrates dewatering and bagging facilities would be located in the main concentrator building. Molybdenum concentrates would be dewatered in much the same way as for the copper concentrates described above.

The anticipated maximum annual recovery of Mo concentrate is only 8,270 tpa, or about 1 tonne per hour. Equipment sizes are significantly lower than the respective units for copper concentrate handling. Settlement rates for molybdenum concentrates will be lower than those for copper, but the concentrate thickener diameter required would be 5 m.

A single Larox PF 7.9/1.6 filter would be installed for molybdenum concentrate filtration.

Molybdenum concentrates tend to be very fine, and final moisture contents of 10% are not achievable by filtration alone. Filtered concentrates would therefore be dried using a diesel fired rotary kiln, or a hollow-flight screw conveyor followed by cooling and bagging prior to transport. The final moisture content is expected to be about 8%.

Assuming a bagging time of 12 hours per day, approximately one bulk bag of concentrate would be required to be filled per hour of bagging time. Bagged concentrate would be transported by forklift to a storage facility.

The dryer would consume about 58 L/h of diesel for each hour of operation.

Off gas from the dryer retort would report to the molybdenum filtration venturi scrubber. Off gas vented from the venturi scrubber and bagging station would report to the molybdenum filtration spray tower scrubber. Scrubber and cooling water discharge will report to the molybdenum clarifier via the filter filtrate tank.

17.2.11 Concentrates loadout

Copper concentrate would be reclaimed from the stockpile by front end loader, which would load concentrate into 30 t rail cars for delivery to the port, and export to overseas smelters. A maximum of about 3,400 t of moist (10% moisture) concentrates would be produced daily.

Molybdenum concentrates would be produced at a maximum rate of about 25 tonnes per day. They would be transported off-site in two tonne, sealed and waterproof bulka-bags, possibly loaded into containers, with about four 30 t container loads being shipped every week.

All concentrates would be transported by rail to the Chilean coast.

17.2.12 Reagents handling and storage

Reagent storage and mixing would be in a dedicated area of the plant, separate from the milling and flotation areas. Care would be taken in ensuring reagents that react with each other are kept separated, with no possibility of mixing.

Two reagent storage sheds would be provided for bags and drums of reagents. One shed would be designed for hazardous reagents, with bays to keep reagents separate, and smoke detection and sprinkler systems for inflammable material.

Both sheds would be provided with ventilation systems, safety showers, and washdown facilities. Sump pumps would be used to pump spillage and washdown to the flotation tailings area for disposal.

Safety showers would be also installed in each reagent make-up area, adjacent to each bag breaker (top platform) and at the mixing tank and transfer pump on the ground level. Ventilation would be installed over bag breakers for dust and fume collection.

The flotation reagent areas would each have a floor sump to collect area spills. A vertical spindle sump pump in each area would collect spillage and transfer it to a common reagent spillage tank from where it would be pumped by one of two horizontal centrifugal pumps to the final tails surge tanks. This would prevent potential plant upsets by the pumping of uncontrolled reagent spillage (either concentrated or diluted with hosing water) into the process.

NaHS spillage would be handled separately, because of the dangers of mixing this reagent with the other flotation reagents. Spillage of NaHS would be individually pumped to rougher flotation tailings for maximum dilution and mixing prior to the tailings thickeners.

An overhead crane would be provided in each area to handle reagent bags and drums, and to assist in maintenance.

Reagents would be distributed to their respective dosage points by dedicated positive displacement pumps, through flowmeters. Reagent dosages would be automatically controlled through the ECS.

17.3 Process plant water circuits

Process water would be generated from thickener overflow solutions (from both flotation tailings and rougher concentrate thickeners). The tailings thickeners would overflow into two overflow tanks, and from there pumped to a single process water tank, which would be the main distribution for the process. Overflow from the rougher concentrate thickeners would be collected in a separate overflow tank and also be pumped to the process water tank.

Overflow from the density control thickener in the flotation circuit and from the bulk and the copper concentrate thickeners would be gravitate to an overflow tank and the water recycled within the flotation circuit for launder sprays, etc.

Make-up water would be provided by fresh water from the fresh (or brackish) water storage tanks; tank levels would be maintained via water pumped from the fresh water borefields.

The process water tank would provide a capacity of about 15,000 m^3 or approximately one hour's requirements.

Process water would also be used for:

- dust suppression sprays on all conveyor feed and transfer points
- make-up water in the milling circuit for cyclone feed dilution
- dilution water for density control in the rougher and the cleaner flotation feed streams
- spray water on all screens
- hosing and wash-down in the milling and rougher flotation areas

Process water make-up would also include pit dewatering water, and any decant water from the TSF. It is currently unclear if any decant water would be available, given the climatic conditions, or how it would be accumulated and pumped from the TSF. The current water balance assumes no TSF decant return.

17.3.1 Concentrate water

It will be preferable to prevent suspended solids in the overflow water from the various concentrate thickeners being recycled to the milling circuit. The suspended solids would comprise fine concentrate particles, and thus overflow water would gravitate directly to flotation spray water tanks. Make-up water would be provided from the raw water system.

17.3.2 Raw (brackish) water

New make-up water requirements would be approximately 2,000 m^3/h. This water would be fresh or brackish water provided from several remote borefields.

The fresh water would be pumped to three raw water tanks dedicated to each borefield. Each tank would have a minimum live volume of 7,500 m^3, providing a total capacity of 12 hours.

Brackish water would be pumped by two of four available pumps from the raw water tanks to supply the six different raw water reticulation circuits.

17.3.3 Water reticulation

Six water reticulation circuits would be associated with fresh/brackish water:

- gland seal water for the horizontal centrifugal pumps – however mechanical seals on all pumps should be evaluated
- service water for the primary crushers
- dust suppression around the crusher, conveyors and stockpile areas
- cooling water for the mill drives and lubrication systems, and for the crusher lube systems
- make-up water for reagent systems, and for sprays and hosing in the cleaner flotation and reagent make-up areas
- potable water
- mine services water
- firefighting water

Each of these systems would comprise one operating and one standby pump drawing water off a raw water tanks.

17.3.4 Gland seal water

Gland water would be supplied to horizontal centrifugal slurry and process water pumps as shaft sealing water, and to the grinding mill feed chutes as lubrication and sealing water.

Gland supply would pass through a ‘y’ strainer at each pump to ensure a clean supply of water to the shaft seals. The pumps would be protected by flow and pressure switches on the gland water supply. A reduced flow of gland seal water would trip the pump. Gland water would be supplied at a pressure of up to 750 kPa.

17.3.5 Cooling water

Cooling water would be required for the mill drives and lubrication systems. A closed loop system is proposed for these duties, to allow high quality water is used and ‘furring’ of heat exchangers is minimised.

Warm water recovered from the cooling systems would be recycled through evaporative cooling towers before being pumped back to the mill and crusher cooling circuits. Evaporative losses would be made up with potable water; chemicals would be added to maintain water quality.

17.3.6 Other brackish water requirements

Brackish water would be used for all reagent make-up and for hosing and flushing water in the cleaner flotation and reagent make-up areas of the plant. A single reticulation circuit would be installed with multiple off takes to service these requirements. These would be of a relatively small flow rate, at a pressure of about 500 kPa.

17.3.7 Potable water

The general quality of borefield water in some of the catchment areas identified in Chapter 7 is expected to be suitable for use as potable water, though it would require treatment. Potable water would be reticulated throughout the plant and mine for use in safety showers and for general consumption. Make-up water to the mill and crusher cooling circuits would also be potable water. The consumption of potable water is described in the mine and other water balance estimate outlined in Item 13.

Raw water would be pumped from a raw water tanks to a water treatment system where it would be treated and filtered to reduce turbidity and bacteria content. The potable water production system would be a dual-media pressurised sand filter water treatment system, comprising filter feed and backwash pumps, an automatic filter, and an ultraviolet (UV) sterilisation system.

Approximately 320 m^3 per day of potable water would be required.

17.3.8 Firefighting water

Fire protection water would be distributed around the plant area by a dedicated reticulation system, to several fire hydrants and hose reels

Firefighting water would be supplied from a dedicated tank, continuously fed from the raw water system. It would continuously overflow back to the raw water tanks thereby always ensuring a guaranteed minimum quantity of water for firefighting purposes. This water would be pumped into the fire water distribution system by the fire water pump. Minor leaks would be addressed by a fire water jockey pump and accumulator.

In the event of a power failure during operation of the firefighting water system, a diesel engine powered fire water pump would start automatically and provide water to the fire hydrants.

17.3.9 Process plant mass and water balance

A summarised mass balance for the 40 Mtpa concentrator is provided in Figure 17-5. From this a water balance around the circuit can be derived and this is presented in Figure 17-6.

This mass balance is based on an annual throughput of 40 Mtpa, 10% mass pull to the rougher concentrates (as defined by testwork), a concentrate make of 72 tpa (LOM average), a thickened tailings density of 70% solids, and a re-diluted (with brine) tailings slurry of 65% solids.

The overall water consumption for the process is dependant primarily on the slurry density discharged from the plant to the tailings storage facility. All other water streams within the process circuits would be recycled and would not contribute to water consumption. The overall balance is based on the block flowsheet in Figure 17-4.

Figure 17-4 Overall water balance for the process facilities

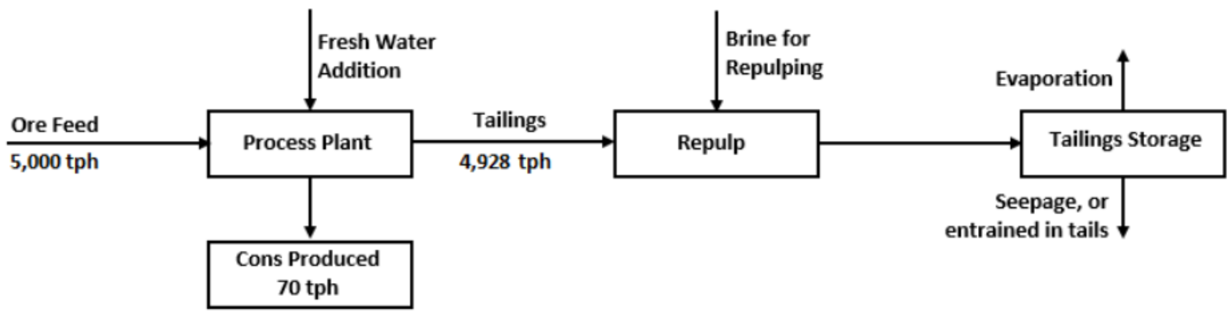


Table 17-2 provides an estimate of fresh water and brine requirements for the process plant at various densities of slurries pumped to repulp and pumped to tailings for final disposal. It should be noted that the water content for repulped slurries is reported in tph and not m^3/h because the density of brine is 1.15.

The density of thickened tailings from the process to re-pulping should be as high as possible to minimise fresh water losses from the circuit (and hence requirements for fresh water addition). However, this density would be limited by the settling characteristics of the tailings, and by the rheology of the thickened tailings (i.e. what density can be pumped).

Table 17-2 Fresh water and brine requirements

Tailings % Solids to Repulp	**68**			**70**			**72**		
Water Content, m^3/h	2,319			2,112			1,916		
Concentrate Water Content, m^3/h (10% Moisture)	8			8			8		
Fresh Water Addition, m^3/h (no water in ore)	**2,327**			**2,120**			**1,924**		
Fresh Water Addition, m^3/h (3% water in ore)	**2,172**			**1,965**			**1,770**		
Repulp % Solids	**62**	**65**	**66**	**62**	**65**	**66**	**62**	**65**	**66**
Water Content, tph	3,020	2,654	2,539	3,020	2,654	2,539	3,020	2,654	2,539
Brine Addition Required, tph (no decant return)	**701**	**334**	**220**	**908**	**542**	**427**	**1,104**	**737**	**622**

Likewise, the repulp density would need to be designed to ensure the repulped tailings can be pumped to the TSF, and can be properly deposited around the TSF, and are able to form a beach. Higher quantities of brine would be required for re-pulping when the thickened tailings are at a higher density.

Table 17-2 shows that with a thickened tailings density of 70% solids, the processing facilities would require a total of 1,965 m^3/h of fresh or brackish water (assuming 3% moisture present in the ore feed).

If the thickened tailings were to be repulped to 62% solids (w/w) with brine, approximately 909 tph (790 m^3/h) of brine would be required, assuming no decant from the TSF. Increasing this repulped density to 65% solids, reduces the brine requirement to 541 tph (471 m^3/h) This higher repulped slurry density is used in the water balance in Figure 17-5. A study on beach angles achievable on the TSF versus the density of the slurry deposited is being undertaken to confirm that this higher slurry density will not create too high a beach angle.

It is unclear whether it will be possible to recover water from the TSF, and the above requirements for brine assume no decant water is available. Any decant return would reduce the quantity of new brine required.

The projected water requirements are based on a copper project at 40 Mtpa, and do not take into account any water required to run a gold heap leach or CIL circuit. If the operation of a gold circuit overlaps the copper operation, then additional water will be needed.

The water balance for a 60 Mtpa operation would be similar, with all flows and water demands increased by a factor of 1.5.

Figure 17-5 Stage 1 40 Mtpa concentrator mass balance

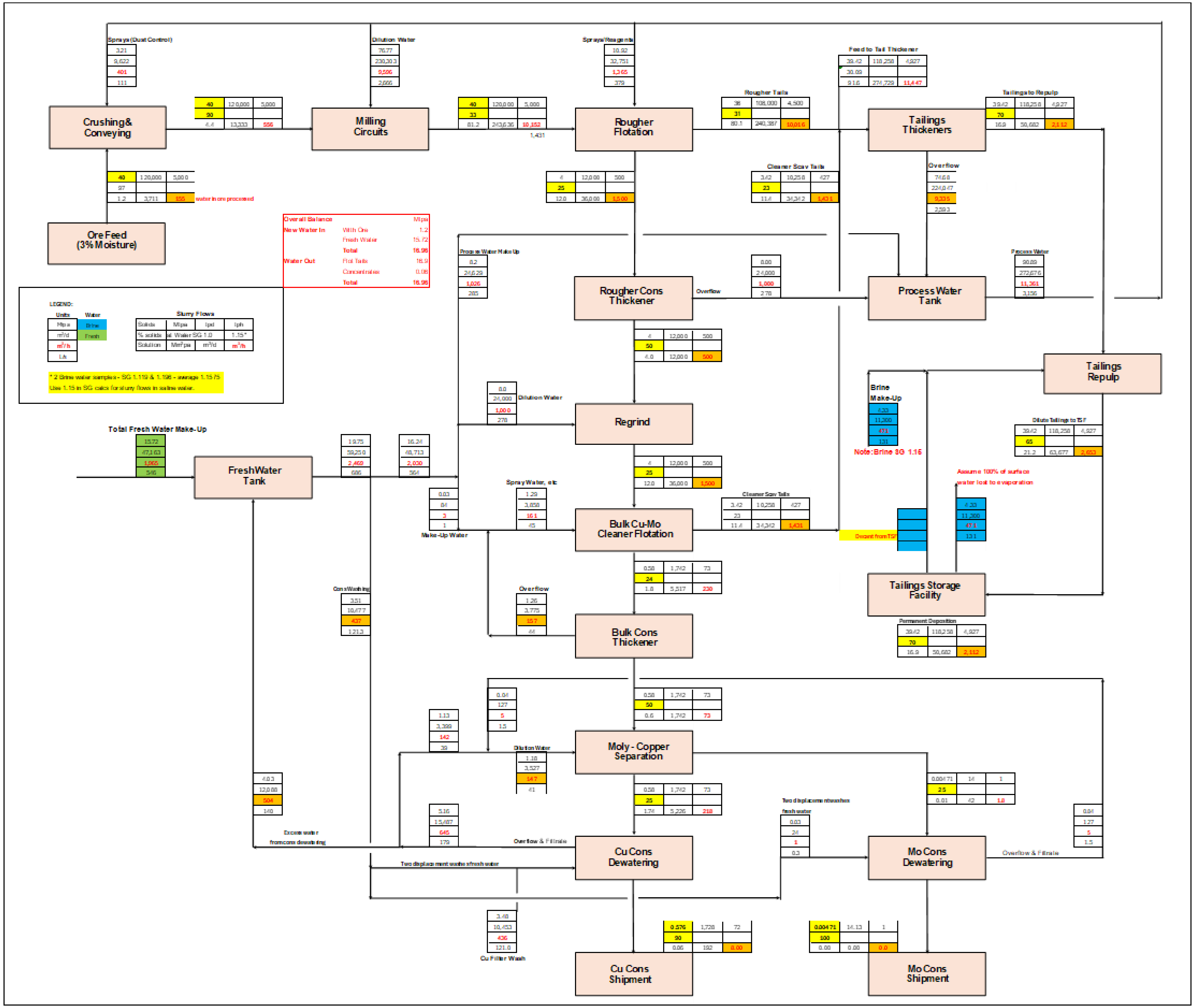
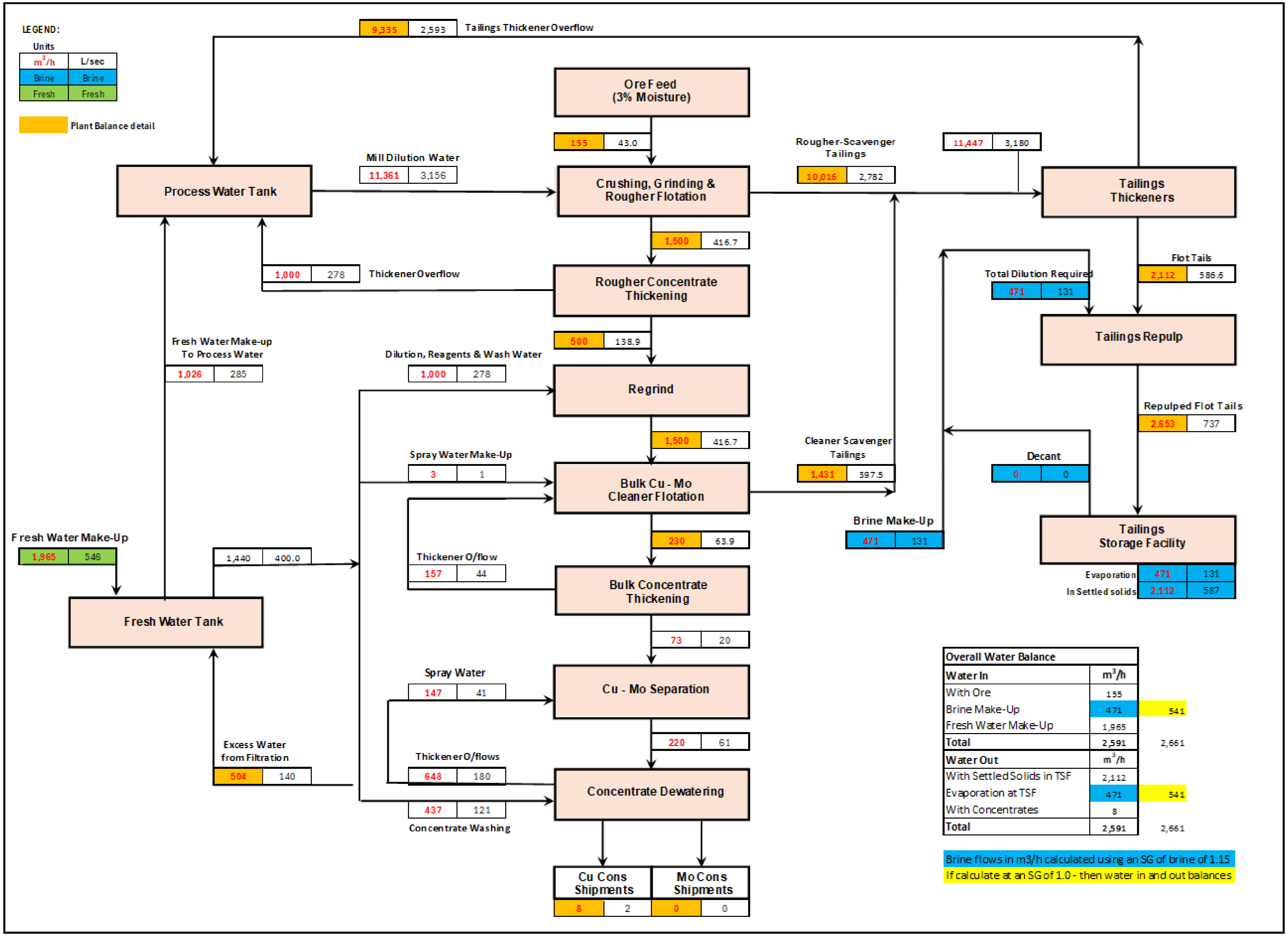


Figure 17-6 Stage 1 40 Mtpa concentrator water balance



17.4 Plant air systems

Five distinct compressed air systems would be required during the operations phase, i.e.:

- plant air
- instrument air
- flotation low pressure air
- flotation medium pressure air
- filtration air

17.4.1 Plant and instrument air

High pressure compressed air (750 kPa) would be generated for general use throughout the plant and for instrument air supply. Instrument air, specifically, would be needed for use in the actuation of valves and for use by several instrument systems.

Compressed air would be supplied through a single reticulation system by three air compressors: two operating and one standby. The compressed air would be cleaned and dried in a series of filters and dryers to provide air of instrument air quality. Air receivers would be strategically located throughout the plant to ensure a continuous air pressure in the circuit, particularly at high off-take areas such as the mills.

A similar system would be provided at the primary crusher area; a single system would supply the two primary crushers.

17.4.2 Flotation air - low pressure

The flotation cells would require forced air addition at a gauge pressure of approximately 65 kPa for the rougher and cleaner flotation cells.

Three blowers (two running and one on standby), each rated at 31,500 Am^3/h would be installed for this duty.

17.4.3 Flotation air - medium pressure

Higher pressure air would be required for the flotation columns and the coarse copper cleaner cell. Two compressors in a duty, standby configuration would be installed for this duty. Compressed air would be supplied at a pressure of 280 kPa and a flow of 7,570 Am^3/h.

17.4.4 Air for concentrate filters

Because of its remote location at the concentrates load-out area, a separate system would be required for copper filtration pressing air.

Two compressors and receivers will be supplied for this duty; a portion of the dried air will also be used to operate the filter instruments.

Molybdenum concentrate filtration air will be supplied by the main plant air compressors, as this circuit is located in the main concentrator area.

17.5 Process control

A significant level of process control and instrumentation would be incorporated into the process design, to minimise the number of process operators required. Process control and stop/start of equipment would be managed from a central control room. The level of process control would be increased gradually after plant start-up, as more information is gathered on different process variables and their mutual interdependency. A process control and management system ('expert control system', ECS) would control the concentrator process.

The main objectives of the process control would be:

1. To maximise the throughput of the plant at desired grinding fineness to achieve sufficient liberation of valuable minerals.
2. To produce high-grade copper and molybdenum concentrates of commercial grade at maximum recovery.
3. To minimise operating costs by optimization of reagent additions, and efficient use of energy in the milling circuit.

The principal measurements and control loops for the concentrator would be:

1. Control of ore and water feed to grinding. Water addition to the mill feed would be in proportion with the ore feed rate.
2. Control of hydrocyclone operation by either constant density in the feed, or constant feed pressure at the hydrocyclone inlet, by varying cyclone feed pump speed, and/or level control and water addition to the cyclone feed sump.
3. Monitoring and control of circulating load in the ball milling circuit by slurry flow and density measurement of hydrocyclone feed.
4. The fineness of grind in the hydrocyclone overflow would be monitored and controlled by means of an on-stream particle size measuring instrument. Control would be achieved by varying the mill feed rates or speeds, and hence the power draw.
5. An on-stream x-ray analyser (OSA) would be used to monitor and control the flotation process by analyses for copper, iron and density of selected process slurries. Ultimately, the OSA would be linked to reagent additions for control by the ECS.
6. Control of slurry level and flotation air flow to all flotation cells.
7. Cameras on each flotation cell would be used to monitor the froth speed and bubble size and stability from the control room.
8. Reagent make-up systems would be fully automated using tank levels and timers; the only operator involvement would be in feeding the reagents to the mixing tanks or feed hoppers.
9. Control of flotation reagents to each individual feeding point would be based on a grams per tonne of ore feed basis. Dosing pumps would be controlled by ratio from the mill feed weightometer.
10. Control of flocculant feed rate would be based on thickener interface level or grams per tonne of material feeding the thickeners. Dosing pump speeds would be controlled by ratio from the bed level measurement devices or concentrate slurry mass flow meters.
11. Monitoring and control of tailings disposal by measurement of slurry velocity and pressure in the tailings pipeline.

In addition to the above, typical instrumentation would be provided, as follows:

- hopper and tank slurry levels would be controlled by variable speed discharge pumps
- spillage sump levels will start and stop spillage pumps
- variable speed underflow pumps would control thickener underflow densities

Vendor installed instrumentation packages would be provided for operation, control and protection of the primary and pebble crushers, mills, thickeners, concentrate filters, and selected reagent make-up systems (e.g. flocculant).

17.6 Plant layout

Figure 17-7 shows a layout plan for the 40 Mtpa concentrator and related facilities. Footprint shapes in this layout provide for a future expansion to 60 Mtpa involving the addition of a second stockpile and milling circuit.

As noted previously, the detailed design and construction of the molybdenum recovery may be delayed for several months until the main copper flotation circuit is operating. The layout drawing thus shows a single box, illustrating where the molybdenum plant would be situated.

Some additional design features common to all areas are noted below.

17.6.1 Dust suppression and shielding

Due to the persistent winds at the Project site, all conveyors would be covered and equipped with water sprays to suppress dust. Washdown facilities would also be provided at each conveyor feed or transfer point.

The primary crusher truck dumping areas would be shielded, and the crushed ore stockpile would be covered.

All processing facilities would be installed in a building.

A wind barrier or cover would be installed on all thickeners to reduce the possible generation of waves caused by strong winds. Wave action on these large diameter thickeners could adversely affect the settlement of solids in the tailings slurry. The wind could also lead to accelerated water loss through evaporation from the large surface area of the thickeners.

17.6.2 Spillage handling

Spillage handling and washdown facilities would be provided at numerous locations around the plant. High tonnage areas would be serviced by water cannons in additions to hose points. Spillage and washdown would be returned to the appropriate location in the circuit using sump pumps.

High pressure water for washdown would be process water which would be provided from the main water reticulation circuit, with booster pumps installed to maintain the pressure at remote locations.

17.6.3 Safety showers

Safety showers would be located throughout the plant, and particularly at points where operating personnel could be exposed to dust or splash.

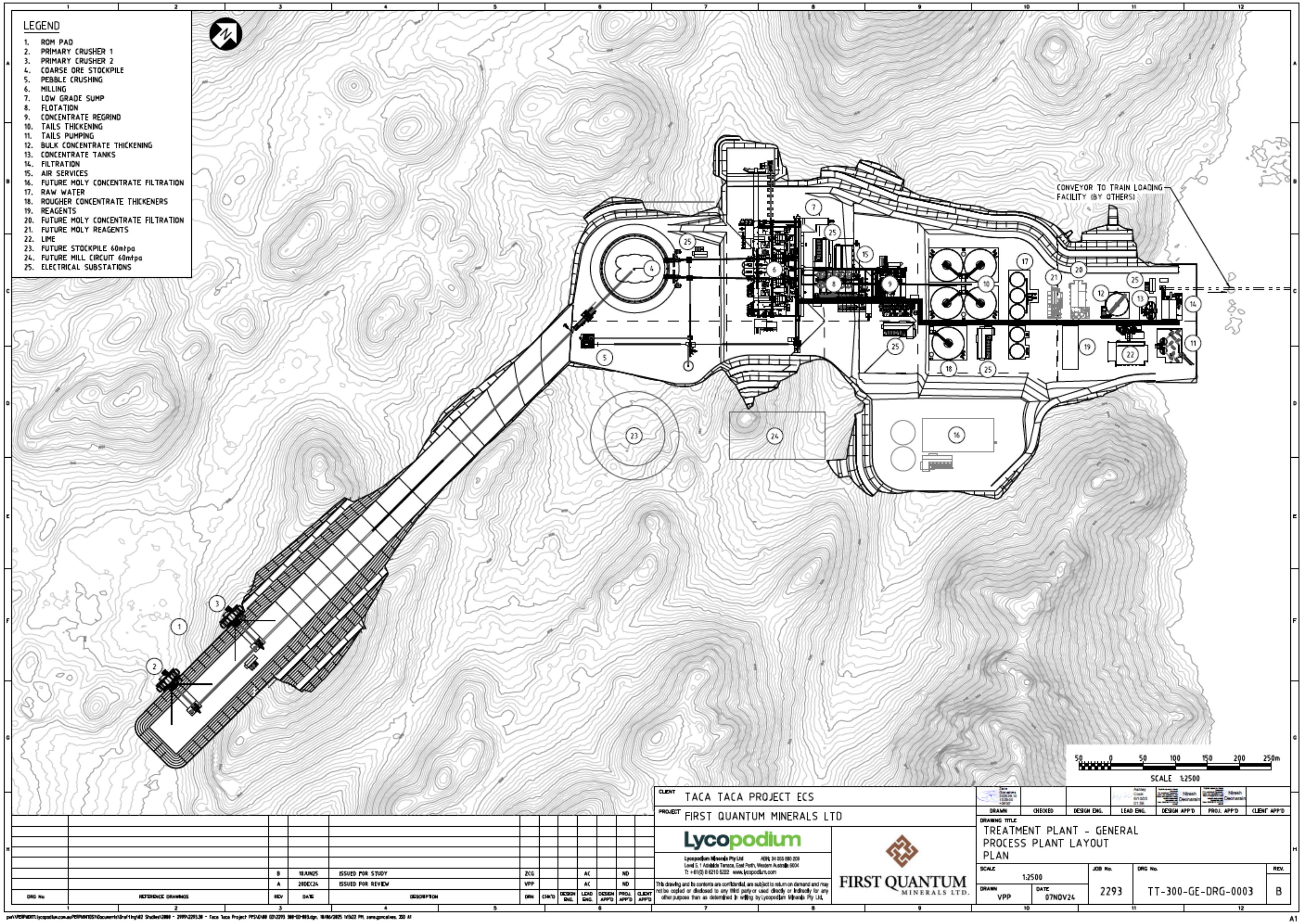
Each regent make-up facility would have a safety shower on the operating level and on the ground floor by the dosing pump.

Safety showers would be provided with potable water supply and would be alarmed when activated.

17.7 Processing consumables

Concentrator reagent requirements have been defined following various testwork campaigns undertaken by Plenge Laboratories in Lima, ALS laboratories in Kamloops, Canada, or estimated from design experience at the Company's other sites.

Figure 17-7 Layout of Stage 1 40 Mtpa flotation/concentrator plant (source: Lycopodium June 2025)



Reagents will be delivered to the site by rail. Dry reagents will be transported in one-tonne bulk bags, excluding NaHS (0.7t), CO_2 (22 m^3 cryogenic liquid containers), and remaining liquid reagents (1 m^3 IBCs). Reagents will be stored in enclosed buildings prior to use in the plant. Buildings will be sized to hold 14 days of stock for each reagent. Plant reagent storage tanks and vessels will be sized to hold sufficient reagent for 12 hours of plant operation where practical. Reagent mixing and loading during normal operations will be conducted during day shift.

Reagent quantities for molybdenum separation from bulk concentrates are calculated on the quantity of bulk concentrates produced, which is related to the copper feed grade. Thus, two sets of numbers for daily and annual consumptions are quoted for these reagents: one based on the high grade ores of 0.8% Cu seen in the early years of the operation and the second based on the LOM average feed grade of 0.42% Cu.

Daily and annual reagent consumptions are presented in Table 17-3.

17.7.1 Grinding media

It is assumed that grinding media (steel balls) would be delivered in two tonne bulk bags. Several ball sizes would be required:

- 140 and 125 mm balls for the SAG mills
- 65 or 80 mm balls for the primary ball mills
- 50 mm balls for the lime mill
- 4 mm ceramic beads for the concentrate regrind mill

The balls would be stored in concrete bunkers in the reagent delivery area, with each ball size having its own separate bunker. The primary mills would be charged via a ball feeder and conveying system which would run over the top of the mill feed chutes. A common conveying system would be used for feeding balls to the two SAG and four ball mills.

Cameras over the ball conveyor would be used to monitor and control the number of balls added to each mill daily. An overhead crane and ball kibbles would be used to load balls to the lime mill via a ball addition chute.

The lime slaking mill would be charged with balls daily using a kibble and davit crane.

The proposed regrind mill design includes a media charging hopper on top of the mill. Bags of 4 mm ceramic beads would be lifted into this hopper by overhead crane daily.

17.7.2 Sodium ethyl xanthate (SEX), or potassium ethyl xanthate (PAX)

SEX or PAX would be used as the primary collector for copper minerals at Taca Taca. The proposed dosage pipework would enable the collector to be added to the same points as the frother within the flotation circuit. Its total consumption rate would be about 16 g/t of ore, or 1,900 kg per day, as defined by the recent testwork programmes. The make-up system would be designed for a higher consumption rate to cater for unforeseen events during operation.

Two storage and dosing systems would be installed to allow for plant trials with alternative collectors.

Collector would be received as a dry powder or pellets in 1,000 kg bulk bags and stored in the reagent shed until needed. A reagent crane would be used to lift the bulk bags into a sealed and ventilated bag enclosure where the bags would be emptied into the collector mixing tank. Empty bags would be stored for disposal in an environmentally responsible manner or returned to the vendor.

The collector would be made up to a solution strength of 15% reagent using brackish or fresh water. The mixing would be performed batch-wise, with mixed solutions being transferred to a storage tank providing about 12 hrs surge capacity.

From the storage tank the reagent would be pumped to one of two day tanks located in each flotation circuit (one each for roughing and cleaning). A series of variable speed peristaltic dosing pumps would be provided for collector dosing to each point in the process, with two additional pumps being provided as standby units.

The make-up system would be fully automated using tank levels and timers; the only operator involvement would be in feeding the dry reagent to the mixing tank.

Flammable gases such as carbon disulphide may be present during the addition and mixing of xanthates. The SEX mixing systems would therefore be totally enclosed and provided with water sprays, and the mixing and storage tanks would be sealed. An extraction fan would vent the tanks to atmosphere via a discharge duct through the roof of the reagent building.

17.7.3 Methyl isobutyl carbinol (MIBC) frother

Methyl isobutyl carbinol (MIBC) would be used as the initial frother reagent for the bulk flotation circuit. The frother make-up system would be designed for an overall consumption rate of 21 g/t of ore, or about 2,500 kg per day. The frother would be added at several points in the rougher and cleaner flotation circuits, with each point being fed through a dedicated dosage pump.

Frother would be delivered as a liquid in 1,000 litre drums or isotainers; these would be loaded as required by fork truck onto an elevated platform and drained by gravity into a small storage tank. From the storage tank a pump would transfer frother to a day tank located in the flotation circuit.

Dedicated variable speed peristaltic pumps would transfer frother to each dosing point through a flowmeter for process control. Two spare pumps would be supplied for maintenance purposes.

Two storage and dosing systems would be installed to allow for testing of alternative frothers during plant operations.

17.7.4 Lime

Lime would be delivered to the site as quicklime in one tonne bulk bags. From preliminary testwork, consumption is expected to be about 680 g/t or 82 t/day at a concentration of 85% w/w calcium oxide (CaO).

Reagent hoists would be used to lift bulk bags to two bag breakers and hopper arrangements, where the dry lime will be pneumatically transferred to a single lime silo with a minimum capacity of 40 dry tonnes.

Quicklime from the lime silo will be discharged via a screw feeder and vibrating feeder directly into a lime slaking mill. Water will be added to the mill to generate a milk of lime product. The slaking mill will operate in closed circuit with hydrocyclones, and the cyclone overflow will pass over a de-grit screen before reporting to a 500 m^3 agitated storage tank at 15% w/w solids pulp density.

The milk of lime slurry will be pumped via a ring main from the storage tank to all dosage points and return to the tank. The duty/standby centrifugal pump arrangement will operate continuously. The addition of lime to each dosing point would be controlled to a set pH using pulsed pinch flow control valves.

Lime addition points are expected to be:

- SAG mill feed hoppers
- Regrind mill feed hoppers
- cleaner flotation circuit (several dose points)

A dust scrubber will be installed to capture lime dust around the lime mixing circuit. Lime plant spillage will be pumped by a spillage pump to the de-grit screen ahead of the storage tank.

17.7.5 Flocculant

Flocculant would be added at several points in the circuit, i.e.:

- rougher flotation tails thickeners
- Cu concentrate thickener
- Mo concentrate thickener

The rougher concentrate thickeners, the bulk Mo-Cu concentrate thickeners, and thickeners installed in the cleaner circuits for slurry density control would be conventional thickeners without flocculant addition, to avoid any deleterious effects on downstream flotation processes. However, it is proposed that a dedicated flocculant make up facility be designed in case it is found that these concentrates do not settle to acceptable densities. Alternatively additional dosing pumps could be provided from the facilities described below.

Total consumption of flocculants for the flotation circuits is expected to be approximately 25 g/t mill feed, or 3,000 kg per day. Due to the location of different off-takes around the plant, the possibility of a requirement for different flocculants for the different duties, and because of different consumption rates, separate flocculant packages would be provided for each duty.

Flocculant for the tails thickeners would be delivered to site in 1,000 kg bulk bags. If the flocculant used in the concentrate thickeners is different, it may be supplied in 25 kg bags.

Each make-up system would be similar. A reagent area overhead crane would hoist the bags onto a bag breaker and flocculant would be discharged into a flocculant feed hopper. Dry flocculant would be pneumatically transported to a series of wetting heads where water would be contacted with the flocculant powder prior to the solution mixing tank. After agitation for 30 minutes, the solution would be transferred to a storage tank for ageing. This tank would provide ageing and surge capacity.

All flocculant mixing systems would be fully automated, controlled by tank levels and timers; one batch would be mixed as soon as the mix tank is empty, and would be transferred to the flocculant storage tank as soon as that tank level has dropped to a pre-determined level.

Flocculant would be dosed to the circuit using variable-speed positive displacement pumps. Each thickener would be provided with a dedicated flocculant supply pump feeding the feed box and a second pump dosing into the feed well to ensure effective flocculation. Prior to addition to thickeners, the flocculant would be further diluted by addition of process water through in-line mixers to the flocculant delivery line.

17.7.6 Fuel oil molybdenum collector

Emulsified diesel would be used as the molybdenum collector at Taca Taca. Diesel, surfactant and water would be individually added to the oil emulsification tank at a ratio of three parts surfactant to five parts diesel to six parts water. The diesel and surfactant consumption rates are expected to be approximately 28 g/t and 17 g/t of bulk concentrate, respectively (equivalent to respective daily consumptions of 87 kg and 53 kg when treating high grade ores, and 48 and 29 kg respectively when treating lower grade ores).

Emulsified diesel would be added to several locations in the rougher flotation circuit via a series of variable-speed positive displacement pumps. One spare dosing pump would be installed to allow for pump maintenance.

Diesel would be collected via IBC from the onsite bulk diesel storage and pumped from a holding tank to the oil emulsification tank. Surfactant will be delivered as a liquid in 1000 L IBCs and pumped from a holding tank to the oil emulsification tank.

The make-up system will be fully automated using tank levels and timers. Loading the reagents from IBCs into their respective holding tank will, however, require manual intervention.

Diesel would also be used to fuel the molybdenum concentrate dryer after filtration. Diesel would be supplied from the bulk diesel storage are in IBCs, pumped from site storage to the dryer diesel tank, and then transferred to the dryer header tank. Diesel would be pumped to the dryer, as required. The dryer diesel consumption is expected to be about 1,000 kg per day.

17.7.7 Polyglycol molybdenum frother

Similar to the Cobre Panamá design, Polyglycol will be used as the primary molybdenum frother. The consumption rate is expected to be approximately 37 g/t of bulk concentrate, equating to between 115 and 64 kg per day, depending on the ore feed grade. Frother would be added to several locations in the rougher, cleaner and column cleaner flotation circuits.

Frother would be delivered as a liquid in 1000 L IBCs. Neat frother would be pumped from the IBCs into the single storage tank on a day-shift basis. A series of variable-speed positive displacement pumps would distribute the frother to individual dosing points throughout the plant. One spare frother pump would be installed to allow for pump maintenance.

17.7.8 Sodium hydrosulphide (NaHS)

NaHS is required in molybdenum flotation to oxidise the residual xanthate added to the bulk flotation circuit and remove the hydrophobic nature of the sulphide minerals.

The overall NaHS consumption rate is expected to be 25 kg/t of bulk concentrate: between 46 and 25 t/day for high and low Cu feed grades respectively. NaHS would be added to several locations in the molybdenum rougher and cleaner flotation circuits. NaHS would be delivered to the plant site as flake in 700 kg bulk bags. Because of the expected consumption (60 bags per day), a dual make-up and storage circuit would be installed.

Reagent hoists would be used to lift the bulk bags into sealed ventilated bag enclosures and then emptied into agitated mixing tanks. Flake NaHS would be mixed to a concentration of 40% w/v using raw water in two agitated mixing tanks and transferred to two storage tanks.

Each NaHS storage tank would pump to a common dosing tank, whilst being further diluted to 21% w/v using inline mixers. The diluted NaHS solution would be dosed to the molybdenum flotation circuits using dedicated variable speed positive displacement pumps.

The NaHS mixing and storage facilities would be installed within a bund area separated from other reagents. The mixing and storage tanks would be sealed and vented through a scrubber, and the tank overflows would be sealed through a seal pot. Spillage from NaHS make-up would be pumped to tailings flotation prior to the tailings thickener for immediate dilution and disposal.

Gas monitors would detect the presence of abnormal levels of H_2S gas in the area and warn of potentially hazardous conditions.

The NaHS make-up system will be fully automated using tank levels and timers. Manual operator involvement will, however, be required for hoisting the reagent bags above the mixing tanks and discharging the contents into the tanks.

17.7.9 Carbon dioxide

CO_2 would be added to the molybdenum circuit to reduce the slurry pH prior to flotation. Liquid CO_2 would be delivered to site in 22 m^3 cryogenic, vacuum insulated containers. The liquid CO_2 would be drawn through a vaporiser to a pressure control manifold and piped to the molybdenum flotation plant. Nominal CO_2 usage would be approximately 126 kg per day.

17.7.10 Hexafluoro polymer molybdenum depressant

The molybdenum flotation depressant consumption rate is expected to be about 1,000 g/t of bulk concentrate, or between 3,200 and 1,800 kg per day. Depressant would be delivered as a liquid in 1000 L IBCs and added to several locations in the cleaner flotation circuits.

Neat depressant would be pumped from the IBCs into the single storage tank on a day-shift basis. A series of variable speed positive displacement pumps would distribute the depressant to individual dosing points throughout the plant.

17.7.11 Sodium silicate dispersant

The molybdenum flotation dispersant consumption rate is expected to be 130 g/t of ore, equating to between 400 and 225 kg per day. Dispersant will be added to several locations in the cleaner and column cleaner flotation circuits. Sodium silicate will be delivered as a liquid in 1000 L IBCs at a concentration of 32% w/v reagent.

Sodium silicate will be transferred from the IBCs into the dispersant holding tank on a day-shift basis. The holding tank will pump to an agitated mixing tank, whilst being diluted to a 5% w/v solution concentration using an inline mixer and raw water. The diluted dispersant solution will be dosed to various points in the Mo cleaner circuit using variable speed positive displacement pumps.

17.7.12 Sodium hydroxide (NaOH)

Caustic would be used in the molybdenum flotation circuit for protective alkalinity for NaHS, plant and would also be required in small quantities for the caustic scrubber on the NaHS mixing and storage tanks and the scrubbing circuit on the molybdenum flotation plant. The overall caustic consumption rate is expected to be about 330 g/t of bulk concentrate, or between 1,000 and 570 kg/day.

Caustic would be delivered to the plant site in either pearl or flake form, in one tonne bulk bags. A reagent hoist would be used to lift the bulk bags into sealed ventilated bag enclosures, where the bags would be emptied into an agitated mixing tank via screw feeder. The caustic would be mixed to a concentration of 20% w/v using raw water and will be transferred to a single storage tank.

The caustic solution would be dosed to the circuit using variable speed positive displacement pumps.

The make-up system will be fully automated using tank levels and timers. Manual operator involvement will, however, be required for hoisting the reagent bags above the mixing tank and discharging the contents into the tank.

17.7.13 Miscellaneous reagents

Various minor reagents would be required for water treatment purposes, for the supply of drinking water, and for water treatment in cooling water circuits. The consumption of these reagents will be comparatively small; they would be supplied in 200 litre, disposable or returnable plastic chemical drums.

17.7.14 Summary of processing consumables

Table 17-3 summarises the estimated annual quantities of concentrator consumables.

Table 17-3 Consumable requirements for the comminution and copper flotation circuits

Consumables		Design Basis	Consumption Rate, t		Delivered as
		g/t ore feed	tpd (24 hrs)	tpa (8,000hrs)	
Grinding Media					
SAG Mill Balls	140 mm	417	50.0	16,680	1 t bulk bags
Ball Mill Balls	65 mm	431	51.7	17,240	1 t bulk bags
Regrind Mill Balls	Ceramic Beads	0.38 kg/t rougher cons	3.5	1,150	1 t bulk bags
Lime Ball Mill Media	50 mm	0.8 kg/t lime	65.6 kg/d	22	1 t bulk bags
Reagents - Copper Flotation					
Collector	SEX or PAX	16	1.9	640	1 t bulk bags
Frother	MIBC	21	2.5	840	1,000 L IBC
Lime (pH Modifier)	Quicklime	683	82.0	27,320	1 t bulk bags
Flocculant	Flot Tails & Cons	25	3.0	1,000	1 t bulk bags

Reagent requirements for the molybdenum flotation circuit are presented in Table 17-4. The daily and annual consumption of these reagents is dependent on the quantity of bulk concentrate produced in the main rougher and cleaner flotation circuits, which is dependent on the copper feed grade. Thus, two sets of requirements are presented – one for the years seeing the peak copper grades of 0.80% Cu (Year 10 of the Project) and the second related to the life of mine average grades of 0.42% Cu.

Table 17-4 Reagent requirements for the molybdenum recovery circuit

Consumables - Molybdenum Flotation		Design Basis	Consumption Rate, t				Delivered as
			129 tph Bulk Cons		72 tph Bulk Cons		
			g/t Bulk Cons	tpa (8,000hrs)	kg/day (24 hrs)	tpa (8,000hrs)	
Collector	Diesel Oil	28	87	29	48	16	1,000 L IBC *
Cons Dryer Fuel	Diesel Oil		956	319	956	319	1,000 L IBC *
Frother	Polyglycol	37	115	38	64	21	1,000 L IBC
Surfactant	NP 10	17	53	18	29	10	1,000 L IBC
NaHS	NaHS	15 kg/t cons	46,440	15,480	25,920	8,640	0.7 t bulk bags
pH Modifier	Carbon Dioxide	1.1g/t ore	132	44	132	44	1 t bulk bags
Depressant	Hexafluoro Polymer	1,021	3,158	1,053	1,763	588	1,000L IBC
Dispersant	Sodium Silicate	130	402	134	225	75	1,000 L IBC
Scrubber Solution	Sodium Hydroxide	330	1,022	341	570	190	1 t bulk bags
*** Note - Diesel Oil supplied from onsite bulk storage in Fuel Farm**							

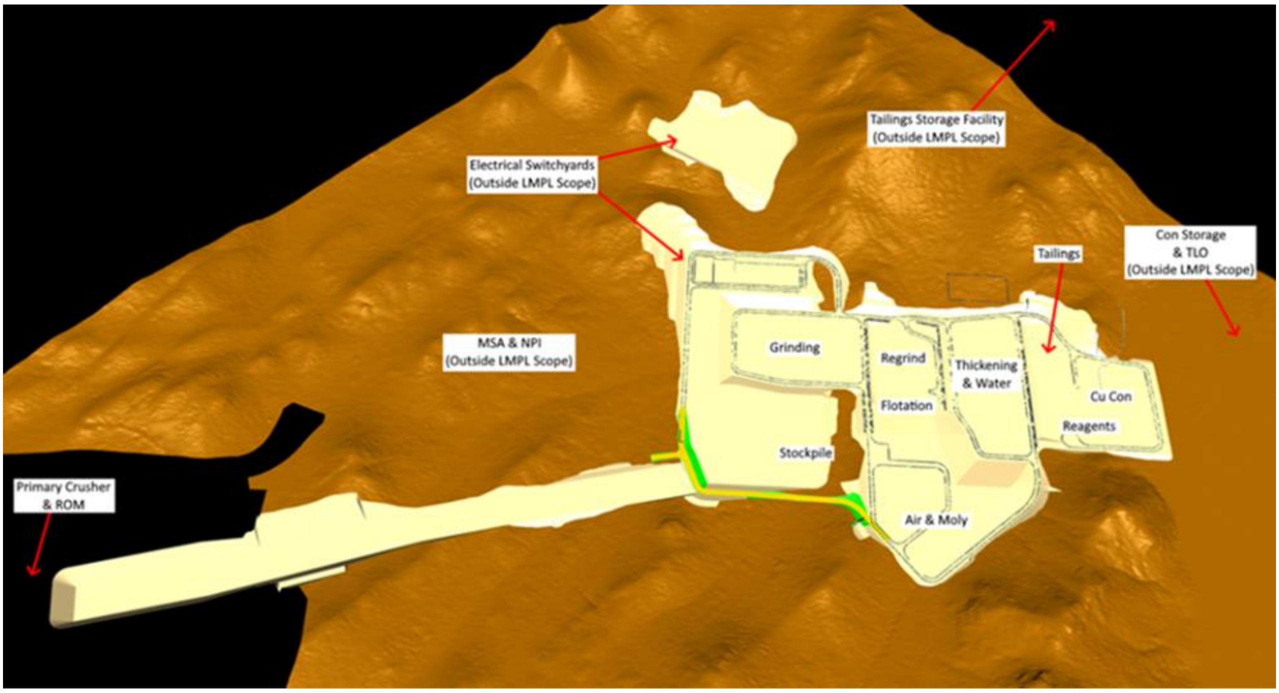
17.8 Plant site earthworks

From the work of Lycopodium (2025A), the earthworks design for the process plant reflects the current layout model, and the proposed plant location. Any further refinement of the layout will require optimisation of the earthworks quantities, in particular the cut/fill ratios, and optimising these on a cost basis with structural steel quantities, and other plant facility requirements.

Figure 17-8 shows the plant site earthworks and layout of pads. The earthworks design includes 1.48 Mm^3 of cut, and 0.98 Mm^3 of fill, distributed relatively evenly across the whole process plant site to produce the 'tiered' arrangement as shown. These tiers have been arranged to provide sufficient flat area for the equipment processes necessary, however possible future arrangements optimising towards slightly increased cut/fill quantities will allow for the rationalising of other materials (i.e. cables, piping, structural steel).

A preliminary earthworks schedule is provided in Chapter 9. This schedule distributes the mined NAF waste volume for site construction purposes and for forming the necessary waste dump base layer on the Salar de Arizaro.

Figure 17-8 Plant earthworks and pad design (source: Lycopodium, 2025A)



17.9 Plant structural engineering

The design of structures for the Project draws upon previous buildings/equipment in previous FQM projects (Lycopodium, 2025A). Most notably, the grinding and concentrate filtration buildings have been designed according to similar Sentinel specifications, although accounting for the seismicity differences between the two locations. The design of other process structures (e.g., flotation, regrind, molybdenum) is based upon the FQM Cobre Panamá structural designs, with external additional buildings as required.

Where a similar structure is not already available, the structural quantities rely upon a factored estimate incorporating seismic and wind loading factors as required.

17.10 Outstanding process design issues

Process designs have continued to evolve on the basis of additional testwork information and from operating experience gained at the Sentinel and Cobre Panamá projects.

Several design issues have not been resolved and will be dealt with during the detailed engineering phase of the Project, through internal discussions within the Project Team or via trade-off studies. The outstanding items that have been identified at present are described below. These are considered to be not material to the key processing design, recovery projections and assumptions made to date.

17.10.1 Gold recovery from the leach cap

The leach cap contains about 56 Mt of material containing no copper, but 0.41 g/t gold. This material has been subject to cyanide leaching testwork at various crush and grind sizes; indicating only about 40% gold recovery at 12 mm crush size but 90% at 80% passing 75 µm.

This material comprises part of the pre-strip and would be mined at a far greater rate than could be processed. It is proposed to separately stockpile this material for possible treatment in the future. Stockpiled material would be subject to further testwork to confirm recoveries and reagent consumptions at various size distributions.

This data would be fed into engineering trade off studies to determine the economics of either a heap leach or a tank leach operation at a potential throughput of 5 Mtpa for a Project life of 20 years.

If a gold circuit is recommended, it would be a separate circuit from the copper concentrator, but could share infrastructure, such as water supply, tailings facilities, technical management.

17.10.2 Molybdenum recovery from bulk concentrates

As discussed previously, the molybdenum recovery circuit will not be designed or constructed with the main copper concentrator but will be delayed for some months. Before it is decided to go ahead with the circuit design, an economic study will be undertaken to determine the economics of molybdenum production. This would be a relatively simple study, looking at the value of the concentrate produced (after transport and refining charges), the cost of production (reagents, labour, energy), and the estimated capital cost of the separation facilities.

17.10.3 Gland seal water or mechanical seals

Mechanical seals should be evaluated for all pumps at Taca Taca to minimise freshwater requirements to the Project. This may however lead to an increase in capital costs, and the cost effectiveness of mechanical seals needs to be evaluated.

17.10.4 Slurry discharge to the TSF

Tailings will be discharged via spigots arranged around the TSF. The shallow gradient from the edge of the salar to the centre may limit the spread of tailings from the deposition point. Likewise, initial seepage and evaporation might quickly dry out the tailings, further limiting flow to the centre of the dam.

A study undertaken on expected beach angles versus slurry density deposited on the TSF was undertaken by Tailpro Consulting (Tailpro, 2025) indicated that a tailings slurry density of 65% solids would be fluid and would achieve an average beach slope of only 0.15%. However, the high evaporation rates may lead to higher beach slopes. If this density creates too high a beach angle, the density can be reduced by increasing the brine quantity used for re-pulping.

There is thus a level of uncertainty as to how deposition of tailings will work in the early phases of operation. Further studies and discussions with other operations would be required to reduce the operating risk in this area of the Project.

17.10.5 Decant water return from the TSF

It is unclear, given the climatic conditions at Taca Taca and the use of a flat area on the salar, whether a supernatant pool of water would be formed that could be recycled to the plant.

This uncertainty impacts on the design of the decant and water return system and the plant water balance. At present it is assumed that no decant water will be available, and the plant water balance allows for increased brine supply for process water to make-up for this deficit.

Additional study work, and discussions with other operations that have used salars for tailings storage, will be necessary to determine the methodology of water return from the TSF, and the expected quantities of water available.

ITEM 18 PROJECT INFRASTRUCTURE

FQM commissioned several consultants to expand upon the preferred Project infrastructure components identified and reported on in the 2021 Technical Report. Suitably comprehensive reports, drawings and spreadsheet information packages were produced by the following engineering firms in respect of the Stage 1 40 Mtpa project:

- Lycopodium Ltd (Lycopodium, April 2025) were engaged to undertake the design work for the treatment plant (copper and molybdenum processing), primary crushing to copper concentrate filtration and tails pumping.
- Fluor Australia Pty Ltd (Fluor, October 2024 and February 2025) were engaged to design the mining and non-process infrastructure including the Salta Operations Centre, the site village, remote water and brine borefields, construction and off-site roads, airstrip, mining service area, copper concentrate conveyor and storage area, plus rail loading facilities.
- Process E&I were engaged to provide the electrical and instrument design drawings across the site, including HV switchyard, power distribution to mine, plant and infrastructure, and support for the HV powerline from La Puna to site.

Lycopodium conducted the design for the treatment facilities in collaboration with FQM's project development team, and leveraging on the design work from the recent Kansanshi S3 project, Cobre Panamá and Sentinel projects. Lycopodium also designed and estimated the cost for the tailings pumping system, referencing design work by SRK (March, 2025) on the tails deposition plan and the tailings embankment on the Salar de Taca Taca.

18.1 Overall site layout

A new design of the overall site layout, including the location of roads, rail and power lines, has been a collaborative effort between FQM and consultants Fluor and Process E&I.

The previously conceptual plant layout has been updated and designed in detail for 40 Mtpa processing, although inclusive of provisions to enable expansion to 60 Mtpa processing. The non-processing infrastructure (NPI) facilities have been located centrally between mining and process infrastructure to provide common utility, whilst separating heavy (HV) and light vehicle (LV) traffic movements. The selected laboratory location allows for easy access from the open pit, specifically for the delivery of grade control and process plant samples.

The mine services area (MSA) location has been selected in proximity to the mine site to maximise production efficiency whilst being able to maintain separation of the mine heavy vehicles and light vehicles traffic. The selected locations for the emulsion plant, ammonium nitrate (AN) storage facility and explosives magazine have been cognisant of both the mine and future operational extents, such that each doesn't hinder possible future expansion.

The administration and laboratory facilities are centrally located, off the mine main LV road. These have been separated from the process, mining and NPI areas whilst allowing direct access after entering the property, but without driving through other facilities. The design of the precinct has incorporated features to reduce as far as practicable, the amount of pedestrian and vehicle interaction.

The camp has been located away from mining, processing, MSA and NPI areas to reduce noise, dust, vibration and light from the operations, whilst minimising travel distance to and from the operations.

The immediate Project site layout and the broader overall layout are shown in Figure 18-1. In this figure, the ore stockpile design has not been updated to match that shown in Item 17. The alignment of the crushed ore conveyor has also been amended since this drawing was produced. Consequently, the MSA and NPI layouts as shown are now superseded. The latest design for each of these facilities is shown in separately updated drawings presented below.

18.2 Power supply

The preferred power supply route into the Project area can be summarised as follows:

- connection to a switching station at La Puna (which includes a solar power generation facility) on the existing 345 kV power line
- a new-build transmission line at 345 kV from La Puna along a preferred access route west of Pocitos and then crossing the Salar de Arizaro in a direct line to the Project site, over a distance of approximately 123 km
- a new-build 345/33 kV substation near the process plant
- 33 kV cables feeding from the site 345/33 kV substation to the plant main switchboard, which is located as close as possible to the large mill drives
- new-build 33kV transmission lines radiating from the plant main switchboard to each of the outlying fresh water borefields at Valle de Arizaro, Socompa, Valle de Chaschas and Valle de las Burras
- a new-build 33 kV transmission line from the plant main switchboard to the mine site and around the pit

18.2.1 Maximum installed power requirements

The estimated total power demand for the mine, processing facilities and infrastructure is expected to be approximately 205 MW to accommodate a future processing rate of up to 60 Mtpa. For the 40 Mtpa Stage 1 project the estimated maximum demand is 137 MW and 144 MVA.

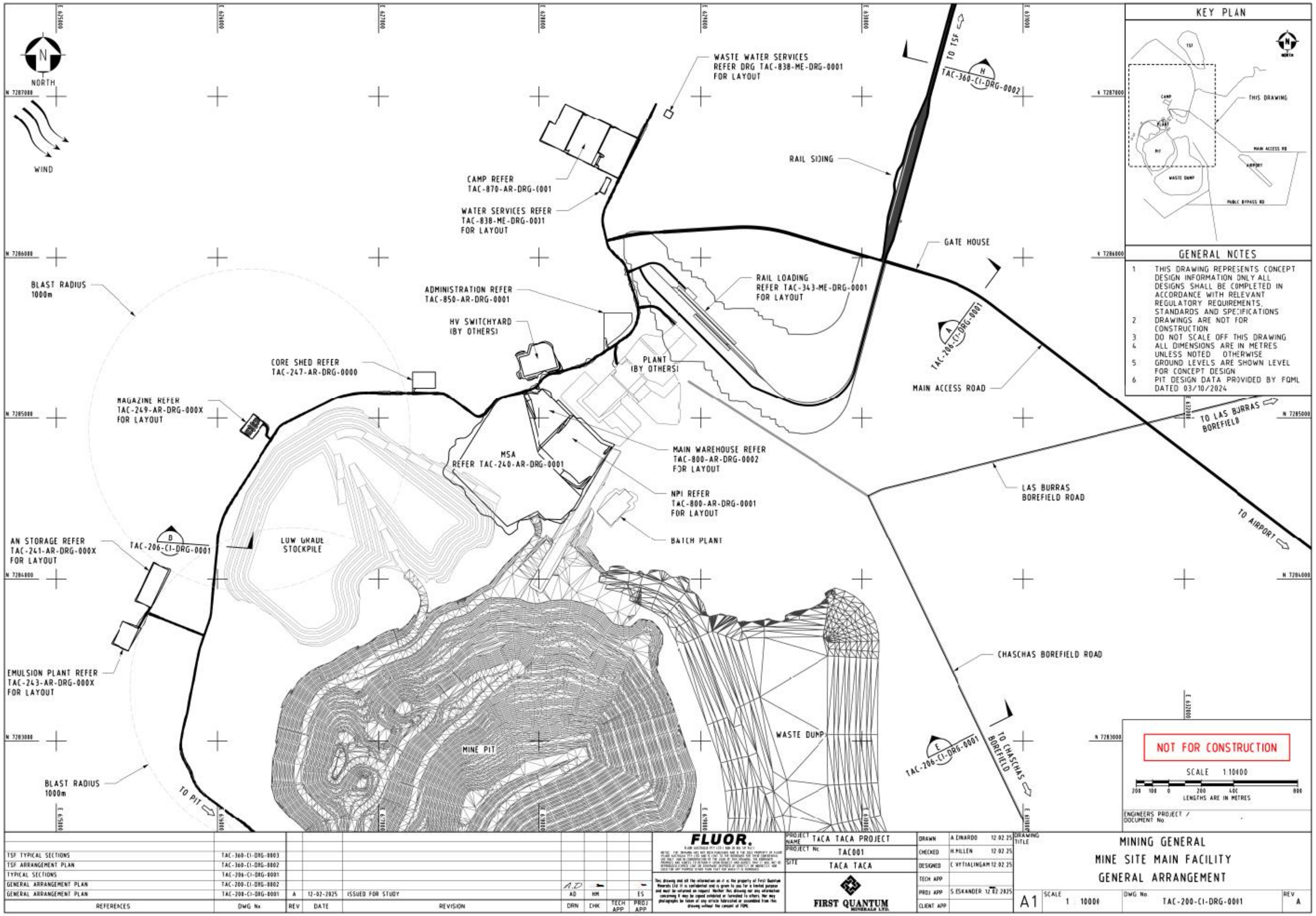
For Stage 1, the main substation design is based on 2 X 155/190 MVA ONAN/ONAF (oil natural air natural / oil natural air forced) transformers. However, only one transformer would be installed initially for the 40 Mtpa project with potential to add the second transformer at a future time during the production phase. As the layout detail for a future 60 Mtpa plant is unknown, it has been assumed that a new separate substation will be required under those circumstances.

To meet the requirements of the supply authorities, the site power factor would be corrected to 0.95 lagging or better, resulting in a site peak demand of 216 MVA. It should be noted that for a mining project at study stage, drive sizing, load estimates and load factors are based on preliminary information, and contingency is typically allowed in the power supply infrastructure.

The capacity of the overhead power line to meet the requirements of the 40 Mtpa initial installed capacity is based on the load demand and certain practicalities governing the transmission line design criteria. The criteria includes using a voltage of 345 kV because that is the voltage directly available at the La Puna substation without an additional step-down substation and also using a minimum conductor size and arrangement that can be used at that voltage level to control the corona effect and maintain the associated corona losses within the acceptable range.

In relation to the corona effect, it should be noted that at high altitude with lower air pressure, voltage derating is required, and so for a nominal operating voltage of 345 kV, the transmission line is designed for approximately 500 kV. Therefore, due to the above criteria, the rating of the transmission line is significantly higher than the demand requirement for the 40 MW rating of mine. Depending on the voltage drop criteria used, the transmission line rating for a load at the substation is approximately 520 MW at 0.95 p.f. (assuming 15% voltage drop).

Figure 18-1 Site infrastructure layout plan (source: Fluor Australia, February 2025)



The future demand for a 60 Mtpa Stage 2 is within the capability of the proposed transmission line. From 'order of magnitude' estimates, there will not be a significant saving using a 220 kV powerline when considering the requirement for a step-down 345/220 kV substation, and the general overhead costs associated with any major transmission line. However, it is expected that this will be examined in more detail during the continuing engineering phase.

18.2.2 On-site supply requirements and substations

Power distribution at the Project site would be primarily for the following areas, at 33 kV, for all areas including:

- the process plant
- the mine
- MSA
- ancillary facilities, administration, workshops, warehouse, etc.
- camp
- remote areas, e.g. TSF, water reservoir, water return pumping system, potable water pumping system, etc.

33 kV powerlines will radiate from the plant main 33 kV substation to each of the outlying fresh water borefields at Valle de Arizaro, Socompa, Valle de Chaschas and Valle de las Burras.

Should there be increases in the estimated borefield loads, changes in location or longer powerline route lengths, then a 33/66 kV transformer and 66 kV powerline to one or more locations may be required.

The overhead line design, method of construction and supply network tie-in point at the Project site are shown in Figure 18-2. Detailed block diagrams and single line diagrams are included in Appendix G for power supply to the mining operations (including provision for trolley assisted haulage), the primary/pebble crushing area, crushed ore reclaim, the processing plant, the NPI facilities, four regional borefields and brine pumping facilities.

Instead of installing a power line to the airport with associated transformer, the airport will be solar powered (including battery support) due to the relatively low estimated load.

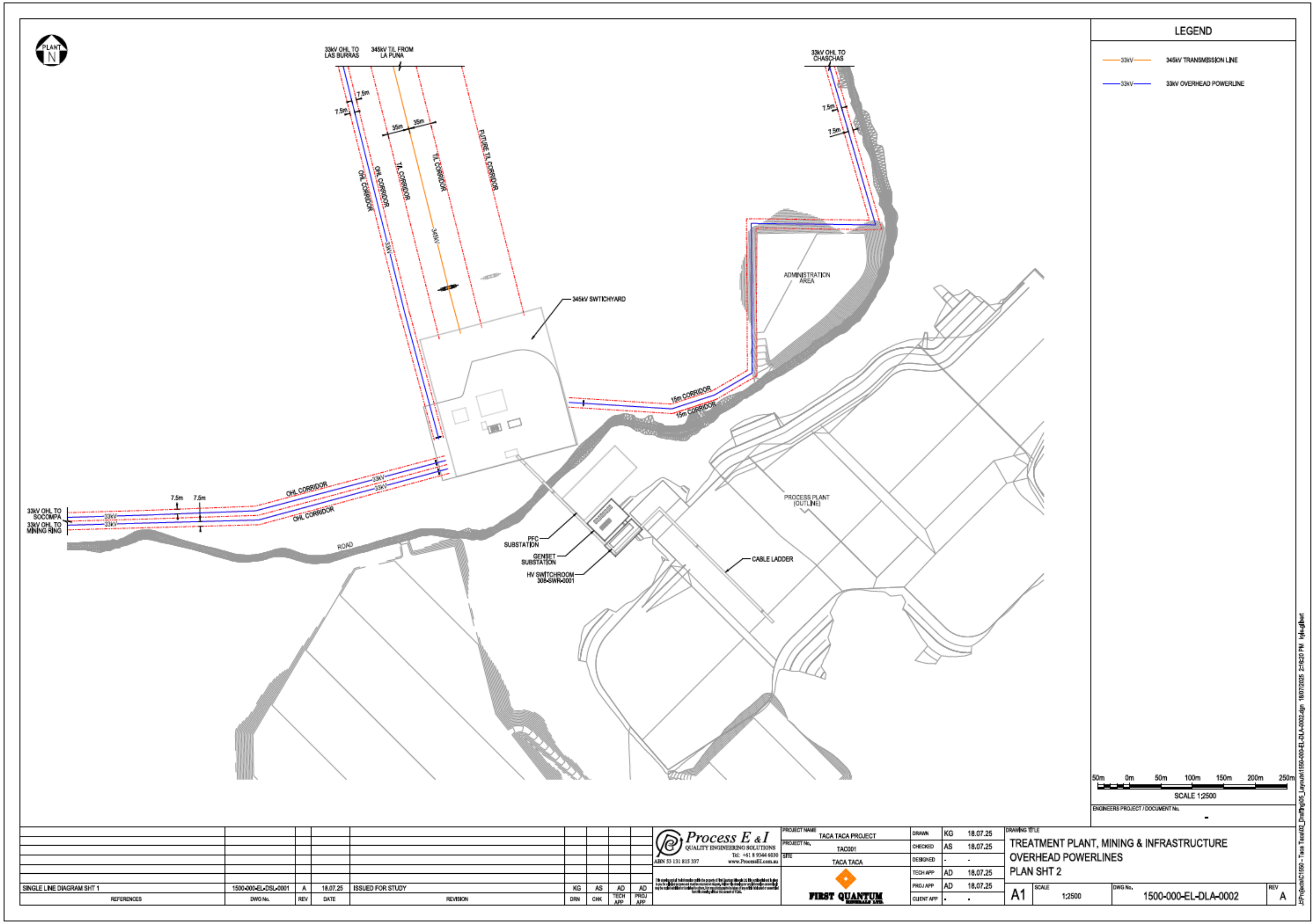
Project site

Referring to Figure 18-2, a new substation would be established near the plant site to step down the incoming 345 kV supply from the La Puna switching station to 33 kV. The various load centres would be provided with substations for the distribution of electricity to the drives and other services, as listed above.

Each substation would include a step-down transformer (pad or pole mounted), a set of pole fuses or switchgear, a set of arrestors for lightning protection, a motor control centre (MCC), lighting and a small power board. The MCC (motor control centre) and the lighting distribution board would be housed in a standard substation building. The power transformer would be oil filled, featuring ONAN cooling with off-load tap changes. KNAN (K non-mineral oil) transformers may be considered for specific locations and applications.

The mine services, NPI facilities and camp would be provided with 'kiosk' type packaged substations, where the power transformer and the low voltage (LV) switchboard are housed in a sealed type transportable 'kiosk'. Simple, air-conditioned, block work substation buildings with concrete floor slabs and metal clad roofs would be provided. Prefabricated switch rooms will be considered for specific applications, especially where loads and design are standard and fixed.

Figure 18-2 Power supply diagram and switchyard at the Project site.



33 kV cables will feed from the plant main switchboard to other distribution switchboards in plant areas, and to the harmonic filters and emergency generator transformers. 33/6.6 kV step-down transformers in plant areas and associated 6.6kV switchboards will supply power to large motors.

The low voltage general power supply is 380/220 V, and the majority of motors are rated at 380 V, whilst the 380V MCCs are fed from 33/0.4 kV transformers. Note that a rating of 690 V was considered for motors, but the altitude-related voltage derating made this unviable in general.

Due to the high altitude, the site 33 kV switchboards are the GIS (gas insulated) type. In general, the 6.6 kV switchboards are AIS (air insulated), but derated for altitude. Low voltage switchgear is also derated for altitude, meaning that the switchboard voltage rating is significantly higher than the nominal voltage applied.

Mine site

Distribution of power for mine operations is planned to be implemented using a surface ring main system distributing power at 33 kV around the pit edge to provide power for the electric mining equipment (rope shovels and drills) as well as surface and inpit dewatering and lighting equipment. Power is planned to be transmitted to the operating benches by a series of portable substations (approximate capacity of 5 to 7.5 MVA) and associated cables. Each substation will be able to provide power for one shovel and two large drills. Power for the trolley assist system (if adopted) is planned to be distributed by a series of dedicated overhead power lines on each haulage ramp and associated substations (‘e-houses’) located at approximately 350 m intervals along the haul ramp

18.2.3 Construction and emergency power supply

From Project approval, and until such time as the permanent power infrastructure is constructed and commissioned to site, the operations will be reliant on local diesel power generation.

Initial site establishment works will rely on small, dedicated diesel generators, with mobile fuel storage and dispensing systems, satisfied by a centralised diesel fuelling facility and regular diesel fuel delivery from Salta.

It is anticipated that the site emergency generators will be established as quickly as practical and using expendable buried cables; short term these will provide construction power to the near-site construction areas.

18.3 Water supply

Regional borefields will be developed to supply fresh and brine water for the Project. Field investigations and analyses are in progress to assess the viability of brine extraction wells in place of collection into and pumping from excavated trenches in the salar. There are several identified fresh water supply basins, namely at Valle de Arizaro, Socompa, Valle de Chaschas and Valle de las Burras, the combined abstraction from which are currently determined to be adequate for a 40 Mtpa operation. Additionally, there are a number of more distant fresh water supply sources being evaluated to support a 60 Mtpa operation.

Because of the fundamental importance of water supply to the Project, detailed information on the status of supply investigations is provided under Item 24.

18.3.1 Project water requirements

The Project at 40 Mtpa, including processing, mining and the camp, is expected to consume:

- an average of 2,130 m^3/hr or 51,120 m^3/day of raw water, and
- an average of 600 m^3/hr or 14,400 m^3/day of brine

For a production rate expanded to 60 Mtpa, the respective average consumption rates would be 76,560 m^3/day and 20,400 m^3/day.

Raw water will be used for potable water production, fire water use, and processing, whilst brine will be used almost exclusively for tailing dilution and dust suppression.

All the water requirements for the Project will be extracted sustainably from ground water sources and below-ground catchments in the region.

Table 18-1 lists the complete water demand from all sources, summarised by fresh and brine demand quantities, and by processing, mining, camp and other consumption activities. Table 18-2 lists the corresponding complete water balance itemisation, including water lost into the ground, into tailings, and due to evaporation.

18.3.2 Fresh water

Four fresh water catchments have been identified and permitted for extraction of fresh water to sustain the 40 Mtpa Stage 1 operational requirements. In each case these catchments represent below ground alluvium basins, providing the storage of snow melt from the mountainous peaks within their catchment.

The catchments include Valle de Arizaro, Socompa, Valle de Chaschas and Valle de Las Burras. For the Valle de Arizaro catchments, approximately half the bore samples suggest brackish water. For this reason, and in this particular catchment, pumping of the water could be separated into “sweet” and “brackish” water lines.

Within the catchments, water will be extracted from several individual bores, and pumped to a central bolted steel catchment tank, and thereafter transferred overland by pipeline to the processing facilities. Power, control and communications will be routed to the catchment along the right of way, a corridor accommodating an overhead power line, pipeline(s) and access road. All in-field tanks have been selected for commonality; they are 5.4 m nominal ID (inside diameter) and 6 m high.

These waters may be stored separately to reduce the extent of treatment required at the plant for the small portion of raw water which reports to the water treatment plant for potable water, concrete batch plant and reagent mixing requirements.

Table 18-1 Summary water demand for all consumption activities

Water Demand; Stage 1 Project	Average, 40 Mtpa				Peak, 40 Mtpa			
	ML/annum	**kL/day**	**m^3/h**	**L/s**	**ML/annum**	**kL/day**	**m^3/h**	**L/s**
Fresh water demand								
for processing	17,214.4	47,162.6	1,965.1	545.9	17,214.4	47,162.6	1,965.1	545.9
for the camp	75.6	207.0	8.6	2.4	189.8	520.0	21.7	6.0
for mining	5.3	14.5	0.6	0.2	15.9	43.5	1.8	0.5
for site services, rail load-out & construction	7.3	20.0	0.8	0.2	21.9	60.0	2.5	0.7
Subtotal	**17,302.5**	**47,404.1**	**1,975.2**	**548.7**	**17,441.9**	**47,786.1**	**1,991.1**	**553.1**
Brine demand								
for processing	4,124.6	11,300.3	470.8	130.8	4,124.6	11,300.3	470.8	130.8
for mining	1,089.6	2,985.1	124.4	34.6	2,179.1	5,970.2	248.8	69.1
for road maintenance & construction	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**5,243.4**	**14,365.4**	**598.6**	**166.3**	**6,376.8**	**17,470.6**	**727.9**	**202.2**
Water in ore processed	1,354.6	3,711.3	154.6	43.0	1,354.6	3,711.3	154.6	43.0
TOTAL	**23,900.5**	**65,480.9**	**2,728.4**	**757.9**	**25,173.3**	**68,968.0**	**2,873.7**	**798.2**
Processing summary								
fresh make-up	17,214.4	47,162.6	1,965.1	545.9	17,214.4	47,162.6	1,965.1	545.9
brine make-up	4,124.6	11,300.3	470.8	130.8	4,124.6	11,300.3	470.8	130.8
water in ore processed	1,354.6	3,711.3	154.6	43.0	1,354.6	3,711.3	154.6	43.0
Subtotal	**22,693.6**	**62,174.3**	**2,590.6**	**719.6**	**22,693.6**	**62,174.3**	**2,590.6**	**719.6**
Mining summary								
fresh	5.3	14.5	0.6	0.2	15.9	43.5	1.8	0.5
brine	1,089.6	2,985.1	124.4	34.6	2,179.1	5,970.2	248.8	69.1
Subtotal	**1,094.9**	**2,999.6**	**125.0**	**34.7**	**2,195.0**	**6,013.7**	**250.6**	**69.6**
Camp and other								
fresh	83.5	228.7	9.5	2.6	211.7	580.0	24.2	6.7
brine	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**112.7**	**308.7**	**12.9**	**3.6**	**284.7**	**780.0**	**32.5**	**9.0**
TOTAL	**23,901.2**	**65,482.6**	**2,728.4**	**757.9**	**25,173.3**	**68,968.0**	**2,873.7**	**798.2**

Water Demand; Stage 2 Project	Average, 60 Mtpa				Peak, 60 Mtpa			
	ML/annum	**kL/day**	**m^3/h**	**L/s**	**ML/annum**	**kL/day**	**m^3/h**	**L/s**
Fresh water demand								
for processing	25,821.5	70,743.9	2,947.7	818.8	25,821.5	70,743.9	2,947.7	818.8
for the camp	76.2	208.7	8.7	2.4	189.8	520.0	21.7	6.0
for mining	5.3	14.5	0.6	0.2	15.9	43.5	1.8	0.5
for site services, rail load-out & construction	7.3	20.0	0.8	0.2	21.9	60.0	2.5	0.7
Subtotal	**25,910.3**	**70,987.1**	**2,957.8**	**821.6**	**26,049.1**	**71,367.4**	**2,973.6**	**826.0**
Brine demand								
for processing	6,186.9	16,950.5	706.3	196.2	6,186.9	16,950.5	706.3	196.2
for mining	1,196.1	3,277.0	136.5	37.9	2,392.2	6,554.0	273.1	75.9
for road maintenance & construction	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**7,412.2**	**20,307.5**	**846.1**	**235.0**	**8,652.1**	**23,704.5**	**987.7**	**274.4**
Water in ore processed	2,032.0	5,567.0	232.0	64.4	2,032.0	5,567.0	232.0	64.4
TOTAL	**35,354.5**	**96,861.6**	**4,035.9**	**1,121.1**	**36,733.2**	**100,638.8**	**4,193.3**	**1,164.8**
Processing summary								
fresh make-up	25,821.5	70,743.9	2,947.7	818.8	25,821.5	70,743.9	2,947.7	818.8
brine make-up	6,186.9	16,950.5	706.3	196.2	6,186.9	16,950.5	706.3	196.2
water in ore processed	2,032.0	5,567.0	232.0	64.4	2,032.0	5,567.0	232.0	64.4
Subtotal	**34,040.4**	**93,261.3**	**3,885.9**	**1,079.4**	**34,040.4**	**93,261.3**	**3,885.9**	**1,079.4**
Mining summary								
fresh	5.3	14.5	0.6	0.2	21.9	60.0	2.5	0.7
brine	1,196.1	3,277.0	136.5	37.9	2,392.2	6,554.0	273.1	75.9
Subtotal	**1,201.4**	**3,291.5**	**137.1**	**38.1**	**2,414.1**	**6,614.0**	**275.6**	**76.6**
Camp and other								
fresh	83.5	228.7	9.5	2.6	211.7	580.0	24.2	6.7
brine	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**112.7**	**308.7**	**12.9**	**3.6**	**284.7**	**780.0**	**32.5**	**9.0**
TOTAL	**35,354.5**	**96,861.6**	**4,035.9**	**1,121.1**	**36,739.2**	**100,655.3**	**4,194.0**	**1,165.0**

Table 18-2 Summary water balance for all consumption activities

Overall Water Balance 40 Mtpa processing		Average m^3/h	Peak m^3/h	Overall Water Balance 60 Mtpa processing		Average m^3/h	Peak m^3/h
Fresh water In				**Fresh water In**			
	regional borefields	2,776.0	3,190.0		regional borefields	3,712.0	4,126.0
	Pit dewatering/pit slope drainage	81.0	108.0		pit slope drains/bores	81.0	108.0
	Subtotal	**2,857.0**	**3,298.0**		**Subtotal**	**3,793.0**	**4,234.0**
To Operations				**To Operations**			
	process plant	1,965.1	1,965.1		process plant	2,947.7	2,947.7
	water in ore processed	154.6	154.6		water in ore processed	232.0	232.0
	mine operations	0.6	1.8		mine operations	0.60	1.8
	camp. etc	9.5	24.2		camp. etc	9.53	24.2
	Subtotal	**2,129.8**	**2,145.7**		**Subtotal**	**3,189.8**	**3,205.6**
Water consumption				**Water consumption**			
	process plant				process plant		
	in concentrates	8.0	8.0		in concentrates	12.0	12.0
	in tailings	2,111.7	2,111.7		in tailings	3,167.6	3,167.6
	mine operations				mine operations		
	to sewage treatment	8.7	21.7		to sewage treatment	8.70	21.7
	evaporation	1.4	4.3		evaporation	1.44	4.3
	Subtotal	**2,129.9**	**2,145.7**		**Subtotal**	**3,189.8**	**3,205.6**
Fresh water supply surplus/defecit		**881.8**	**1,306.9**	**Fresh water supply surplus/defecit**		**835.2**	**1,260.4**
Brine water In				**Brine water In**			
	salar trenches and bores	828.0	1,008.0		salar trenches and bores	828.0	1,008.0
	tails seepage	180.0	180.0		tails seepage	180.0	180.0
	TSF return	0.0	0.0		TSF return	0.0	0.0
	Subtotal	**1,008.0**	**1,188.0**		**Subtotal**	**1,008.0**	**1,188.0**
To Operations				**To Operations**			
	process plant	470.8	470.8		process plant	706.3	706.3
	mine operations	124.4	248.8		mine operations	136.5	273.1
	road maint. etc	3.3	8.3		road maint. etc	3.3	8.3
	Subtotal	**598.6**	**727.9**		**Subtotal**	**846.1**	**987.7**
Water consumption				**Water consumption**			
	process plant				process plant		
	evaporation at TSF	470.8	470.8		evaporation at TSF	706.3	706.3
	mine operations				mine operations		
	to the ground	14.9	29.9		to the ground	16.0	31.9
	evaporation	112.8	227.2		evaporation	123.9	249.6
	Subtotal	**598.6**	**727.9**		**Subtotal**	**846.1**	**987.7**
Brine water supply surplus/defecit		**409.4**	**460.1**	**Brine water supply surplus/defecit**		**161.9**	**200.3**
Water to TSF				**Water to TSF**			
	Brine				**Brine**		
	Brine consumption in plant	-470.8	-470.8		Brine consumption in plant	-706.3	-706.3
	Brine to TSF (evaporated)	470.8	470.8		Brine to TSF (evaporated)	706.3	706.3
	Fresh				**Fresh**		
	Fresh consumption in plant	-2,119.7	-2,119.7		Fresh consumption in plant	-3,179.6	-3,179.6
	Fresh to concentrates	8.0	8.0		Fresh to concentrates	12.0	12.0
	Fresh to TSF	**2,111.7**	**2,111.7**		**Fresh to TSF**	**3,167.6**	**3,167.6**
	Balance	**0.0**	**0.0**		**Balance**	**0.0**	**0.0**
TSF water balance				**TSF water balance**			
	Total to TSF	2,582.6	2,582.6		Total to TSF	3,873.9	3,873.9
	TSF decant	0.0	0.0		TSF decant	0.0	0.0
	Total evaporation	-470.8	-470.8		Total evaporation	-706.3	-706.3
	Total in settled solids	-2,111.7	-2,111.7		Total in settled solids	-3,167.6	-3,167.6
	Balance	**0.0**	**0.0**		**Balance**	**0.0**	**0.0**

18.3.3 Brine

Brine is proposed for use in dust suppression applications across the site, but predominantly for dilution of thickened tails prior to their deposition on the Salar de Taca Taca. The use of brine allows for a higher amount of water recovery and recycling of process water in the processing facilities. The introduction of brine as a diluent for the thickened tails allows for practical pumping and deposition whilst inhibiting dust generation through the development of a salt crust at the surface.

The use of trenches for brine extraction has been used with success in lithium bearing salars; they are seen to be cost effective and fast to establish but could be unproductive over time due to a shallow drawdown curve and propensity to exhaust the available brine within the trench depth. Whilst bores are more costly to establish, they are more suitable for long term sustained abstraction. Despite this observation, recent Company investigations indicate that deep bores may prove to be more effective than brine collection in excavated trenches.

Item 24 mentions modest individual brine trench abstraction rates of 10 L/s to 12 L/s (i.e., up to approximately 45 m^3/hr) indicated from preliminary assessments. Piteau (2025) scaled this supply up to a potential average of 234 m^3/hr of brine from an array of conceptual trenches.

The corresponding Piteau estimate of brine supply from a borefield was 450 m^3/hr. Field investigations in the second half of 2025 have since indicated a potential for up to 1,000 m^3/hr of borefield supply. Further information on this projected increase in brine borefield supply is provided in Item 24.

18.3.4 Fresh water borefields and pipelines

Figure 18-3 shows the location of the identified potential primary sources of Project fresh water supply. This figure also shows the overland pipeline routes from the water basin catchments/borefield sources to the Project site.

Valle de Arizaro

The Valle de Arizaro borefield is anticipated to yield an average of 380 m^3/h (with a peak of 460 m^3/h). The borefield has been devised to allow separation of “sweet” and “brackish” quality water for treatment and consumption, to enable gravity flow and keep all piping within HDPE pressure capability.

The Valle de Arizaro bore pumps are designed for a nominal flowrate of 17.7 L/s and an average head of 222 m (Figure 18-4). The in-field piping for Valle de Arizaro is listed in Table 18-3 with no throttling valves required.

Figure 18-5 and Figure 18-6show the Valle de Arizaro borefield layout.

The common tank sizing and the Valle de Arizaro flow rates result in a residence time of the in-field tanks of 19 minutes.

Figure 18-3 Potential fresh water supply sources and pipeline routes (source: Fluor, 2024)

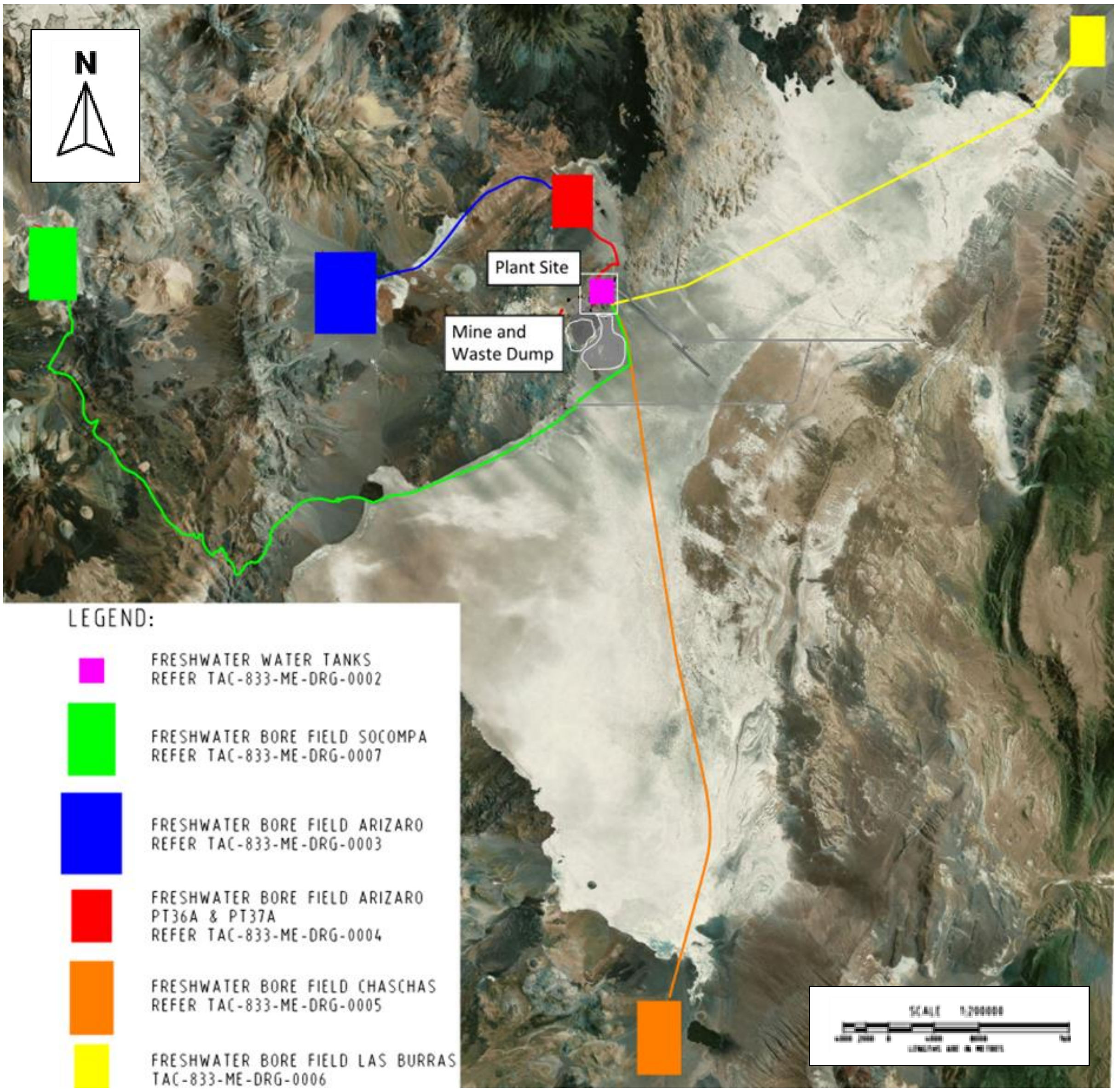


Figure 18-4 Profile of freshwater pipeline routes from Valle de Arizaro to the plant (source: Fluor, 2024)

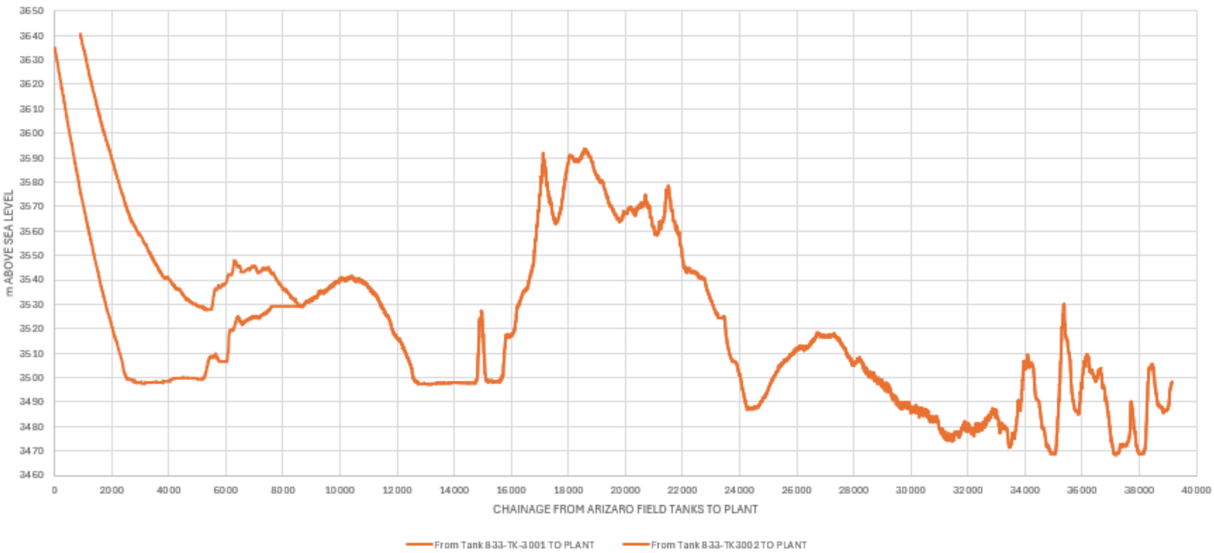


Table 18-3 Valle de Arizaro borefield pipework details

Bore/well	Distance to tank (m)	Pipe specification
Valle de Arizaro 1		
TW14	1,378	DN250 PN10 SDR17 HDPE
T22	2,987	DN250 PN16 SDR11 HDPE
pAPW-VdA-02	3,498	DN250 PN16 SDR11 HDPE
pAPW-VdA-03	1,400	DN250 PN10 SDR17 HDPE
pAPW-VdA-05	1,887	DN250 PN16 SDR11 HDPE
pAPW-VdA-08	3,828	DN250 PN16 SDR11 HDPE
Arizaro Tank 1 subtotal	**14,978**	
Valle de Arizaro 2		
TW12	1,040	DN250 PN10 SDR17 HDPE
TW13	2,283	DN250 PN10 SDR17 HDPE
pAPW-VdA-01	2,989	DN250 PN16 SDR11 HDPE
pAPW-VdA-04	1,228	DN250 PN10 SDR17 HDPE
Arizaro Tank 2 subtotal	**7,540**	

The water will be gravitated through transfer lines back to the plant storage where throttling valves will be required for discharge into the plant raw water tank farm:

- Tank 1 branch consists of 1.3 km of DN450 PN 10 SDR 17, 8 km of DN450 PN 16 SDR 11 HDPE pipelines and then the main line.
- Tank 2 branch consists of 1.3 km of DN450 PN 10 SDR 17, 7.4 km of DN450 PN 16 SDR 11 HDPE pipelines and then the main line.
- The main line, duplicated for each tank, consist of 15.5 km of DN630 PN16 SDR 11, 0.6 km of DN 500 PN20 SDR 9, 4.8 km of DN630 PN16 SDR 11 and 9.6 km of DN500 PN20 SDR 9 HDPE pipelines.

Figure 18-5 Layout of Valle de Arizaro borefield – sheet 1 (source: Fluor, 2024)

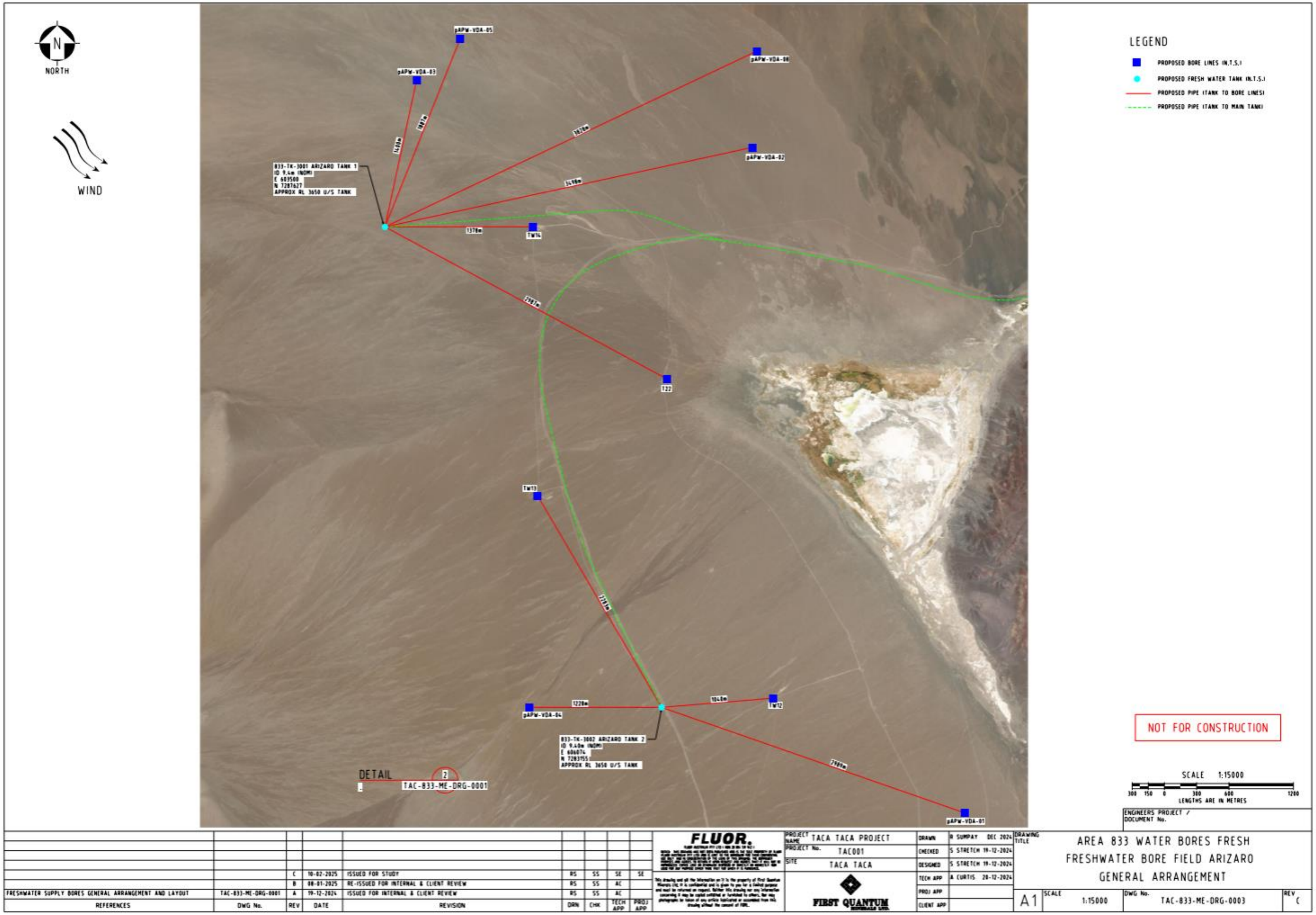
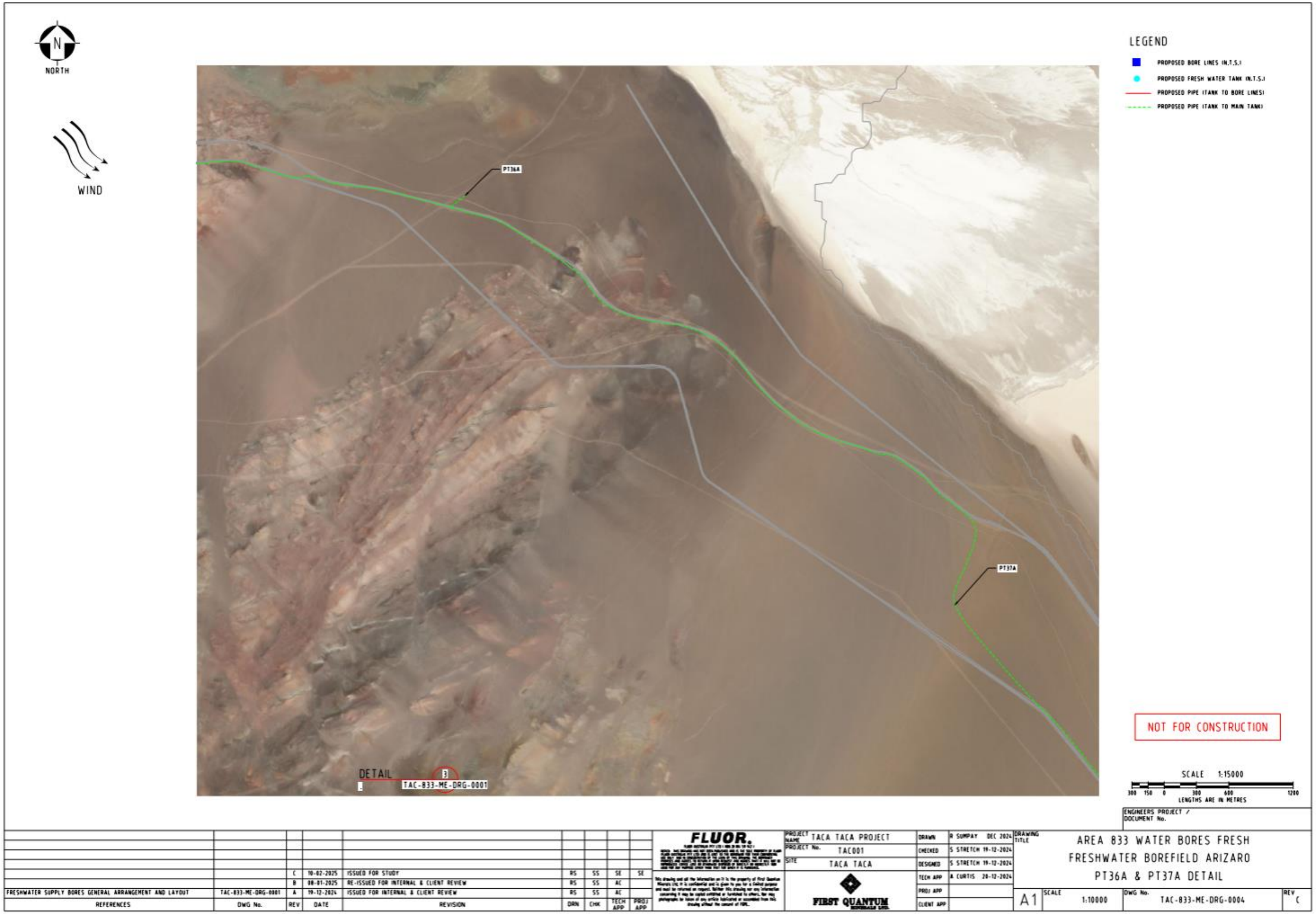


Figure 18-6 Layout of Valle de Arizaro borefield – sheet 2 (source: Fluor, 2024)



Socompa

The Socompa borefield is anticipated to yield an average of 666 m^3/h (with a peak of 810 m^3/h).

The bore pumps have been designed for a nominal flow rate of 35.1 L/s and an average head of 119 m (Figure 18-7). The piping required is as listed in Table 18-4, with no throttling valves necessary.

Table 18-4 Socompa borefield pipework details

From	To	Distance to tank (m)	Pipe specification
TW21	pAW-SOC-04	709	DN250 PN10 SDR17 HDPE
pAW-SOC-04	TW20	3,148	DN355 PN10 SDR17 HDPE
TW20	pAPW-Soc-01	535	DN400 PN10 SDR17 HDPE
pAPW-Soc-01	Qda. Del Agua	1,325	DN450 PN10 SDR17 HDPE
Qda. Del Agua	pAPW-Soc-03	1,554	DN500 PN10 SDR17 HDPE
pAPW-Soc-03	pAPW-Soc-02	697	DN630 PN10 SDR17 HDPE
pAPW-Soc-02	Socompa Tank	991	DN630 PN10 SDR17 HDPE
Socompa	Socompa Tank	697	DN250 PN10 SDR17 HDPE
TW19	Socompa Tank	50	DN250 PN10 SDR17 HDPE
subtotal		**9,706**	

The layout of the Socompa borefield allows one main pipe header collecting from most of the bores and requiring a larger pipe as more flow is collected from each bore. TW19 and the Socompa bore are individually routed to the Socompa collection tank. Figure 18-8 shows the layout of the Socompa borefield.

Figure 18-7 Profile of freshwater pipeline route from Socompa (source: Fluor, 2024)

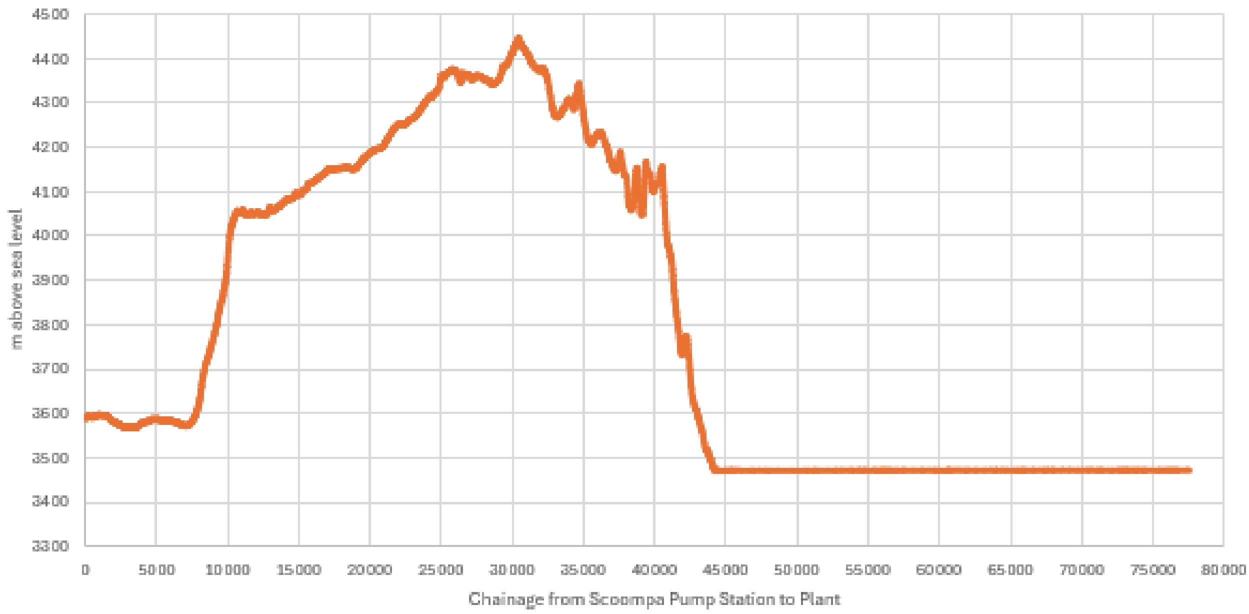
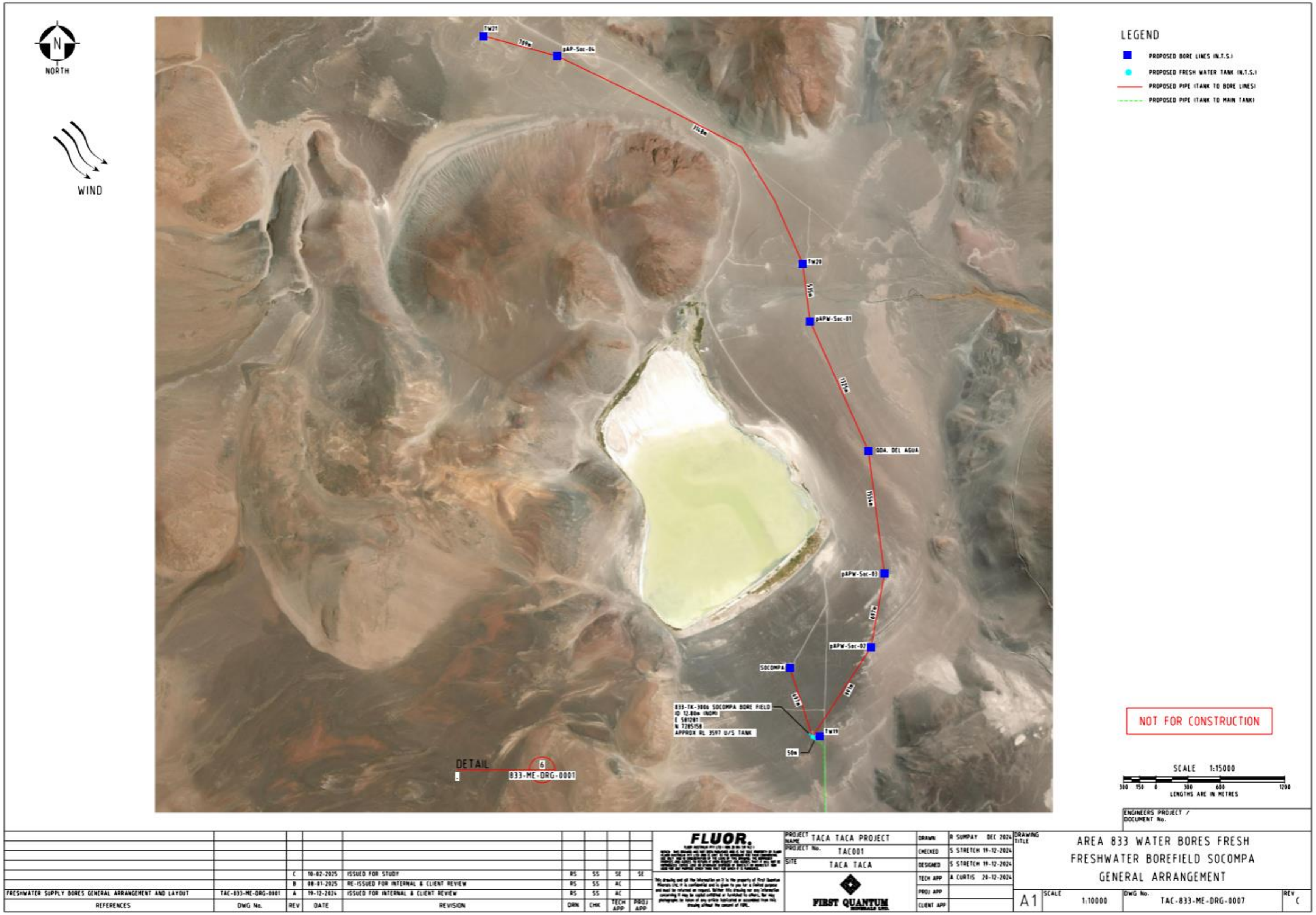


Figure 18-8 Layout of Socompa borefield (source: Fluor, 2024)



The common in-field tank sizing with the nominal Socompa flow rate gives a residence time of the in-field tank of seven minutes.

The water would be pumped via a high pressure positive displacement pump to overcome the terrain-imposed flow constraints. The pump duty is 246 L/s with a TDH requirement of 890 m, without any additional factors. There is a section of transfer line requiring steel pipe to contain the pressures associated with the transfer of the water across the terrain, namely:

- 11.9 km of DN500 SCH 80
- 16.5 km DN500 SCH STD WT
- 18.12 km of DN400 SCH STD WT
- 3.16 km of DN400 SCH XS

There is then a break tank with the same sizing as the in-field collection tank. From this break tank, the water would be gravitated to a plant storage tank by 36.1 km of DN630 PN16 SDR 11 HDPE pipe, with a throttling valve on tank discharge.

Valle de Chaschas

The Valle de Chaschas borefield is anticipated to yield an average of 570 m^3/h of water (with a peak of 705 m^3/h). The borefield pumps are designed for a 33.3 L/s flowrate and an average head of 185 m (Figure 18-9). The in-field piping required for Valle de Chaschas is listed in Table 18-5.

Figure 18-10 shows the layout of the Valle de Chaschas borefield.

Table 18-5 Valle de Chaschas borefield pipework details

Bore/well	Distance to tank (m)	Pipe Specification
Valle de Chaschas		
TW22	1,890	DN 280 SDR 11 PN16 HDPE
TW23	195	DN 280 SDR 11 PN16 HDPE
pAPW-CHA-01	1,468	DN 280 SDR 11 PN16 HDPE
pAPW-Cha-02	1,796	DN 280 SDR 11 PN16 HDPE
pAPW-Cha-03	3,547	DN 280 SDR 11 PN16 HDPE
pAPW-Cha-04	1,615	DN 280 SDR 11 PN16 HDPE
pAPW-Cha-05	5,289	DN 280 SDR 11 PN16 HDPE
subtotal	**15,800**	

Due to elevations along the pipe route, bores pAPW-Cha-05 and pAPW-Cha-03 require throttling valves at the discharge into the field tank.

The common in-field tank sizing and the nominal flowrate of 720 m^3/h give a residence time of nine minutes.

The transfer line would be gravity flow back to the plant storage tank with throttling valves required for discharge. Due to elevations and distance along the route, there are three different HDPE pipe specifications required, namely:

- approximately 2.5 km of DN630 PN10 SDR 17
- approximately 2.7 km of DN630 PN12 SDR 13.6
- approximately 62.6 km of DN630 PN16 SDR 11

Figure 18-9 Profile of freshwater pipeline route from Valle de Chaschas to the plant (section viewed to the west) (source: Fluor, 2024)

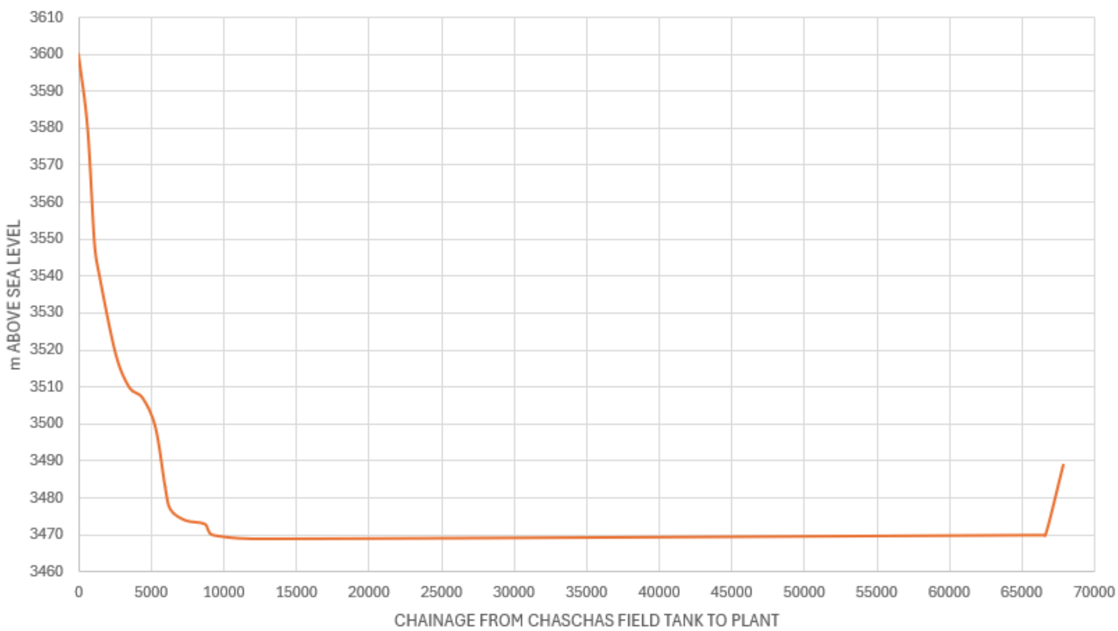
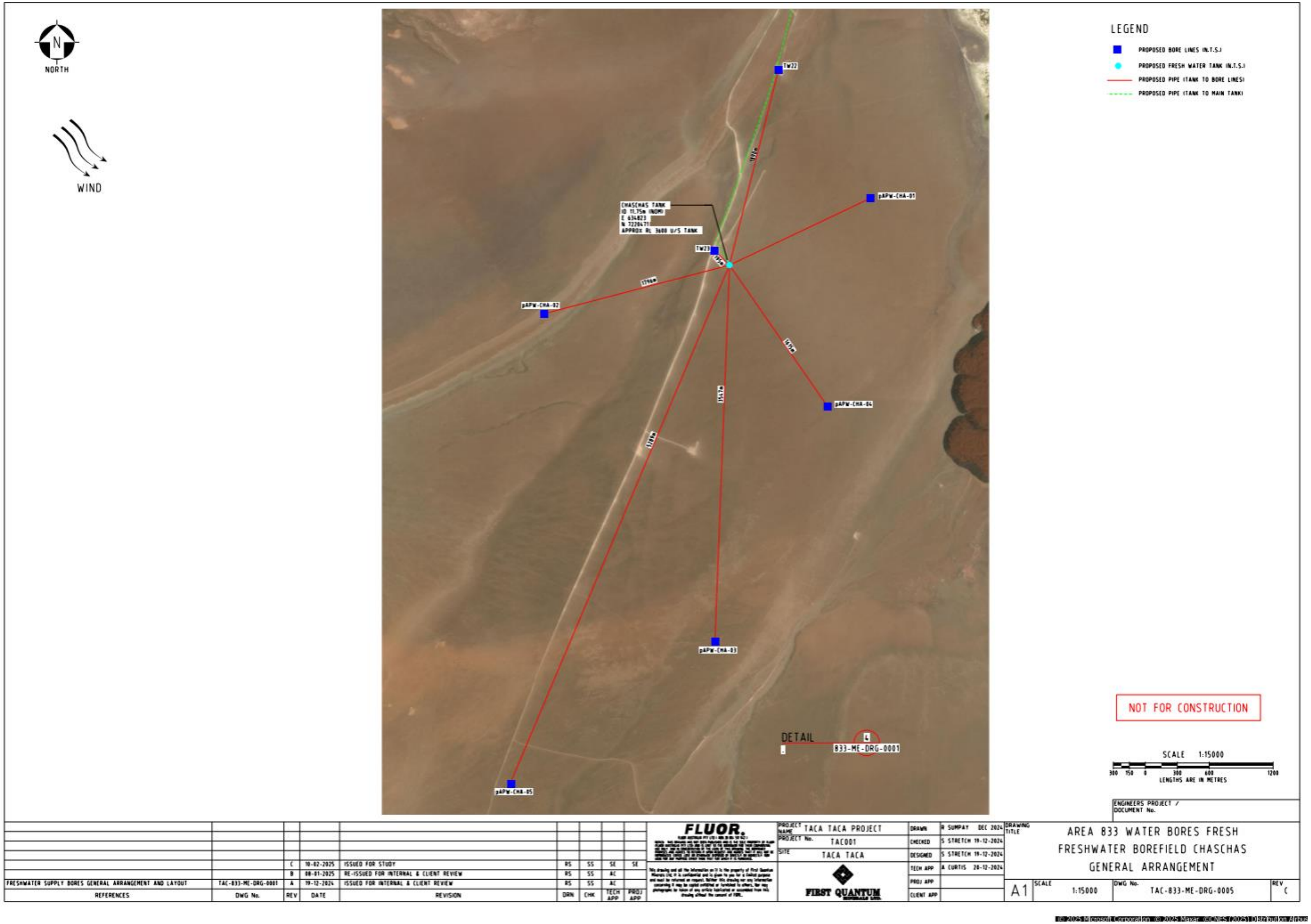


Figure 18-10 Layout of Valle de Chaschas borefield (source: Fluor, 2024)



Valle de las Burras

The Valle de Las Burras borefield is anticipated to yield an average of 457 m^3/h (with a peak of 511 m^3/h).

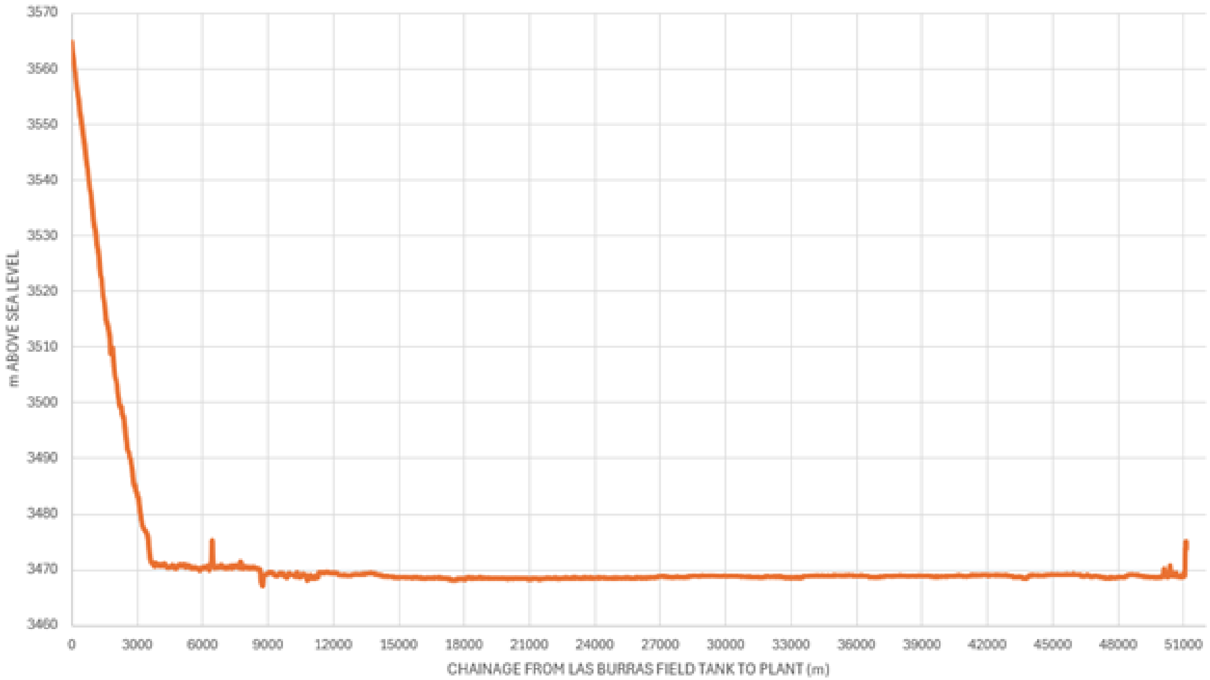
Due to the large volume of water provided by the Valle de las Burra borefield, it has been split into two fields, with two field tanks and two transfer lines to provide redundancy. The las Burras bore pumps are designed for a 63.3 L/s flowrate with an average head of 185 m (Figure 18-11). The in-field piping required for Valle de las Burras is listed in Table 18-6.

Figure 18-12 shows the layout of the Valle de Las Burras borefield.

Table 18-6 Valle de las Burras borefield pipework details

Bore/well	Distance to tank (m)	Pipe specification
Las Burras 1		
TW10	39	DN315 PN16 SR11 HDPE
TW24	1,846	DN315 PN16 SR11 HDPE
pAPW-VIB-03	2,120	DN315 PN16 SR11 HDPE
Las Burras Tank 1 subtotal	**4,005**	
Las Burras 2		
TW26	2,667	DN315 PN16 SR11 HDPE
TW27A	1,690	DN315 PN16 SR11 HDPE
pAPW-VIB-01	754	DN315 PN16 SR11 HDPE
pAPW-VIB-02	66	DN315 PN16 SR11 HDPE
Las Burras Tank 2 Subtotal	**4,777**	

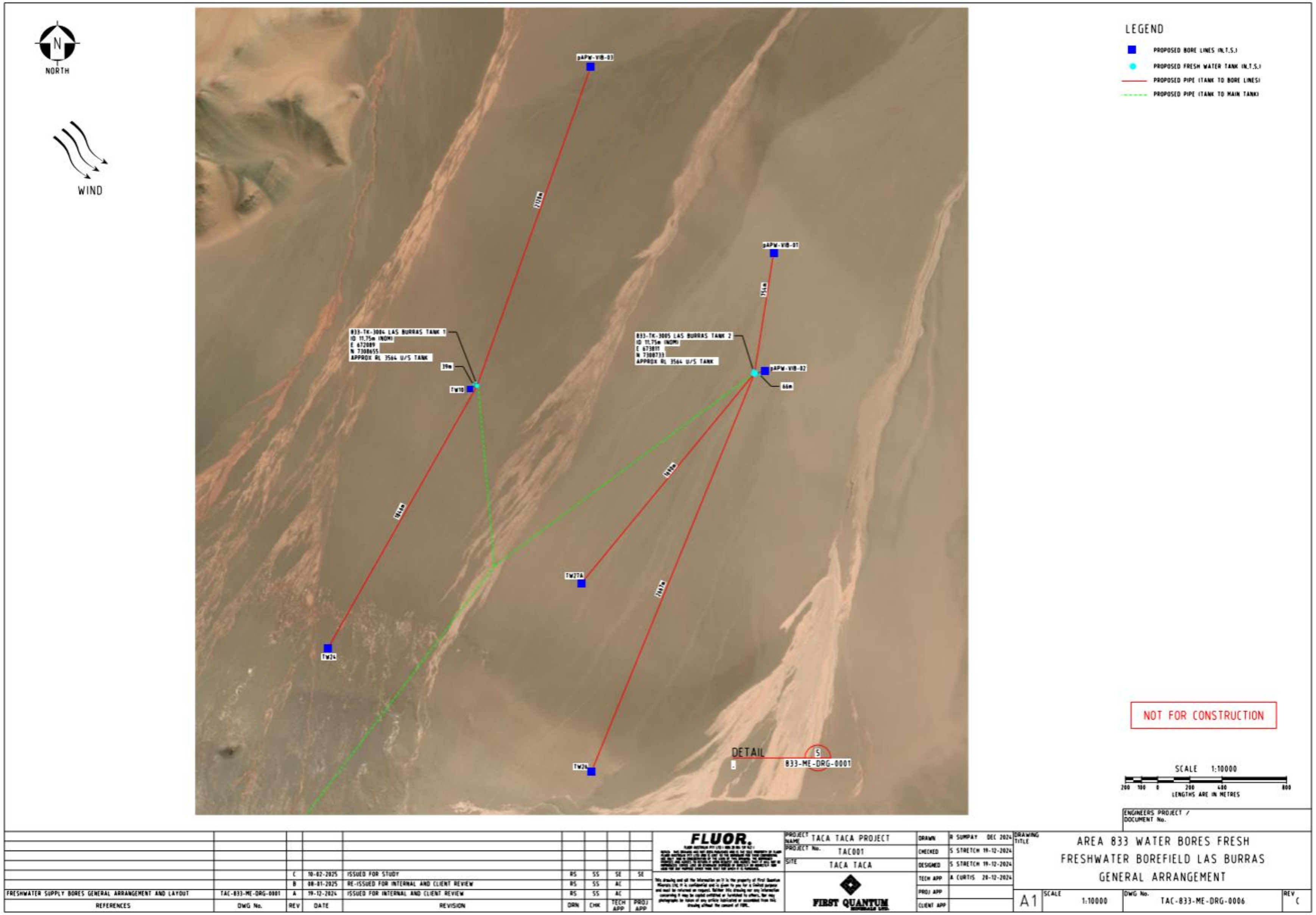
Figure 18-11 Profile of freshwater pipeline route from Valle de Las Burras to the plant (source: Fluor, 2024)



The lines from pAPW-VIB-01 and pAPW-VIB-03 will require throttling at tank discharge. The common tank sizing and the nominal flowrate of 684 m^3/h give a residence time of ten minutes.

The water would be gravitated from the in-field tanks to plant storage tank by a 51.1 km DN 630 PN12 SDR 13.6 HDPE pipeline.

Figure 18-12 Layout of Valle de las Burras borefield (source: Fluor, 2024)



18.3.5 Brine borefields and pipelines

A total of between 28 and 31 bores will be located on the Salar de Arizaro and Salar de Taca Taca, around:

- the periphery of the waste dump, for beneficial use of brine in the process, to support stable compaction of waste on the Salar de Arizaro, and eliminate seepage of brine to the pit and associated pressurisation of pit walls
- the entrance to the Salar de Taca Taca

Refer to Figure 18-13 and Figure 18-14 for the layout of the brine bores, in-field tanks and overland piping routes. With high level approximations, these bores have been estimated to be between 14 and 150 m in depth. The proposed bores are nominally 150 m in depth x 12” diameter. Flow from these bores is unknown, however flow rates for each have been assumed, between 4 and 23 L/s each.

Brine from bores 1 to 10 would report to a waste dump brine tank whilst brine from bores 11 to 18 would report to an Arizaro brine tank.

Figure 18-13 Waste dump and Salar de Arizaro brine bores (source: Fluor, 2024)

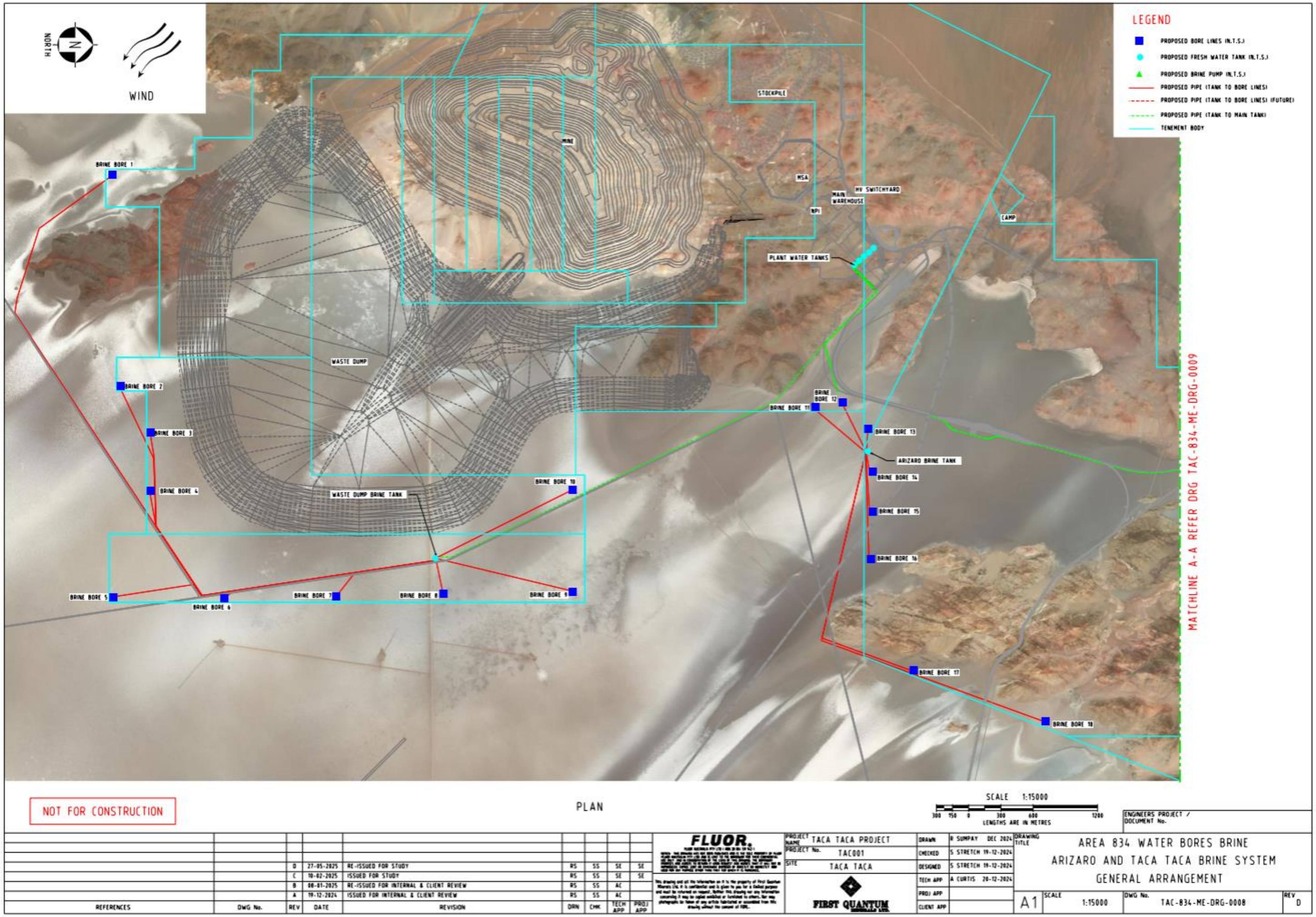
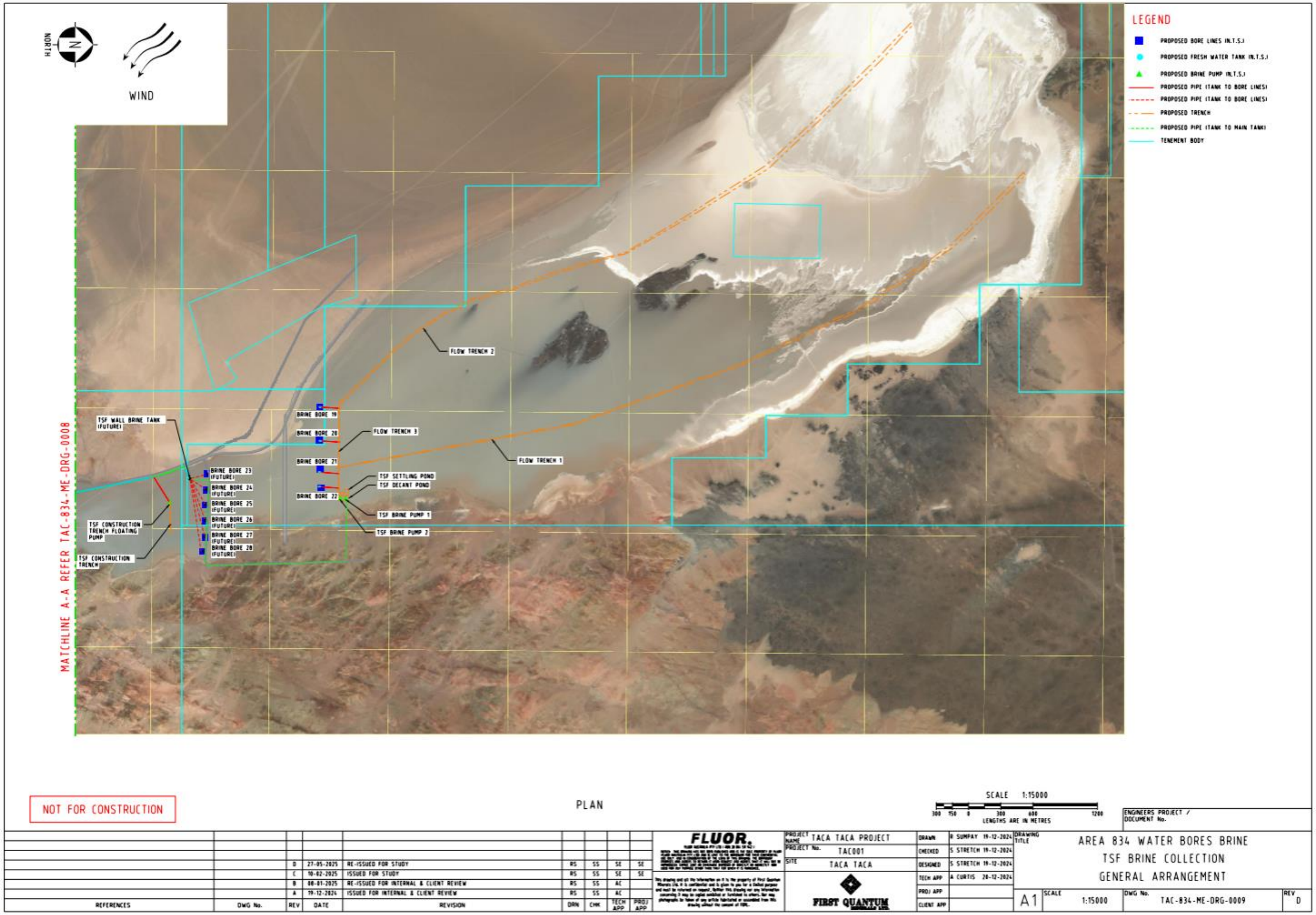


Figure 18-14 Tailings storage facility brine bores and trenches (source: Fluor, 2024)



Brine bores 1 to 10 which feed the waste dump brine tank will be connected with DN160 PN6.3 SDR 26 HDPE piping, the total length of which will be approximately 38.6 km. Brine bores 11 to 18 which feed the Arizaro brine tank will be connected with DN160 PN6.3 SDR26 HDPE piping over a total length of approximately 13.5 km. Each brine field tank (2) shall be 6.1 m diameter and 8 m high, constructed from bolted rolled sheeting, with an internal HDPE liner. Piping to and from the tanks shall be HDPE, to reduce scaling and corrosion. Transfer lines from the tank to the plant brine tank farm will be:

- from the waste dump brine tank; ~ 4.0 km of DN630 PN6.3 SDR 26 HDPE piping
- from the Arizaro brine tank; ~1.47 km of DN630 PN6.3 SDR 26 HDPE piping

Transfer pumps to the plant brine tank farm would be installed initially, with the following duties:

- from the waste dump brine tank; pump with a duty of ~ 525 m^3/h at 20 m total dynamic head
- from the Arizaro brine tank; pump with a duty of ~300 m^3/h at 20 m total dynamic head

Figure 18-14 also shows the location of the proposed brine abstraction trenches within the TSF footprint. The nature of the tails deposition determines that the excavated trenches on the Salar de Taca Taca will be temporary in nature. These longitudinal trenches will be serviced by a permanent lateral trench (running west to east) which routes the brine to a settling pond and a collection pond serviced by two pumps. The pumps will feed directly to the process plant.

The transfer lines from the tank to the plant brine tank farm will be:

- from TSF pond; ~ 8.67 km of DN500 PN16 SDR 11 HDPE piping

Initial transfer pumps to the plant brine tank farm will be installed with the following duties:

- from the TSF pond; two pumps with a duty of ~405 m^3/h each at 154 m total dynamic head

18.3.6 Overland pumping and pipelines

Table 18-7 lists the overland pipeline distances and pumping lift duties for the distant fresh water and brine supply sources.

Table 18-8 lists the internal piping distances within each for the freshwater borefields, with respect to the distance between each individual bore/well and collection tank.

Table 18-7 Fresh water and brine supply pipeline details

Source/Location	Overland pipeline distance (km)	Elev. Of borefield (mRL)	Max. elev. of route (mRL)	Elev. of tank (mRL)
Fresh Water Supply				**Tank**
Valle de Arizaro	40 + 41.1	3,652	3,693	3,489
Socompa	86	3,585	4,338	3,489
Valle de Chaschas	71	3,599	3,593	3,489
Vega de las Burras	2 x 53.6	3,565	3,550	3,489
Subtotal	**345.3**			
Brine Supply				**Tank**
Waste Dump Brine Tank	4.0	3,470	N/A	3,489
Arizaro Brine Tank	1.5	3,470	N/A	3,489
TSF Wall Brine Tank	5.84	3,470	N/A	3,489
TSF Pond	8.67	3,470	3,540	3,489
Subtotal	**20.1**			

Table 18-8 Freshwater borefields, internal piping distances

Bore/well	Distance to tank (m)
Fresh water supply	
Valle de Arizaro 1	
TW14	1,378
T22	2,987
pAPW-VdA-02	3,498
pAPW-VdA-03	1,400
pAPW-VdA-05	1,887
pAPW-VdA-08	3,828
Arizaro Tank 1 subtotal	**14,978**
Valle de Arizaro 2	
TW12	1,040
TW13	2,283
pAPW-VdA-01	2,989
pAPW-VdA-04	1,228
Arizaro Tank 2 subtotal	**7,540**
Socompa	
TW21	709
pAPW-SOC-04	3,148
TW20	535
pAPW-SOC-01	1,325
QDA. DEL AGUA	1,554
pAPW-SOC-03	697
pAPW-SOC-02	991
Socompa	697
TW19	50
subtotal	**9,706**
Valle de Chaschas	
TW22	1,890
TW23	195
pAPW-CHA-01	1,468
pAPW-Cha-02	1,796
pAPW-Cha-03	3,547
pAPW-Cha-04	1,615
pAPW-Cha-05	5,289
Subtotal	**15,800**
Vega de las Burras 1	
TW10	39
TW24	1,846
pAPW-VlB-03	2,120
Las Burras 1 Subtotal	**4,005**
Vega de las Burras 2	
TW26	2,667
TW27A	1,690
pAPW-VlB-01	754
pAPW-VlB-02	66
Las Burras 2 Subtotal	**4,777**

18.3.7 Borefield power supply

The power lines to be erected for the borefields will depend on the type of supply, i.e.:

- fresh water supply: 66 kV power line
- brine supply: 33 kV power line

Table 18-9 lists the borefield power line distances and transmission voltage. It is assumed that a multicore fibre optic connection will be provided alongside the high voltage lines for control and communications.

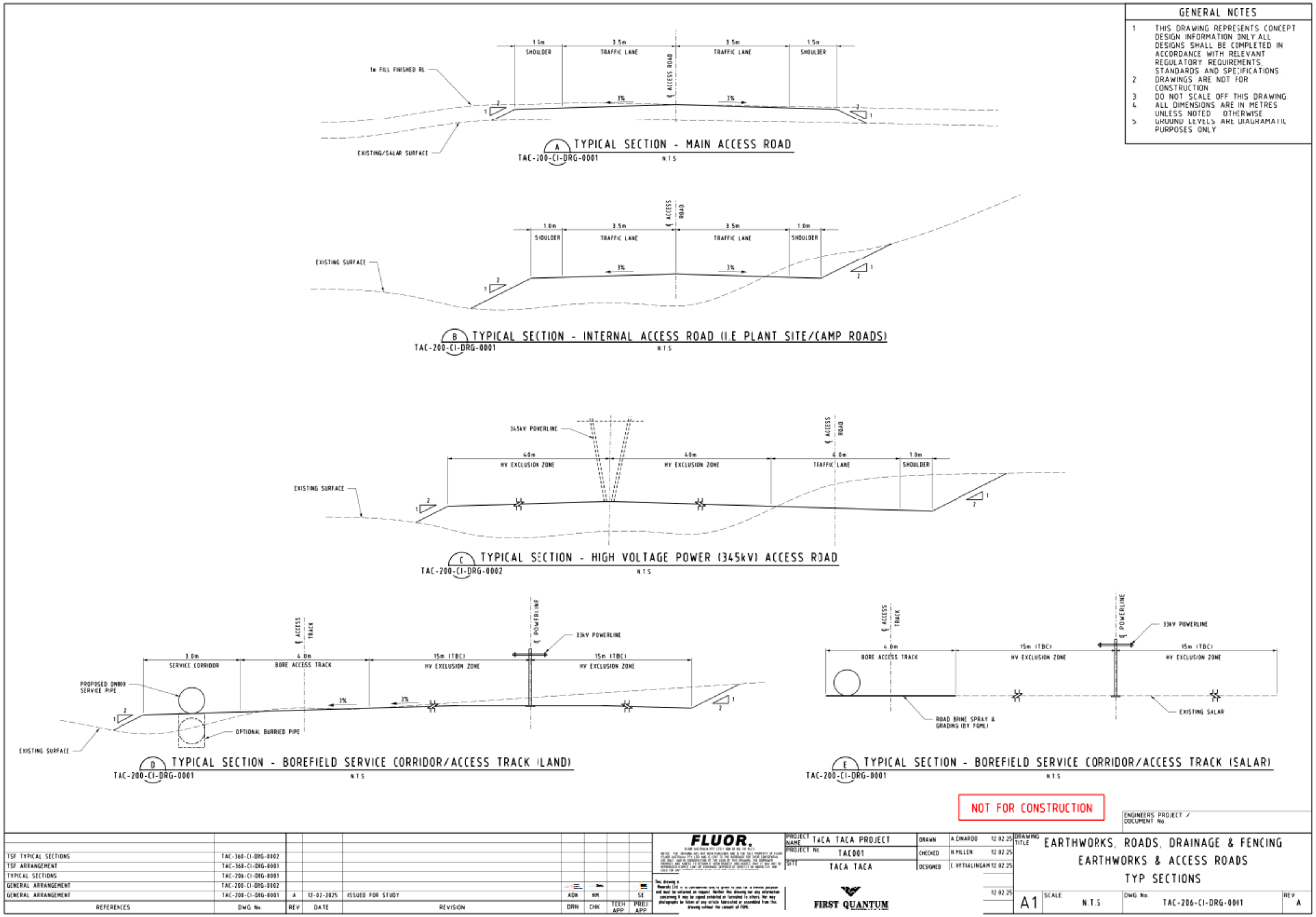
Table 18-9 Borefield power line distances and transmission

Borefield	Transmission length (km)	Transmission type
Fresh water supply		
Valle de Arizaro	38.4	66 kV twin bear
Socompa	65.2	67 kV twin bear
Valle de Chaschas	64.8	68 kV twin bear
Vega de las Burras	48.5	69 kV twin bear
	216.9	
Brine supply		
Pit depressurisation	3.1	33 kV twin bear
TSF seepgae bores	6.3	
Salar de Arizaro trenches	13	33 kV twin bear
Salar de Taca Taca trenches	15.8	
	38.2	

Figure 18-15 presents a construction diagram of the type of power line designed to supply the Project's integrated water supply system. It also indicates the relative position of the aqueducts and the service road, all within the same right-of-way. Up to three pipes will be supported on the ground at a distance of no less than 7 m from the centre of the right-of-way. The pipes can be positioned on either side of the right-of-way, typically occupying a width of no more than 3 m.

Internal power supply will be routed to each field tank by an overhead power line, with a local transformer and MCC providing power and control to the field tank transfer pumps, and out to each bore pump.

Figure 18-15 Cross-section through a borefield power line (and water pipeline) easement (source: Fluor, February 2025)



18.4 Road access

18.4.1 Existing access route

The Project site is readily accessible by road from the city of Salta, which is approximately 230 km to the east and is the nearest major population centre in Argentina (Figure 18-16). The road distance from Salta, along national route RN Nº51 via San Antonio de los Cobres, Cauchari, and then along provincial route RP Nº27 via Tolar Grande, totals approximately 400 km.

The National and Provincial Governments have embarked on an improvement project to seal the roads from Salta to Tolar Grande, with several contracts awarded and work in progress on improving accessibility to the La Puna region. There is an industry sponsored scheme, with annual contributions proportionate to mining investment size, which provides ongoing maintenance along the highways.

Transit times and reliability of access are expected to improve significantly before and during the commencement of the Project.

From the Chilean side, the Project site can be accessed via Routes 23 and 27, to Calama, San Pedro de Atacama and to Paso de Jama on the border, and then south via RN Nº51 to Cauchari and onwards to Tolar Grande and the Project site. Alternatively, Route 23 continues from San Pedro de Atacama and the Argentine border can be crossed at Paso de Sico before continuing to Cauchari, Tolar Grande and the Project site. The more direct road access to the site from the west is on lesser roads, either via route B-55 and Paso Socompa or via Baquedano, Peine, Socaire and then south before Paso de Sico.

18.4.2 Proposed route improvements

Proposed alternative route to site

Following on from the original alternatives analysis, the 2021 Technical Report described a shorter alternative access route involving a deviation from RN Nº27 at a point south of Cauchari (at km 28), passing over the Cerro Maćon in the north to later re-join RN Nº27. This would have bypassed Tolar Grande and is now not supported by the communities of both Pocitos and Tolar Grande.

This route will now purely be used as a construction access track alongside the 345 kV overhead power line to site, with grades and surfacing appropriate to a construction road only.

Proposed deviation

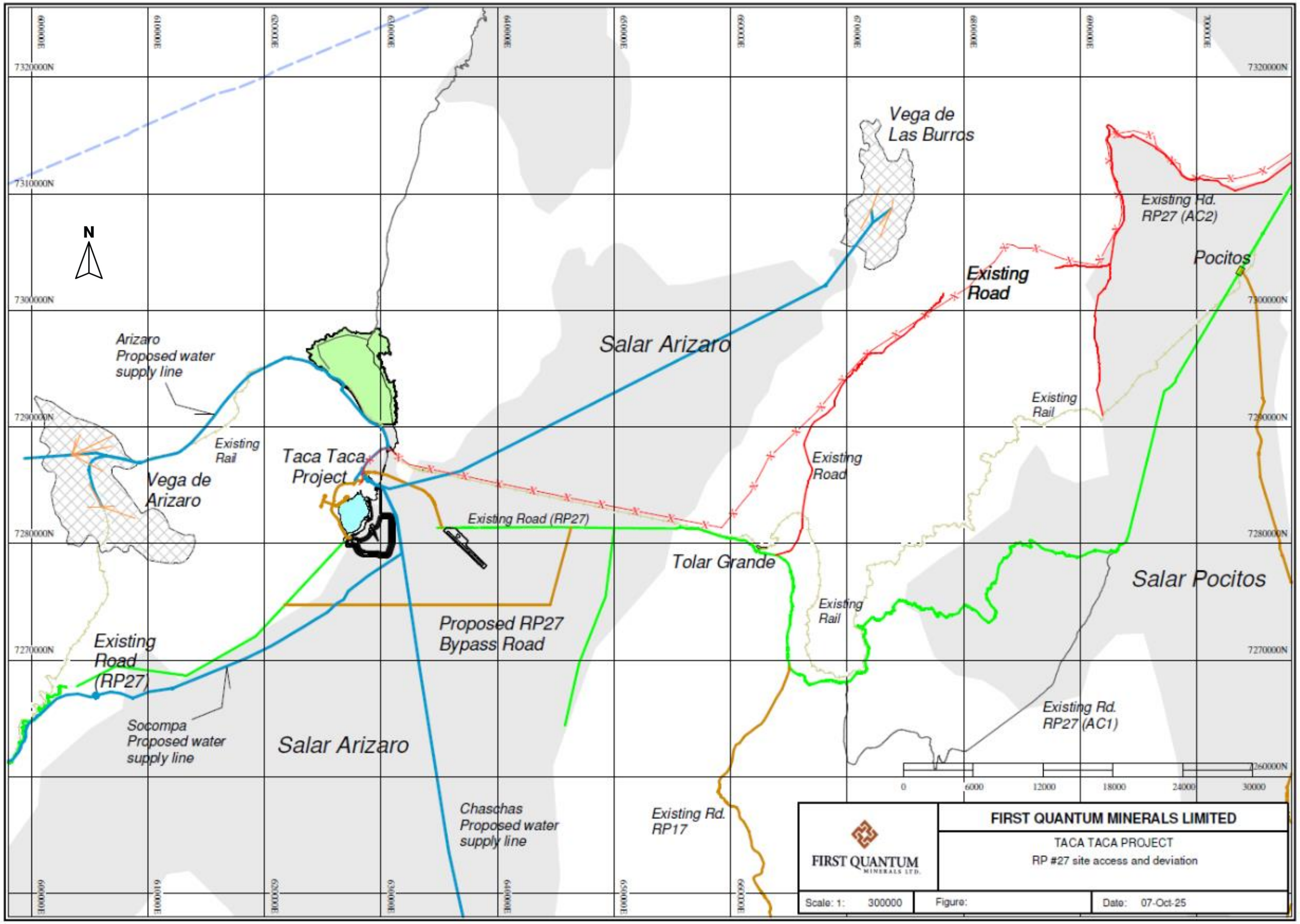
Existing public roads (RP Nº27) will provide access to the Project site (Figure 18-16). From Tolar Grande, RP Nº27 extends west across the Salar de Arizaro, and then skirts the western edge of the salar, heading south west. The existing road passes by the proposed airport and passes within 28 km of the proposed mining and processing operations thereby potentially compromising safety and security of the operations.

FQM has committed to a deviation of the road across the salar, maintaining public access beyond the plant, site. The preferred deviation requires an approximate 25 km length of new road construction on the eastern embankment of the Salar de Arizaro and then across the salar surface along the public road that extends from Tolar Grande. The proposed location for commencing the deviation is approximately 15 km east from Tolar Grande and will reconnect with RP Nº27 on the west side of the salar. Construction will consist of localised grading of the salar surface and brine treatment.

Figure 18-17 shows RP Nº27 after passing through Pocitos and continuing to Tolar Grande. The proposed deviation around the Project is also shown, as is the proposed eastern road for power line access.

Figure 18-16 Road access to the Project site

Figure 18-17 Route RP Nº27 access deviation



Supporting traffic on the route

On the main access route, a truck parking area will be developed on the Salar de Pocitos surface at the Pocitos town site, with no additional imported fill. It will be a private bypass for heavy vehicles, approximately 1 km in length with two-way access, and located north of the townsite to allow for trucks moving to and from Salta and Taca Taca to park as a rest site. Cut and borrow to fill will be undertaken for the formation of the access road.

Six parking bays, 5 m wide x 15 m long, will be provided at the bypass location, allowing for rest stops and walking access through to the Pocitos townsite. There will not be ablution or recreation room facilities. It is expected that ablutions will be available at the townsite.

Power line and TSF access roads

From RN Nº51 a new high voltage power line access road will be constructed approximately 20 km east of Olacapato Chico, which will traverse around the perimeter of the Salar de Pocitos and join the existing public access road at approximately 91 km from RN Nº51 and 15 km east of Tolar Grande. The high voltage access road will utilise the previous public access road and continue to site for a further 30 km across the salar. Construction will be a brine treatment to salar sections, with cut and borrow to fill for the construction access road elsewhere.

Public access will not be permitted on the high voltage access road. The formal access into site will be via the public road (airstrip access road) and this will be the only point of access, via a gate house.

18.5 Rail access

18.5.1 Existing access route

The Project site is located approximately 5 km from an existing narrow gauge (1 m) railway line between Salta and Antofagasta, line C14. After a time of dereliction in certain parts, this line has been refurbished to allow resumed services over its full length. The line has been refurbished to the extent that since 2016, reagents are being railed from Chilean ports to lithium mining companies operating in Salta Province.

On the Argentine side of the border (Figure 18-18), the railway is operated by state-owned *Belgrano Cargas y Logistica SA* (Belgrano). The line itself (and related infrastructure) is owned by *Administration de Infraestructuras Ferroviarias Sociedad del Estado* (ADIFSE). While there are numerous stations along the line between Salta and Taca Taca, only the stations at Salta, San Antonio de los Cobres and Tolar Grande are manned as track maintenance depots.

The C14 line continues past Taca Taca, through to the border at Socompa, and then via Ferronor and FCAB owned track to Mejillones ports (Figure 18-19).

It is proposed to upgrade the rail infrastructure between Taca Taca site up to Socompa and along the Ferronor section of the Chilean rail to accommodate 16 tonne axle loads. The proposed upgrade includes the installation of a new rail spur and signalling facility at the Taca Taca Project site to support copper concentrate export via Chile and some reagents deliveries to site. These works plus the provision and operation of the associated dedicated rolling stock will be provided as part of an operating contract.

The extent of project development associated will include the copper concentrate storage and rail loading system, fuel storage and locomotive refuelling, minor maintenance facilities and bulk earthworks to support the rail spur and sidings adjacent to the process plant facilities.

Figure 18-18 Rail access to the Project site, through Argentina

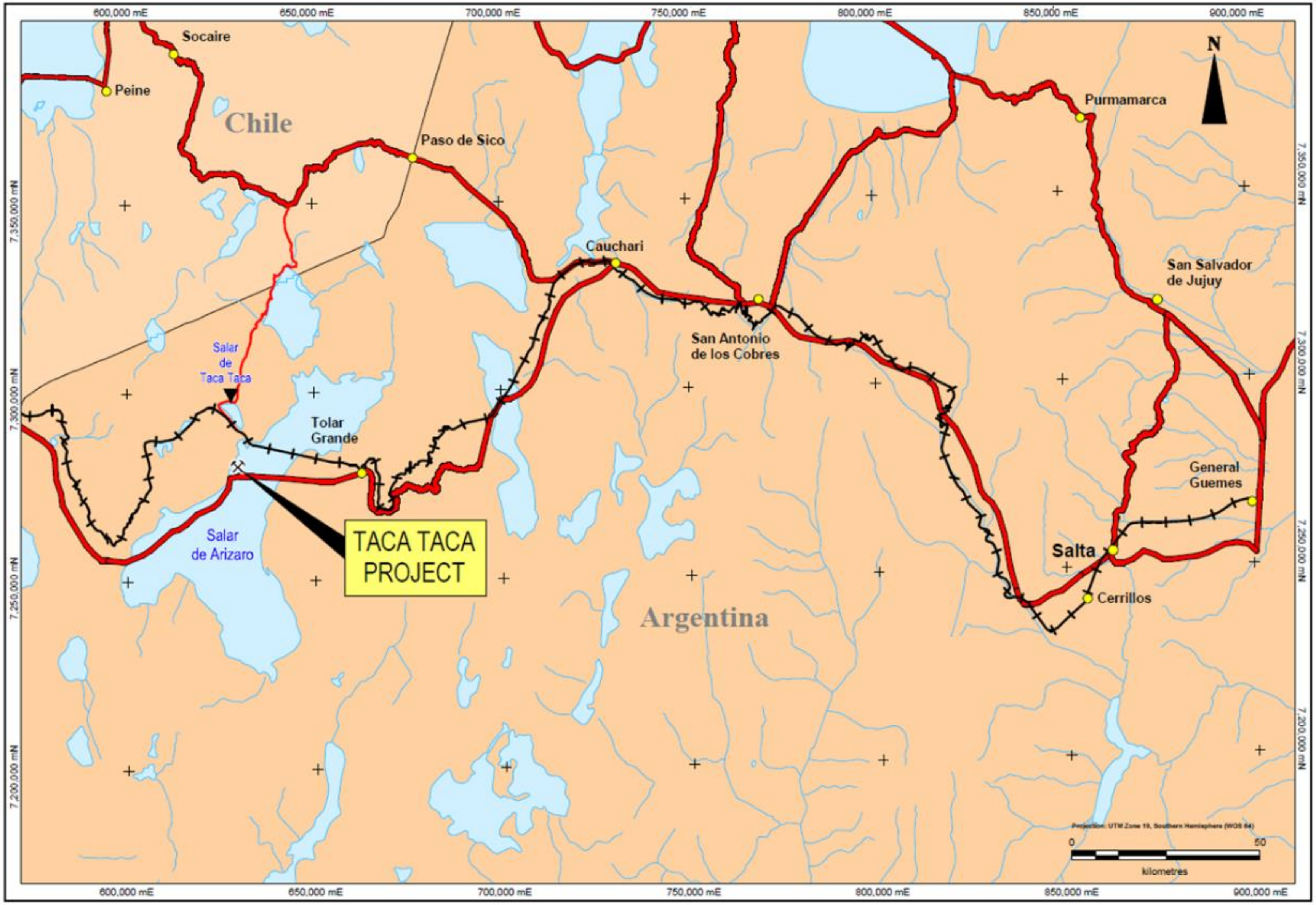
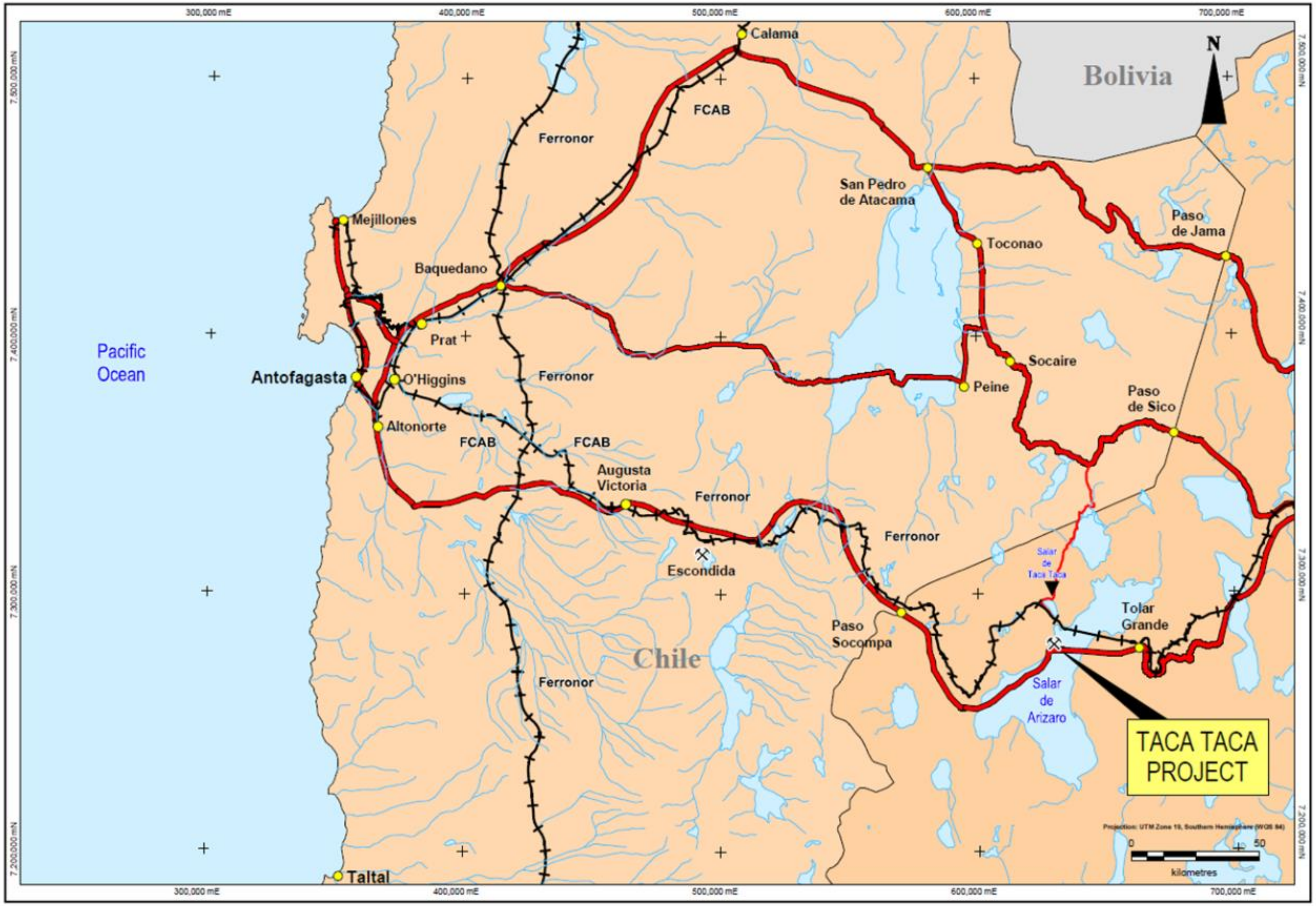


Figure 18-19 Rail access to the Project site, through Chile



18.5.2 Future rail line deviation

The existing rail alignment runs along a shallow embankment on the north western side of the proposed tailings storage facility on the Salar de Taca Taca.

Operations on the rail will be unhindered by tailings deposition until some six to eight years after the process plant operations commence.

At this time a deviation has been proposed further up the embankment which would support the continuing life of mine tailings disposal. The deviation in question is approximately 11 km in length. The addition of junctions allowing for the seamless concurrent installation of the future deviation with ongoing operation of the current route alignment with upgraded axle loading is proposed as part of the rail operations contract.

Figure 18-20 shows the tie-in arrangement for the existing railway line and the proposed site rail sidings. This figure also shows the future deviated railway line route.

18.6 Port facilities

Bulk copper concentrate will be railed to the Mejillones (Chile) port in rotainers. Molybdenum concentrate will be railed to the same port in bulk bags.

The Mejillones port environment benefits from a protected bay, with excellent availability. Historically, around ten days per year, generally between early January and late February, have disruptive tides. The area is serviced by multiple port facilities, several key copper mining operations, and numerous bulk concentrate carriers (Figure 18-21).

Opportunities exist to ship parcels of concentrate of between 10,000 to 11,000 DWT in size, consistent with the capacity of a single hold.

FQM has engaged in a commercial process with multiple existing and aspiring port operators. The proposed port facilities to service Taca Taca assume a new rotainer unloading facility, bulk concentrate storage, reclaim and bulk ship loading. Most prospective operators have ESIA preapproval and will develop the facilities concurrent with the Company’s development of the Taca Taca Project.

Port operations will be provided on an operational cost basis and has been negotiated for award upon Project investment approval.

Figure 18-20 Rail line tie-in and future TSF deviation

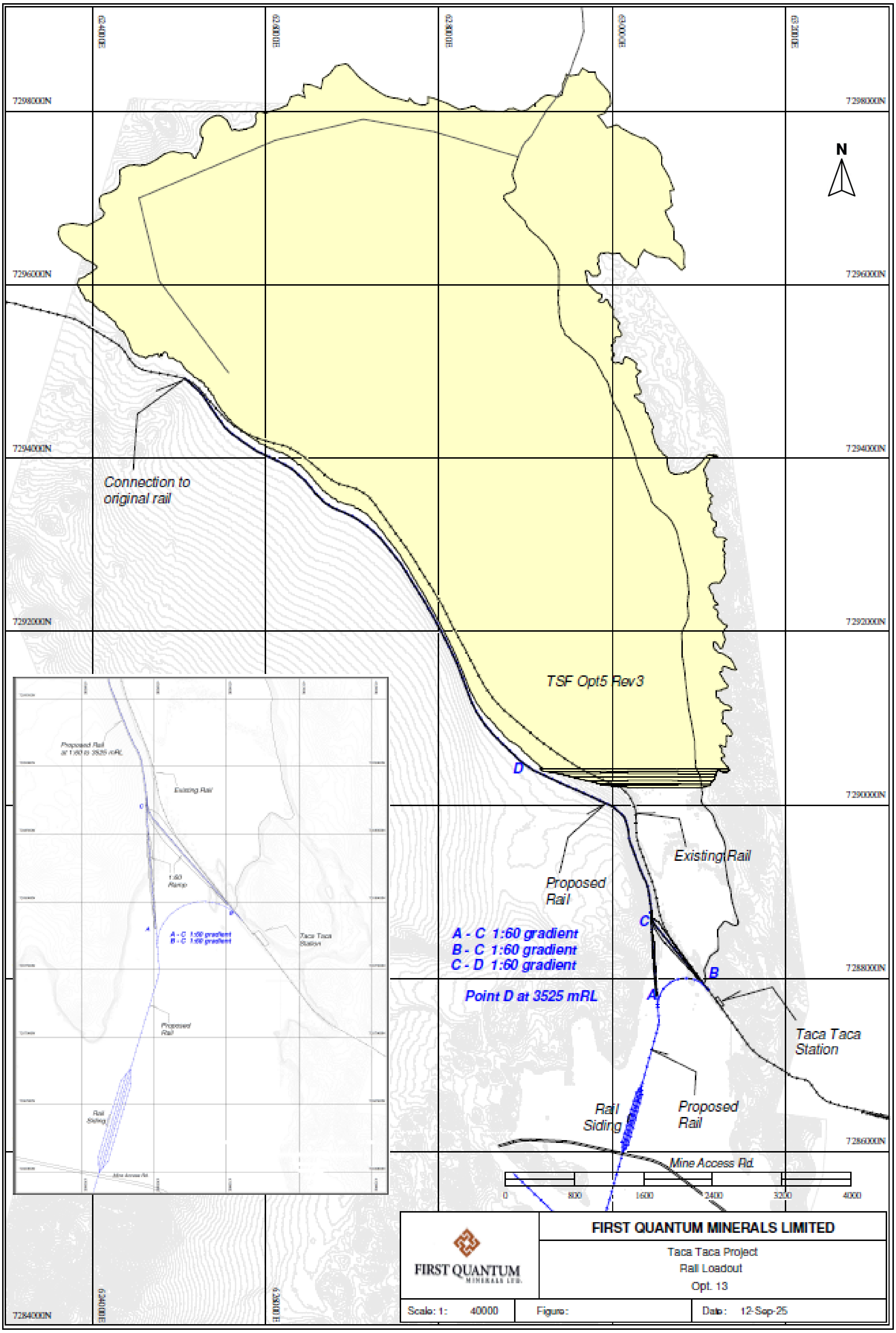
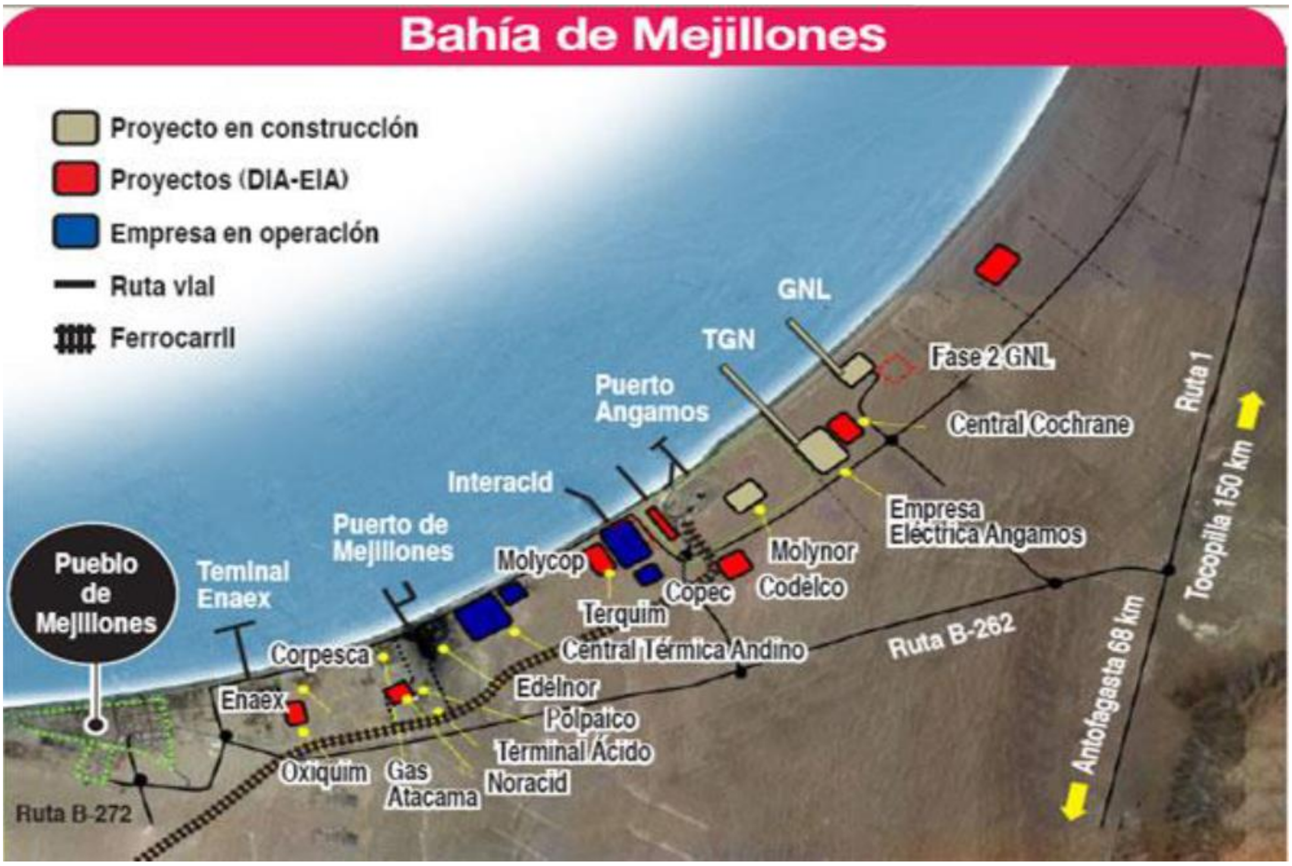


Figure 18-21 Ports on Mejillones Bay



18.7 Mining facilities

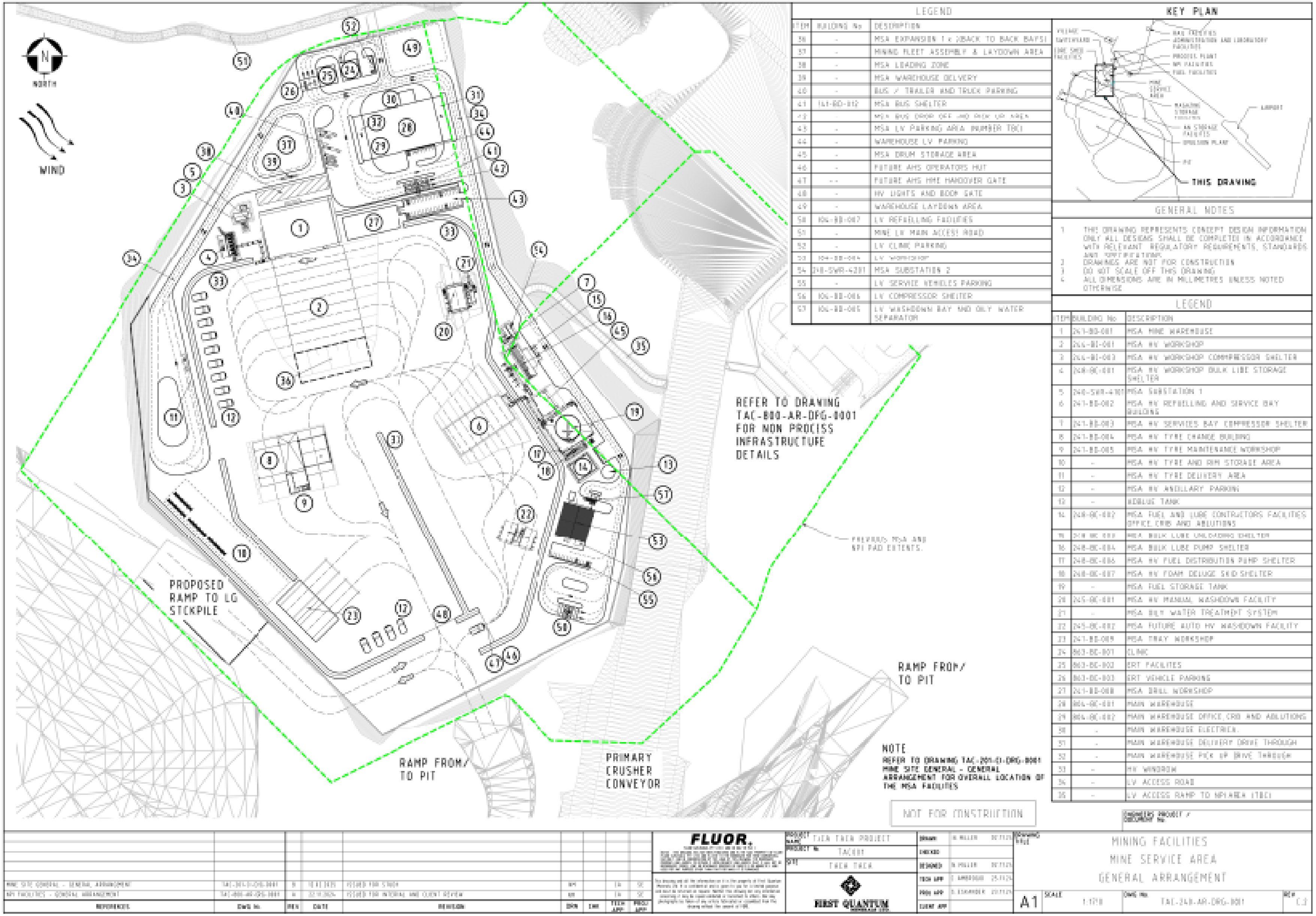
Information on mining facilities as presented in the 2021 Technical Report has been updated according to the following commentary, sourced from Fluor Australia Ltd (Fluor, February 2025A). Supporting information, specifically in relation to design parameters and requirements for non-process infrastructure (NPI), has also been sourced from Fluor Australia Ltd (Fluor, February 2025B).

18.7.1 Mine services area (MSA) and mine maintenance workshops

There will be comprehensive mine maintenance facilities and workshops erected as part of the Project. The facilities as shown in Figure 18-22, will consist of the following:

- A combined heavy vehicle (HV) refuelling and lubrication area (three bays for concurrent servicing) being the first facilities encountered upon entering the facility:
  - Whilst space for a future auto wash bay has been allowed for and given the absence of wet clay and scarcity of water, this is not proposed in the initial development.
  - This facility is similar to the FQM Cobre Panamá MSA facility; however, from experience at the FQM Sentinel operation it has been determined that a faster turn-around is achieved where simple lubrication functions are undertaken concurrent with refuelling.
- A manual wash station is provided for cleaning of the fleet prior to presentation at the maintenance workshops.

Figure 18-22 MSA layout plan (source: Fluor, February 2025)



- A separate drill workshop is allowed for maintenance of the drill and blast fleet, and is located at the rear of the facility:
  - The drill workshop will have three active bays: i.e., one dedicated boilermaker bay and two workshop bays.
  - There are an additional two bays which will serve as stores and racking, with a gas bottle storage shelter.
  - The workshop will have air-conditioned office facilities.
- The proposed HV maintenance workshop adjoins the MSA warehouse, and includes five lanes, and ten bays accommodating both tracked and wheeled mining fleet maintenance:
  - This workshop is similar in design to the FQM Cobre Panamá workshop, with additional bays large enough to support an ultra-class fleet if required.
  - The increased number of bays reflects the greater quantity of mobile fleet.
  - The workshop will be serviced by two 30 tonne overhead cranes, over each side of the workshop (sharing five bays), powered bays and lubrication filling stations, and with tool lockers in each bay.
  - The adjoining warehouse will include instrument workstations, electrical maintenance workstations, a mining hose bay, and local offices for mining maintenance planners.
  - A warehouse front desk arrangement allows for movement of parts directly from the warehouse along a central aisle enabling maintenance activities in each bay.
  - The proposed warehouse is located upwind of the workshop; the workshop orientation provides protection from the wind and dust entry into the facility.
  - Two of the bays nearest the wash bay will include cast-in rails to support track fleet maintenance.
  - The workshop will be serviced by a compressor shed for pneumatic services, sump pumps, oil and water separators.
- Tyre change building:
  - This facility provides one covered bay for tyre change-out and includes a nitrogen generator.
  - It is within forklift tramming distance from the tyre storage area, which will hold a one month supply of tyres, i.e., with additional storage held in Salta.
  - This facility is located on the departing side of the MSA, on the basis that tyres are infrequently changed over.
- The MSA tray workshop:
  - This workshop is also located on the departure side of the MSA area.
  - The workshop provides two bays for concurrent tray removal, repair and replacement.
  - The workshop will allow for either tray jacking, or otherwise a trestle mounted overhead crane which can remove and replace truck trays without the requirement for mobile crane usage.
  - Delineated entry and exit lanes with counterclockwise traffic movement will be incorporated.
  - Separation of HV and LV traffic will be maintained using 2 m high bunds and additionally where other LV traffic and pedestrian areas are located outside of these areas.

18.7.2 Emulsion and ammonium nitrate facility

The proposed emulsion and ammonium nitrate (AN) facility is located behind a gatehouse to control access into this restricted area. The facility is located approximately 2.5 km WNW of the MSA / NPI area along a road suitable for MPU's (mobile processing units, i.e. trucks used to blend and deliver bulk explosives directly into the mine). The location is shown in Figure 18-22 whilst the proposed arrangement of this facility is shown in Figure 18-23.

The NPI and processing areas are largely isolated from the emulsion and AN storage areas by the low grade stockpile area (Figure 18-23). The facility mirrors that designed and built for the FQM Cobre Panamá operation and consists of two distinct entities:

- the AN storage facility, consisting of a large open plan warehouse, accommodating discrete piles of AN bulk bags, allowing for 1 month of operational demand during peak mining production, and
- the emulsion production facility, which is separated by a safe operating distance from the AN storage and includes AN receival, emulsion production, a covered drive through shed, and emulsion storage and MPU loading facilities.

This facility also includes a QA room and an air-conditioned office.

18.7.3 Explosives magazine

The proposed explosives magazine is located approximately 1.3 km west of the mine stockpile and beyond the mining blast radius (Figure 18-1). The designed facility is comprised of two 40' standard sea containers, which are located and isolated from each other and the surrounding area, behind earthen berms of over 6 m height (Figure 18-23). Access to the magazine will be restricted by a gatehouse and boom gate with limited swipe card access.

18.7.4 Quarries and aggregate crushing (batch) plant

In the 2021 Technical Report it was mentioned that nearby quarries may be required for construction aggregate and also during operations (e.g., as a source for suitable blasthole stemming material).

Since then, several potential quarry sites have been identified, although the most convenient and suitable source of construction rock would be from the waste pre-strip mining volume. This granite host rock, composed primarily of quartz and feldspar, is leached of sulphides and is free of salt and gypsum.

Item 16 includes a commentary and table of scheduled mined waste that could be suitable for construction purposes.

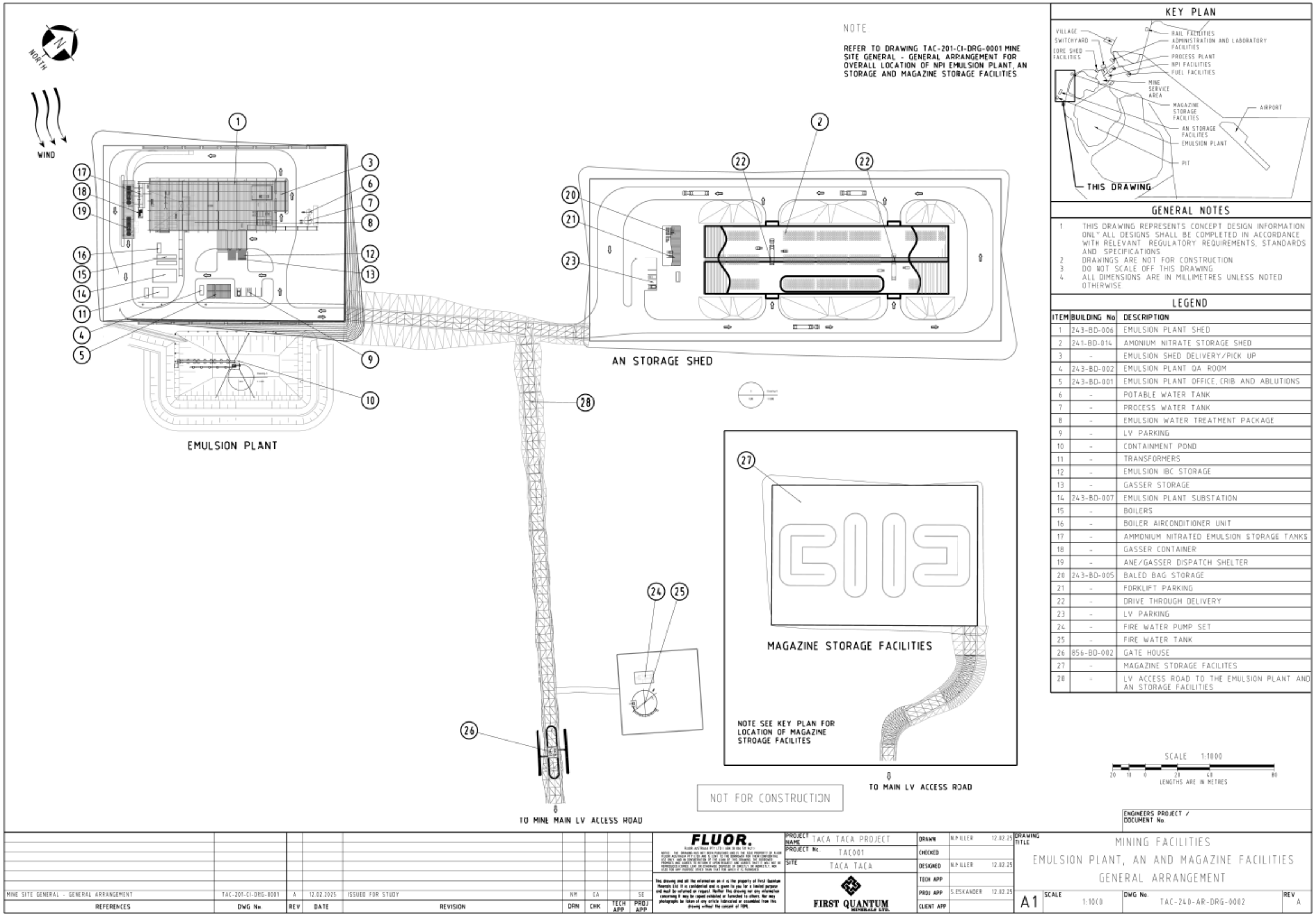
It is proposed to erect a permanent aggregate crushing plant which will:

- allow an initial contractor aggregate crushing plant to be demobilised
- allow for the long term production of stemming material, road base, wear coarse materials and future concrete aggregates

The permanent plant will be a typical two-stage crushing and screening facility, together with a fines washing circuit to remove ultra fines created during the crushing process, and relevant when making fine aggregate (sand) for concrete. Additionally, the aggregate production process will include a standalone potable water plant to remove out-of-specification iron trace elements which would otherwise affect long term concrete quality.

This crushing facility will have a peak capacity of 500 tph, when producing base coarse material, but is significantly less for stemming coarse aggregate and fine aggregate.

Figure 18-23 AN storage/emulsion plant and explosives magazine arrangement (source: Fluor, February 2025)



18.8 Administration and plant site facilities

The proposed location of the administration related service facilities is shown in Figure 18-1 whilst the general arrangement of these facilities is shown in Figure 18-24.

18.8.1 Administration and security

A gatehouse controlling access to the site will house offices and interview rooms and will be situated adjacent to a vehicle weighbridge.

The site administration and security building will be located in proximity to the process plant, MSA and NPI areas. This building will include a reception area, offices for the staff, meeting rooms, ablutions and other typical office facilities.

The administration building and surrounding facilities are to be located adjacent to the main LV access road, and on a platform that will be protected from the prevailing winds by the hill behind it. It will have an outlook over the processing facilities and over the rail spur line to the east. To foster collaboration, the majority of site based senior personnel will work from this location.

18.8.2 Emergency response team (ERT) and clinic facilities

These specific facilities will be located adjacent to the main site administration facilities, and central to the MSA, mine and process plant at the Project village.

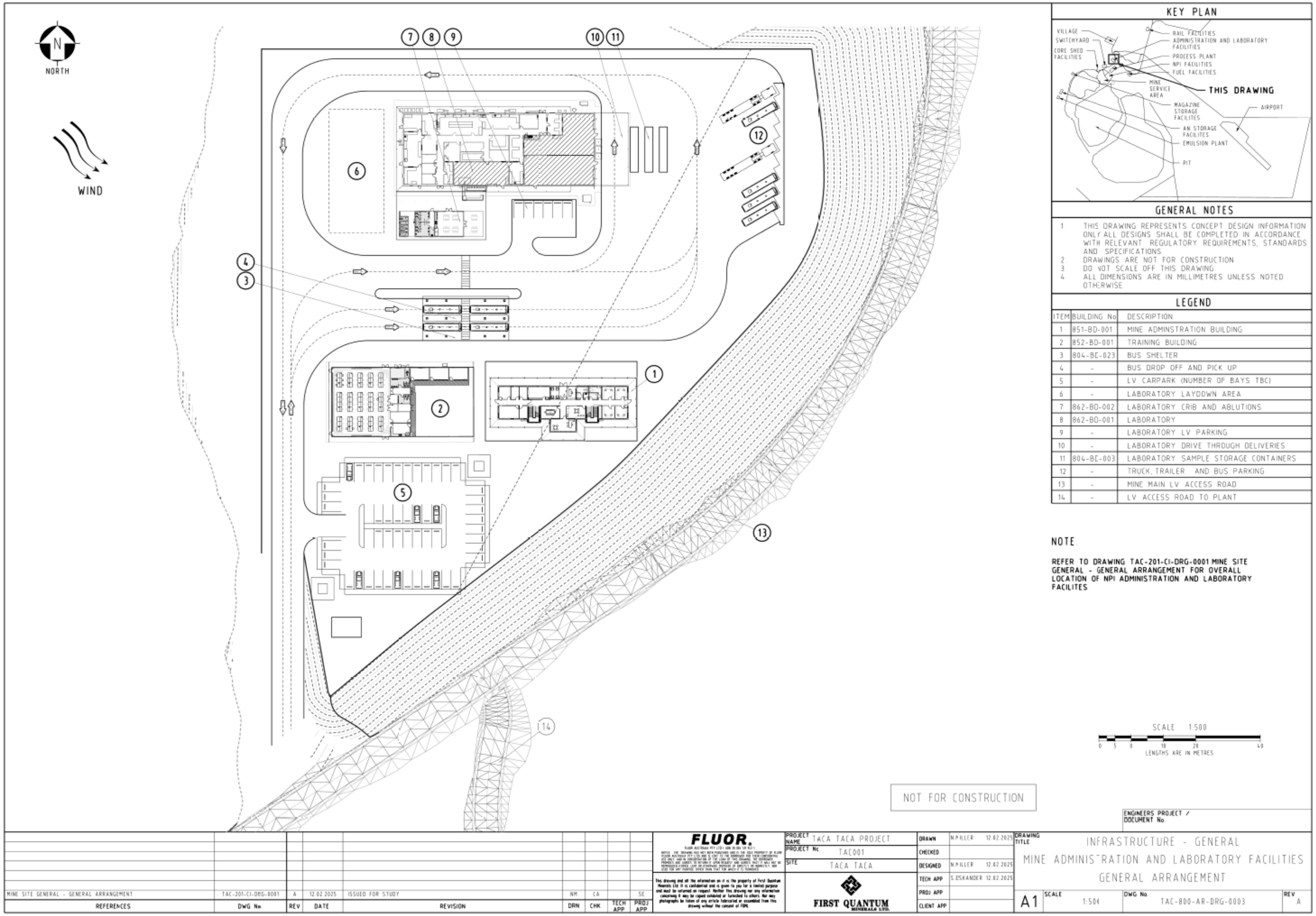
The clinic will be staffed by site based medical personnel including a doctor and nurses, with the ability to attend to minor workplace injuries, diagnose minor ailments and administer health care, to triage significant issues, and to make stable and sustain patient health until off-site medical evacuation can take place, as necessary. The ERT facilities will accommodate a dedicated team of emergency professionals with the capacity to provide access to and egress of personnel from common emergency situations including confined space rescue and working from height incidents. The facility will also include firefighting capability, with equipment and personnel to deal with fires and chemical spills. This facility will be located adjacent to the clinic.

18.8.3 Metallurgical laboratory

The metallurgical laboratory will have several rooms for mine grade control purposes, process plant sample analysis and process plant project evaluation.

The laboratory will include a sample preparation area with jaw crushers, rotary sample dividers, riffle splitters, pulverising mills, rod grinding mills, filters, drying ovens and a range of scales. There is provision for a flotation room containing float machines, flotation cells of various sizes, filters and bead mills. There will also be several wet chemistry areas, a digestion area, an ICP-OES room and a water analysis area, amongst other related facilities. An electrical room and a server room are included in the design.

Figure 18-24 Arrangement of administration and laboratory facilities (source: Fluor, February 2025)



18.8.4 Maintenance workshops

The following workshops and facilities will be located on-site:

- A light vehicle (LV) workshop (Figure 18-25), inclusive of:
  - eight servicing bays, an overhead gantry crane and appropriate hoists,
  - complete with concrete aprons, a LV washdown area, and with oily water separator,
  - stores for consumables, lubricant and oils storage,
  - LV refuelling, located near the HV fuel tank and LV workshop, and
  - air-conditioned offices
- Process plant maintenance facilities:
  - located adjacent to the covered coarse ore stockpile
  - serviced by access roads within proximity to the plant warehouse and immediately adjacent to the process plant
  - inclusive of two long, covered workshop spaces with partitioned walls, specifically:
    - A primary workshop (Figure 18-25) comprising:
    - a 100 tonne main and 15 tonne ancillary overhead crane,
    - a heavy fabrication area for chutes and platework items,
    - machine shops for shafts and small machining requirements,
    - a mechanical rebuild shop for portable equipment repairs,
    - an electrical and instrument shop,
    - a small parts and industrial gases storage area,
    - a workshop office in a mezzanine area of the facilities, and
    - a covered bus shelter for personnel drop off.
    - A secondary workshop (Figure 18-25) comprising:
    - a ceramics workshop,
    - painting and sandblasting facilities, and
    - rubber lining and splicing shops (including refrigerated containers for splice kit and solvent storage).

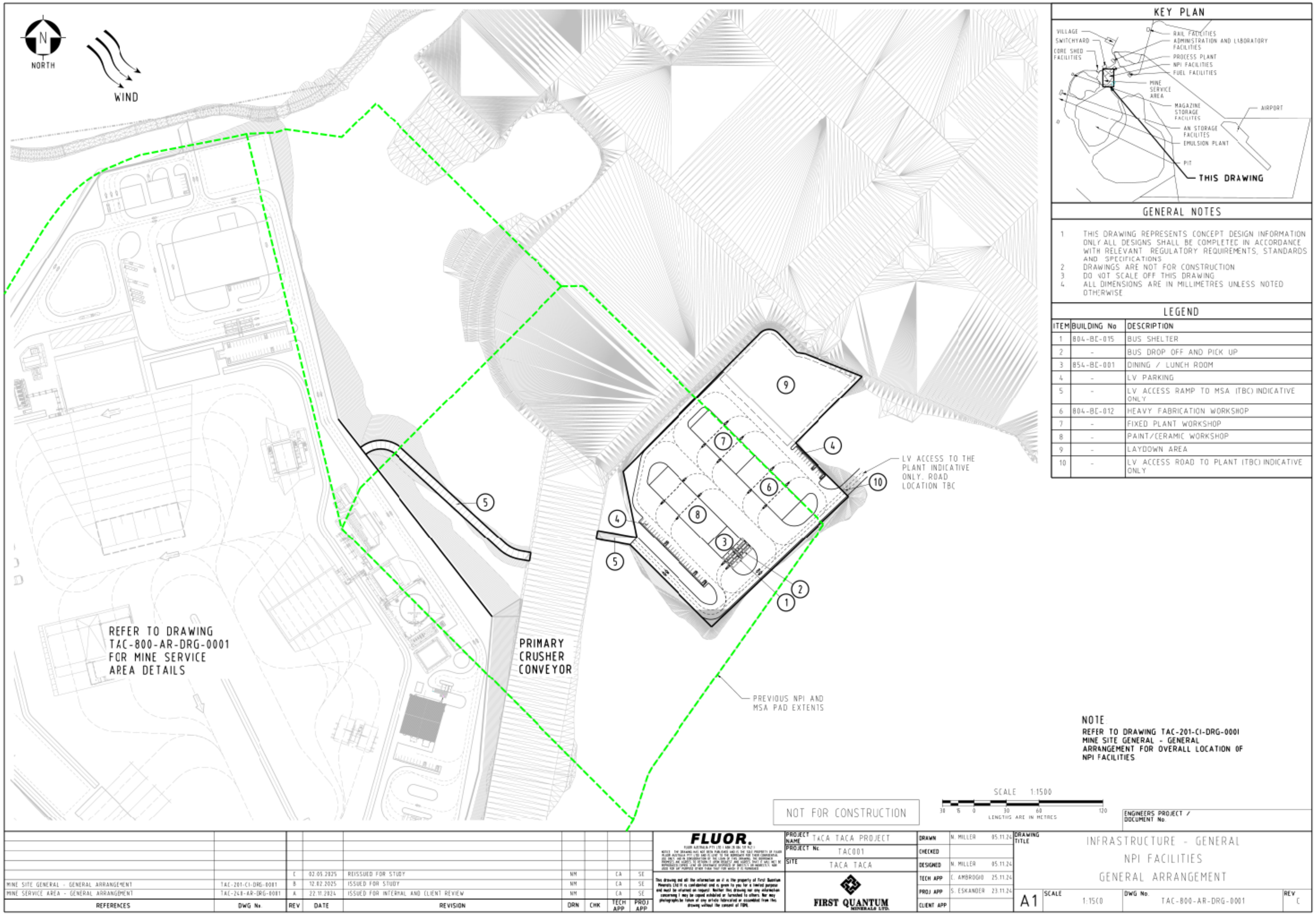
18.8.5 Warehouse

The proposed site warehouse is located on the main LV road, past the administration area, and ahead of the MSA facilities (Figure 18-1). It will be used as a forward store and will support the HV and LV maintenance activities as follows:

- receive and store the necessary consumables including lubricants, diesel fuel, equipment spares and tools
- control and issue spare parts and consumables for maintenance use on the plant and for HV and LV vehicles, and any other mine site facilities
- ensure ease of materials handling by providing concrete aprons for entries into the warehouse

The warehouse includes offices (one partitioned office and one open plan office with service counter), ablutions and tea preparation area, library, clean room, parts issue, tool store, ablutions, change room and lockers, parts staging area, electric charging area, and electrical repair. The warehouse will have air-conditioning to office areas only, along with two large roller doors to each end, and emergency exit doors, as required. The general arrangement for the site warehouse is shown in Figure 18-25.

Figure 18-25 NPI facilities: primary workshops and warehouse (source: Fluor, February 2025)



The primary warehouse and laydown yard for the Project will be located in Salta.

18.8.6 Process water storage

Raw water will be sourced primarily from remote water supply sources. The raw water will report to various enclosed tanks in the process plant area and then be partly pumped to the water treatment plant for potable water production and partly to reagent mixing. Most of the stored water will be used as top up water for the process water system.

The process water storage tanks will also receive return water flows from the tailings thickener overflow, the copper concentrate filter, and as minor contributions from sanitised waste water return.

The layout of the processing facilities, and specifically the process water storage area, is shown in figures included in Item 17.

18.9 Tailings storage facility

FQM engaged SRK Consulting (July, 2022 and 2025) to evaluate the technical feasibility of a tailings storage facility (TSF) for the Project, including a review of all prior studies and reports.

18.9.1 Basis of design

The proposed TSF site is approximately 4 km from the plant site, located on the Salar de Taca Taca. SRK described a design with a 22 m high embankment and a storage capacity of approximately 1,850 Mt of tailings or 1,240 Mm^3, sufficient for the life of mine. Subsequent to the SRK work, an increase in Mineral Reserves indicates the LOM tailings generated will be 1,960Mt. The SRK designs will need to be updated to reflect the additional capacity required.

The design of the TSF addresses the underlying structural stability, whilst a dam break analysis (assuming water rather than slurry) indicates a limited flow of tails in the event of an embankment failure.

18.9.2 Design criteria

The TSF is designed to meet the Australian National Committee on Large Dams (ANCOLD) guidelines, with considerations for seismicity, dam classification, and risk category. The facility is engineered for the life of mine, with a maximum dam height of 22 m. As noted above, a recently identified increase in Mineral Reserves will require an amendment to the current designs.

Design criteria is provided in Table 18-10. The raise concept in this table refers incorrectly to “centreline”, whereas it will be a downstream raised embankment.

Future detailed TSF engineering design would conform to the GISTM (Global Industry Standard on Tailings Management).

18.9.3 Seismicity

The site is in a region of moderate seismic hazard. Design parameters are based on local and international standards, with a focus on ensuring dam stability under seismic loading.

Table 18-10 TSF design criteria

Design input	Criteria
Consequence category	
ANCOLD Dam failure consequence category	Significant
Design storm	
Allocated storage for wet season	1:10 AEP of wet season theoretical runoff
Allocated storage for extreme storm	1:100, 72h-rainfall
Spillway capacity	1:1,000 AEP + 1:10 AEP wind
Minimum total freeboard	1.0 m
Minimum operative freeboard	0.5 m
Contingency freeboard	0.3 m
Dam stability	
Long term undrained (potential loss of containment)	1.5
Short term undrained (no potential loss of containment)	1.3
Post-seismic, residual	1.0 – 1. 2
Seismicity	
Operating basis earthquake (OBE)	1.475 AEP
Safety evaluation earthquake (SEE)	50th fractile; 1 : 1,000 AEP
Post closure earthquake (PCE)	MCE or 1:10,000 AEP; (85th fractile)
ANCOLD OBE	PGA = 0.33 g; (1 : 475 AEP)
ANCOLD SEE	PGA = 0.59 g; (1 : 2,475 AEP)
ANCOLD (post=closure) MCE (Equivalent to 1:10,000 AEP)	PGA = 0.123 g; (1 : 10,000 AEP, 85th
Climate	
Annual average rainfall	48 mm
Annual average pan evaporation	3,740 mm
Rainy season	Dec- Mar
Average wind speed (annual)	18.4 m/s
Average maximum speed	74.6 m/s
Tailings production and deposition	
Annual design production rate	40 Mt/a (years 1 – 7); 60 Mt/a (years 8 -33)
Life of Mine (LoM)	33 years
Average tailings stored dry density	1.5 t/m^3
Tailings beach slope	1.50%
Tailings slurry parameters	
Target slurry solids concentration (% w/w)	67%
Specific gravity	2.7 t/m^3
Dam geometry	
Final height and level	22 m, RL 3,498 m
Dam design	
Crest width	10 m
Crest slope	2%
US batter	1V:4H
DS batter	1V:4H
Safety bund	0.5 m
Raise concept	Centreline
Protection berm design	
Crest width	2 m
Height	1 m
US batter	1V:1H
DS batter	1V:1H

18.9.4 Geotechnical investigations

SRK (July, 2022) carried out a geotechnical investigation campaign involving diamond drilling between December 2021 and January 2022 to define the geotechnical conditions of the tailings dam foundation and impoundment area. Four investigation holes were drilled in the locations shown in Figure 18-26, displaying the varying lithological horizons that are shown in Figure 18-27. The main conclusions from the SRK investigation were:

- five geotechnical units were identified within the Salar de Taca Taca, comprising:
  - poorly consolidated sand with variable contents of halite, clay and, to lesser extent, volcanic ash
  - halite with different degrees of compacity (i.e. very compact halite, compact halite, porous halite and unconsolidated halite)
  - highly consolidated clay, halite-containing clay and carbonate-containing clay
  - moderately consolidated sandstone cemented with halite
  - granite belonging to the Taca Taca formation, which constitutes the stratigraphic basement of the salar
- bedrock was encountered at 76 m below the ground surface at the future embankment location
- the phreatic level was near the natural ground surface, at between 0.5 m and 1.0 m depth

Figure 18-26 Location of geotechnical investigation holes, Salar de Taca Taca (source: SRK, July 2022)

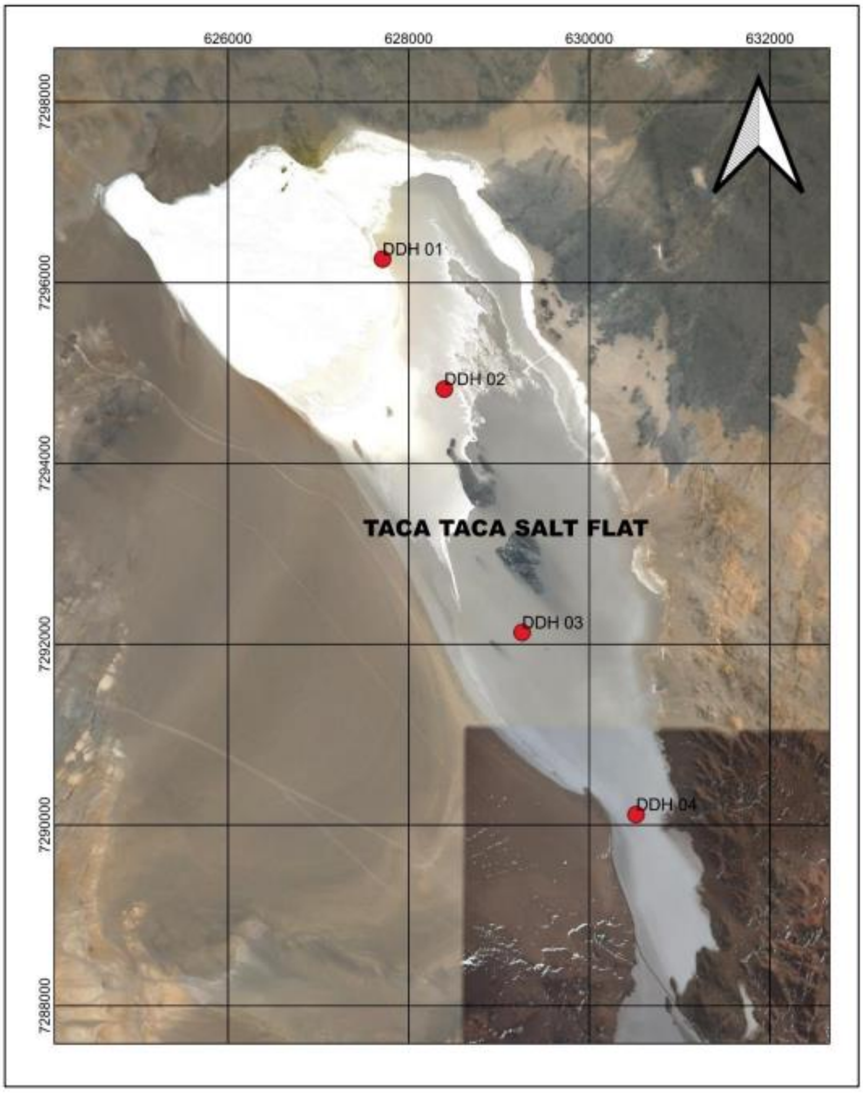
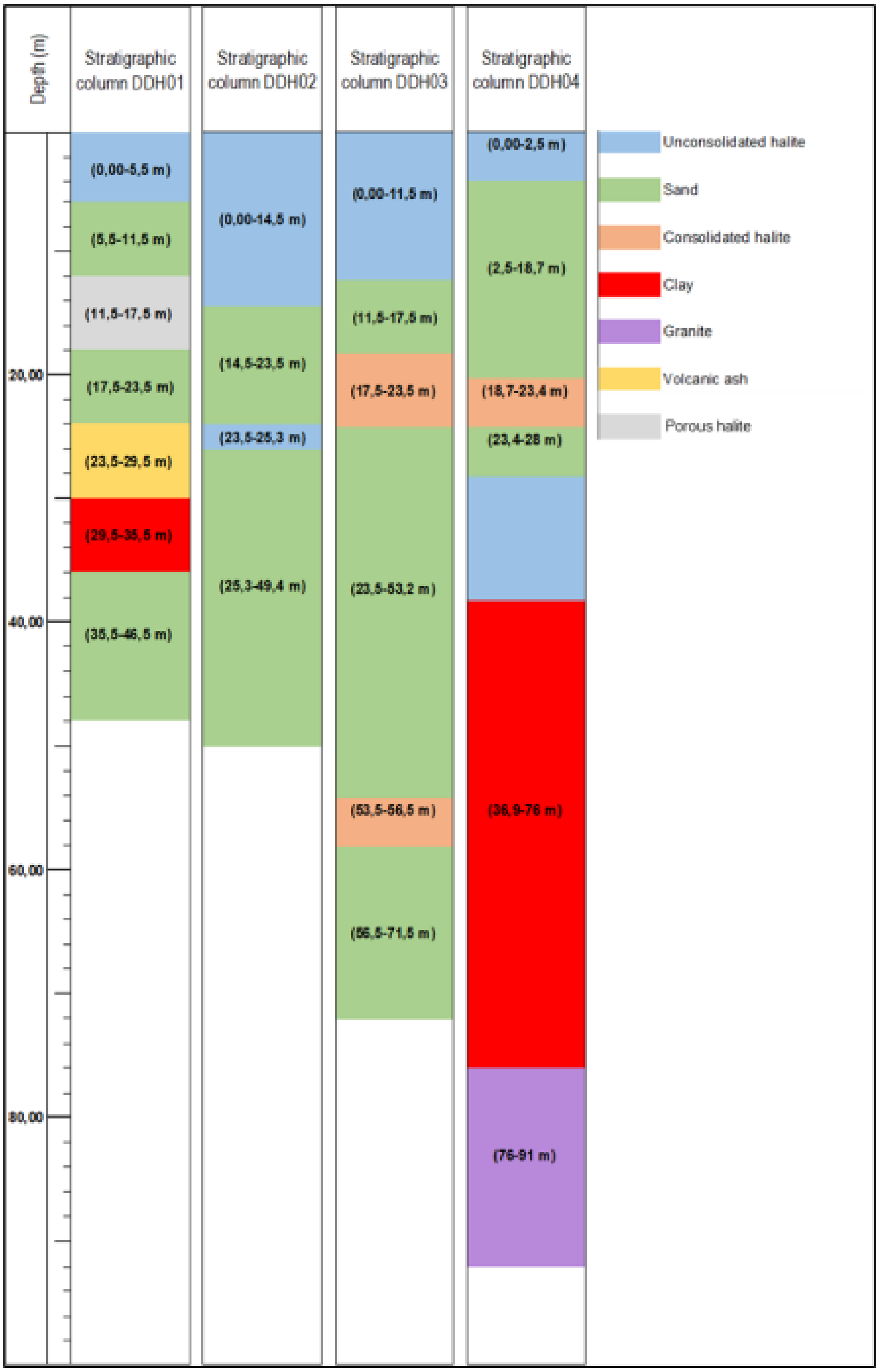


Figure 18-27 Lithological horizons in four geotechnical drill holes, Salar de Taca Taca (source: SRK, July 2022)



18.9.5 TSF configuration, stability and settlement

The Salar de Taca Taca is characterised by a basin filled with a complex mix of sedimentary and evaporitic deposits, resting atop granite bedrock. This geological setting introduces challenges for tailings storage, most notably due to the potential for halite (salt) dissolution and subsurface caving. Such processes could threaten the integrity of any structure built on them, so the design of the tailings storage facility (TSF) must ensure long-term stability, even in the event of ground collapse or significant deformation beneath the dam. Preventing liquefaction and uncontrolled flow of tailings is a central objective of the design.

To address these challenges, the TSF will employ a thickened tailings deposition strategy, creating a sloping beach with gradients of between 1% and 3%. Testwork undertaken by Tailpro (Tailpro, June 2025) provided estimates from two independent consultants of the beach angles to be achieved at various slurry densities.

Dr McPhail of Water, Waste and Land Australia suggested that at 65% solids the tailings would be sufficiently fluid that a beach slope of only 0.15% would be achieved (one of the consultants advising Tailpro). Prof. Andy Fourie of the University of Western Australia (the other consultant advising Tailpro) reported a beach slope of 0.97% to 1.88%, depending on the number of spigots used. SRK (July, 2022) used a figure of 1.5% in their study.

The facility is designed with perimeter embankments, internal drainage, and decant structures to manage water efficiently and maintain the stability of the impoundment.

For the construction of the embankment, rockfill sourced from mine waste that is neither acid-generating nor capable of leaching metals should be used. On the upstream slope of the dam, a transitional material zone, potentially incorporating geotextile fabric, should be installed to prevent the migration of tailings into the dam body.

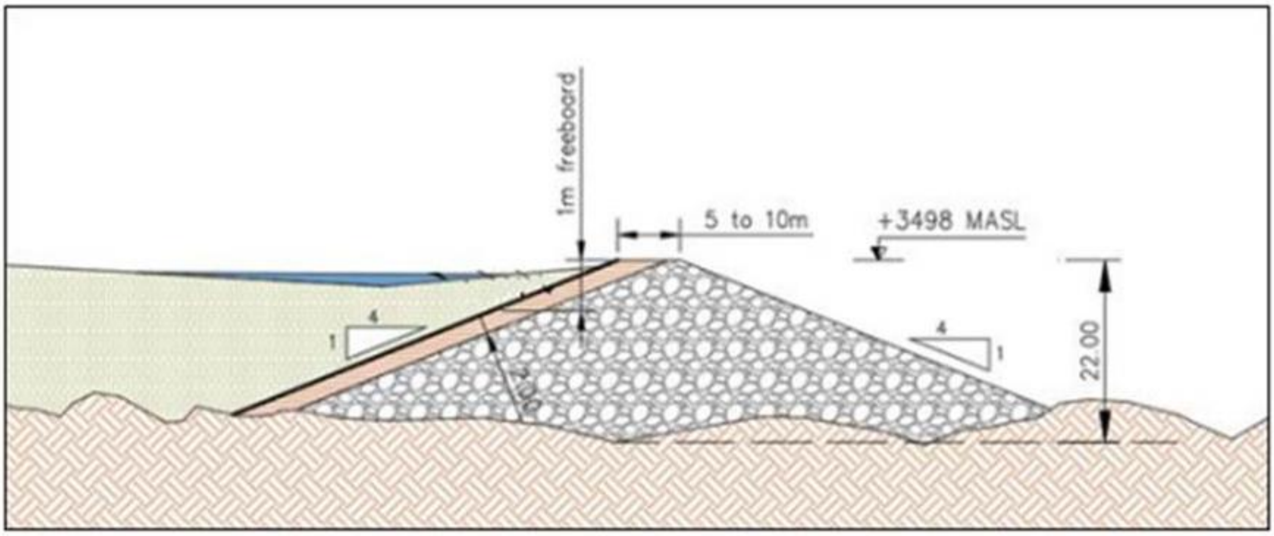
Comprehensive stability analyses have been performed for a range of loading conditions, including static, seismic, and rapid drawdown scenarios. In all cases, the calculated factors of safety meet or exceed international guidelines, providing confidence in the design's resilience. However, further work is recommended to refine the design and confirm key parameters. To this end, SRK suggested a targeted drilling campaign, complemented by geophysical studies along the proposed embankment alignment, to support the next phase of engineering.

Potential settlement of the embankment due to creep and halite dissolution has also been evaluated. Using proprietary software and following the ANCOLD guidelines, the assessment considered deep-seated failure mechanisms under various scenarios. The results, as detailed in the SRK report, indicate that in no scenario does the failure surface pass through the tailings material, and the stability is relatively insensitive to the cohesive strength values adopted for the halite layers. Analysis of long-term settlement due to creep showed that such effects are negligible, primarily because the volume of halite strata is smaller than anticipated and the deformation rate is low.

To further reduce the risk of caving caused by halite dissolution, the design includes the removal of unsuitable materials beneath the dam footprint. Additionally, the operational plan calls for the use of brine as dilution water for the tailings, which will help to minimise the risk of further halite dissolution. The embankment Is designed to be large enough that it will remain stable even in the absence of dewatering activities.

Figure 18-28 is a conceptual cross-section through the TSF embankment design. Excluding an upstream layer, the total volume of the TSF embankment is 3.95 Mlcm. The construction timeframe would commence in Year 7 with an initial 4 m high starter embankment.

Figure 18-28 Conceptual cross-section through the TSF embankment (source: SRK, July 2022)



18.9.6 Dam break assessment

A dam break model was developed to evaluate the potential inundation extent from a hypothetical TSF failure. The Study utilised Shuttle Radar Topography Mission (SRTM) data to model topography. For initial modelling simplicity and due to uncertainties in tailings rheology, water was substituted as the stored material, which may overestimate inundation extents when compared with actual tailings flow, while under-estimating the damage caused by a more viscous material. This analysis will be repeated once testwork results for slurry rheology become available.

The analysis assumed a stored volume of 225 Mm^3 and simulated an overtopping failure mechanism using the U.S. Army Corps of Engineers' HEC-RAS 6.6 software. Results indicated that whilst Tolar Grande village lies outside the predicted inundation zone, critical Project infrastructure including the rail line and adjacent station, could be affected. This preliminary assessment provides a conservative baseline for understanding risks, with refinements anticipated in future studies incorporating detailed tailings behaviour and site-specific topography.

18.9.7 Material take-offs and cost estimate

Material quantities and costs were estimated for bulk earthworks, embankment construction, drainage systems, and associated infrastructure. These have been incorporated into the overall capital cost estimate.

18.9.8 TSF operations

The tailings deposition strategy has been developed to minimise impacts on environmentally sensitive areas, specifically the Plumas Verdes and Vega Salitrosa regions, as well as adjacent rail infrastructure. For a minimum period of eight years following initial processing operations, these areas will remain unaffected by deposition activities. To provide additional protection, a nominal two-metre-high berm will be constructed around the designated vegetation zones.

The initial phase of the deposition plan spans ten years, comprising three years allocated for construction activities and a subsequent seven years of tailings deposition at a rate of 40 Mtpa. It is anticipated that, after this period, the sensitive areas and rail infrastructure will have been relocated, thereby removing the geometric constraints on the northern and northeastern margins of the TSF and allowing for continued deposition to achieve the required LOM storage volume.

Operational parameters for the TSF are detailed in Table 18-10. Tailings will undergo dewatering within the process plant to maximise the recovery of fresh water. Following dewatering, the tailings will be diluted with brine to achieve a pumpable rheology suitable for hydraulic transport.

The tailings slurry will be conveyed from the process plant to the TSF via high-density polyethylene (HDPE) pipelines. The deposition system is designed with five discharge lines, each equipped with spigots spaced at 40 m intervals to ensure controlled distribution of tailings. The discharge sequence is as follows:

1. west discharge line, located at 3,509 mRL
2. southeast discharge line, located at 3,495 mRL
3. southwest discharge line, located at 3,492 mRL
4. west (long) discharge line, located at 3,546 mRL
5. west (short) discharge line, located at 3,555.5 mRL

This approach enables efficient management of tailings placement while maintaining the integrity of protected areas during the initial operational period.

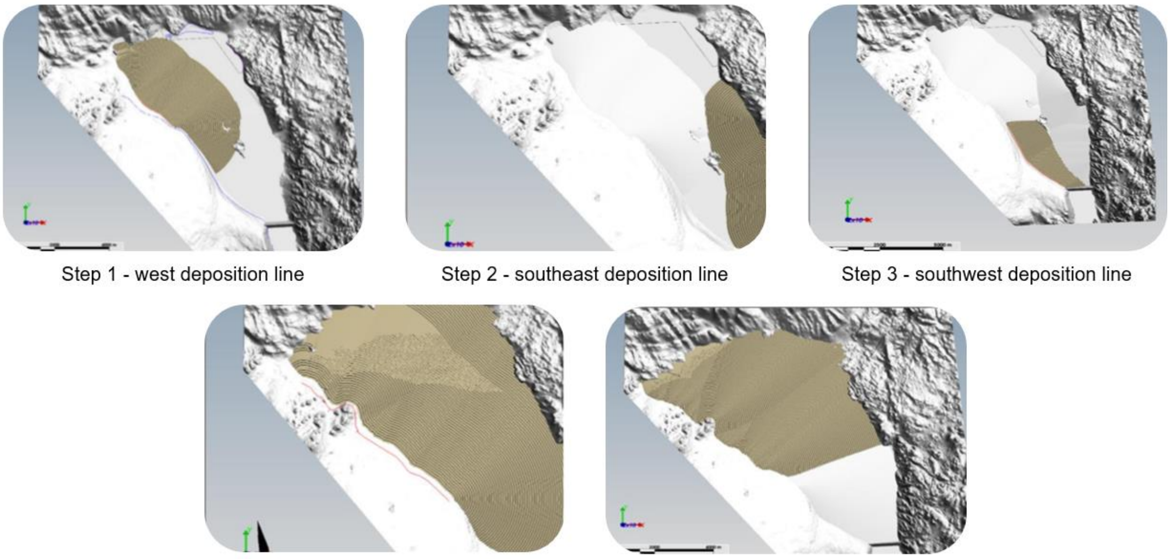
Table 18-11 summarises the proposed deposition strategy.

Table 18-11 Summary of deposition strategy

Step	Deposition line ID	Length (m)	Elevation (RL m)	Storage vol. (Mm*)	Cum. vol. (Mm^3)	Tails footprint (ha)	Cum. time (years)
	West	1,960	3,509	152.5		1,240.3	**5.7**
	Southeast	3,730	3,495		223.6	743.6	**7.9**
	Southwest	3,550	3,492	33.0	256.6	387.1	
	West (long)	5,780	3,546	812.4	1,069.0	3,279	**29.0**
	West (short)	4,990	3,557	180.0	1,249.0	2,699.6	

Tailings will be discharged through spigots positioned along the deposition lines, generating a controlled downward slope toward the southern portion of the facility. The Salar de Taca Taca's topography is characterised by an exceptionally flat gradient, with an elevation drop of approximately one metre over a 7 km span. This minimal natural slope inherently restricts the lateral spread of tailings from each deposition point. Over time, as tailings accumulate, the northern section of the facility will experience progressive increases in deposition depth. Concurrently, the active boundary of the tailings beach will advance southward, as illustrated in Figure 18-29.

Figure 18-29 Progress of tailings deposition



Step 1 - west deposition line

Step 2 - southeast deposition line

Step 3 - southwest deposition line

Step 4 - west (long) deposition line

Step 5 - west (short) deposition line

18.9.9 Tailings pipelines and spigots

Two pipelines would be installed from the concentrator to the TSF, both operational. These lines would pump tailings direct to different sections of the TSF for controlled distribution of tailings into the facility.

Around the TSF perimeter, tailings would be spigotted into the facility through 150 mm diameter spigot lines spaced at 40 m apart. Discharge points would be moved around to allow surface areas of the TSF to dry out and consolidate, and to control the position of the pool of supernatant liquid. Movement of the piping and spigots would be by wheeled equipment, as carried out at other FQM sites.

Coarser material would settle out around the spigot discharge points, and the fines and water would flow into the centre of the TSF. The coarse material would form a cone of material which would overlap with the cone from the adjacent spigot, forming the beach of the TSF.

The discharge of tailings around the TSF would be managed to drive the decant pool to the northwestern end of the TSF, far away from the embankment. The decant pool would be kept adjacent to the natural ground in this area, so that there would be access to the decant pumps from the edge of the facility.

18.9.10 Tailings return water

A water balance model was developed to estimate inflows, outflows, and storage requirements. The model accounts for high evaporation rates and limited precipitation, emphasising the need for efficient water management.

Results indicate that water recycling and conservation will be critical. Sensitivity analyses highlight the impact of climate variability and operational parameters on water balance. During the first seven years between 200 and 500 m^3/h could be recovered during the first eight (wetter) months of the year, while the period after year 7 will recover between 500 and 800 m^3/h.

Under ideal climatic conditions, the design of the TSF would allow decant water and any rainfall to be recovered from the tailings after placement. Where possible, water return from the storage would be maximised and tailings return water (decant water) would be used in the plant in preference to make-up water from borefields.

As the tails are deposited predominantly from the South and West sides of the TSF, beaches will be established, and it is the intent that the trenches will be located with a practical operating length away from the toe of the developing tails beach.

The trenches will collect both localised seepage of brine and fresh water from the tails, and eventually as the operation matures, some surface run-off.

There may be an ability to return some of the decant water from the starter TSF by means of excavating a collection trench.

In the first years of the Project, the flat surface of the salar could be an impediment to the formation of a defined pool of supernatant water. Initial tailings deposition may lead to a shallow depth of water spreading across the salar surface and being evaporated. To avoid this situation, a deep pond could be created by excavating a trench below the salar surface, at a location where deposition is taking place, and then installing decant pumps to return water before it evaporates.

Returned decant water, brine, will be returned to the tails thickeners and used exclusively for tails dilution.

It is envisaged that other remote brine bore pumps will supplement the flow of brine, if the evaporation and fluid bound in tails exceeds the loss of fresh water, plus brine dilution from tails. The design of the brine make-up facilities will assume that zero tailings decant water is available (the worst-case scenario applicable to the dryer months of the year).

18.10 Camp

Information on the camp location, capacity and facilities as presented in the 2021 Technical Report has been updated according to the following commentary, sourced from Fluor Australia Ltd (Fluor, February 2025A).

18.10.1 Concept location, capacity and facilities

The Project camp is proposed to be a staged construction to account for the increasing demands of the workforce as pre-stripping is undertaken, to be followed by construction, commissioning and operations. It is anticipated that the camp will house 4,000 people in total, in two stages of 2,000 capacity.

Initially, a tranche of 500 construction quality rooms (with two beds per room) will be mobilised and established on site, which together with the initial central core facilities will allow for the contractor temporary accommodation facilities to be demobilised. In total, with construction quality rooms, the camp will accommodate a peak of 5,000 personnel.

During operations, the initial 500 rooms will be reserved for major shutdown times and the majority of the junior rooms will be changed from two beds per room to single rooms, in accordance with the Mining Union requirements.

The camp will have the following facilities:

- administration and check-in building
- clinic
- facilities management office
- commercial sized laundry
- kitchen
- dining room
- recreation rooms
- three levels of accommodation (executive, supervisor and employee) will be provided, as follows:
- gymnasium

- indoor sporting facility
- covered walkways that interconnect all facilities

A layout drawing for the concept camp is shown in Figure 18-30.

During the initial mining pre-strip phase, a temporary camp will be required to accommodate mining personnel and mining equipment assemblers.

18.10.2 Facilities and activities for management and/or disposal of solid wastes

Waste water treatment plant

A waste water treatment plant (WWTP) will be located to the north of the camp, downstream of the prevailing winds, but close enough to service the majority of flows on a daily basis. In order to be able to utilise the waste water, a treatment plant sufficient to produce class A effluent will be required.

The proposed treatment plant is a membrane bio-reactor (MBR) designed in two stages, each capable of handling the waste water from 2,000 personnel at a rate of 150 L/person/day.

In stage 1 the effluent will be disposed of in a spray field, sized at approximately 70 hectares depending on soil conditions, and located to the west of the process facilities, sheltered by a major rise on the landscape. In stage 2 the effluent will be treated by a granulated activated carbon (GAC) filter and ultraviolet (UV) sterilisation or chlorination for use as process water top-up.

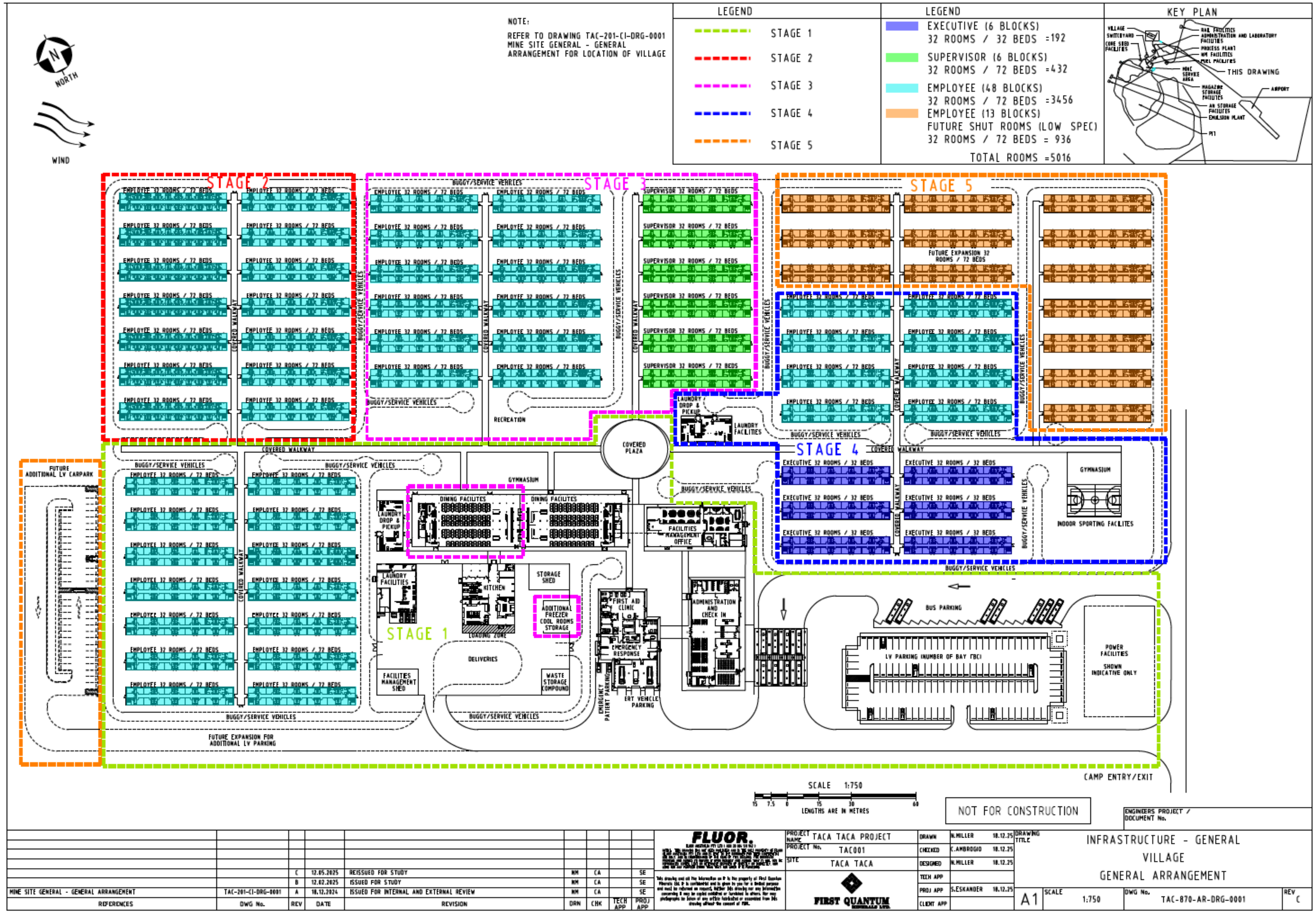
A vendor has indicated that the waste water from the WWTP could be mostly recovered and the sludge waste output disposed of via a waste suction truck. This vendor has confirmed that the quantity of water recovered will be in the range of 80% to 95%. There is an additional option to pelletise the waste sludge (for easier disposal); this option could be considered during the Project design phase.

Waste water will be collected in sealed, below-ground HDPE tanks and a combination of pump lift stations, servicing buildings or clusters of buildings, and then transferred by daisy-chained pits and pump stations from the MSA/NPI area to the process plant, and ultimately to the WWTP. The camp waste water would be transferred directly to the WWTP.

Solid waste disposal

Two incinerator units will be provided in the solid waste disposal area located NE of the camp. The slab is to be bunded and a roof fitted with ridge venting.

Figure 18-30 Concept camp layout (source: Fluor, February 2025)



18.11 Other facilities and infrastructure

18.11.1 Concentrate storage

Copper concentrate will be discharged from the filters direct to a concentrate conveyor, and thence to the enclosed copper concentrate storage shed, the general arrangement for which is shown in Figure 18-31. Within the storage facility, concentrate will be transferred to a tripper conveyor, which would discharge the concentrate along a 120 m long stockpile area.

The proposed storage area is totally enclosed, except for the rail access gallery which will be sheltered from wind and concentrate dust egress. The facility is designed to hold 45,000 tonnes of product, or approximately two weeks of production, and is located on the edge of the salar, aligned with the new rail spur.

Rotainers are the proposed means by which copper concentrate will be transported in rail wagons to the port at Mejillones. Within the storage area, an overhead crane will lift the rotainer air-sealed lids. Two front end loaders will permanently operate inside the facility and tram concentrate from the stockpile to and from the empty rotainers until filled and the lids replaced. Weightometers over the rails will ensure filling within tolerance, and to balance the overall load on the rail system. Once filled, the lids will be replaced, providing for airtight and dust free containment, as required for exportation via Chile.

Dust extraction over the rotainers will seek to collect and return concentrate dust back to the stockpile, and a concrete sealed floor below the wagons is to be periodically cleared using sweeping machines and bobcats to recover spillage.

The storage facility will incorporate a stockpile drainage system to recover any residual water that may be released from the concentrate within the stockpile. The collected water will return to the process plant.

18.11.2 Concentrate loadout and rail spur lines

Approximately 2 ½ trains are expected to arrive at Taca Taca per day, returning 36 wagons, each carrying two rotainers. The rotainers have a carrying capacity of up to 24 tonne of concentrate but will be loaded with around 16 tonne of concentrate in order to meet the axle loading limitations of the rail system.

Trains will arrive at the facility and marshall along the shunting lines, from where the locomotives would decouple, refuel and then reconnect before hauling a full train of filled concentrate wagons on the return journey to Mejillones. Shunting locomotives would break the empty train consist into four individual trains of nine wagons (18 rotainers). Each of these smaller trains would be shunted around and into the concentrate storage area.

Once full, the shunting locomotive will assemble a fresh train of empty wagons, and return the full wagons to the marshalling sidings, where they will be assembled awaiting the next locomotive arrival.

Two rail lines are allowed in the concentrate area, one through the building for loading, as described above. The second line will allow the shunting locomotive to move from the front to the back of the consist, and alternatively allow for a consist of general cargo to park, from where reach stackers can unload containers of reagents and grinding media, and also load containers of molybdenum concentrate bags.

Another siding is allowed for on the eastern side of the concentrate shed, for additional general cargo loading and unloading flexibility.

Figure 18-32 shows the proposed layout for the rail sidings and concentrate loadout facilities.

Figure 18-31 Concentrate storage building general arrangement (source: Fluor, February 2025)

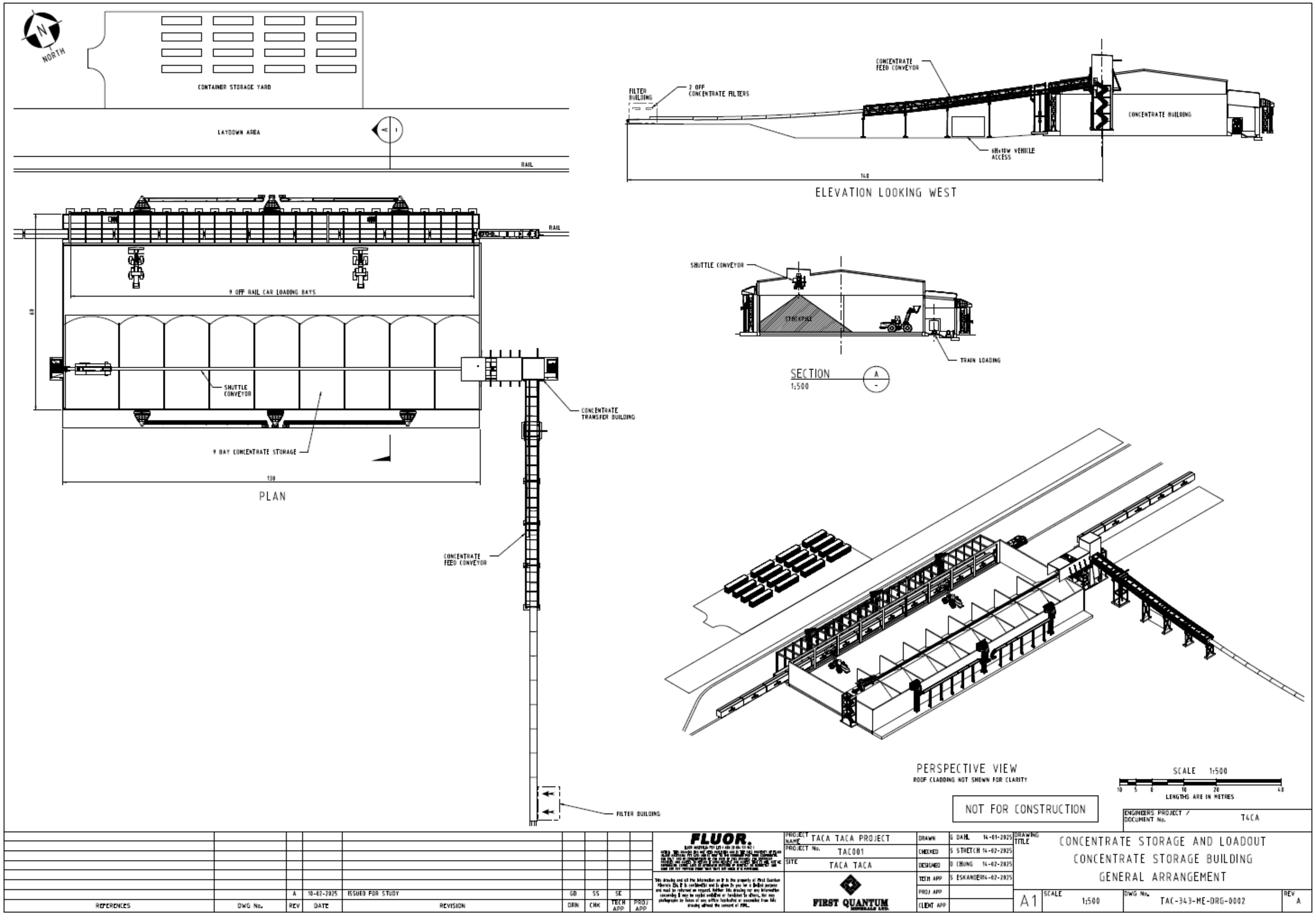
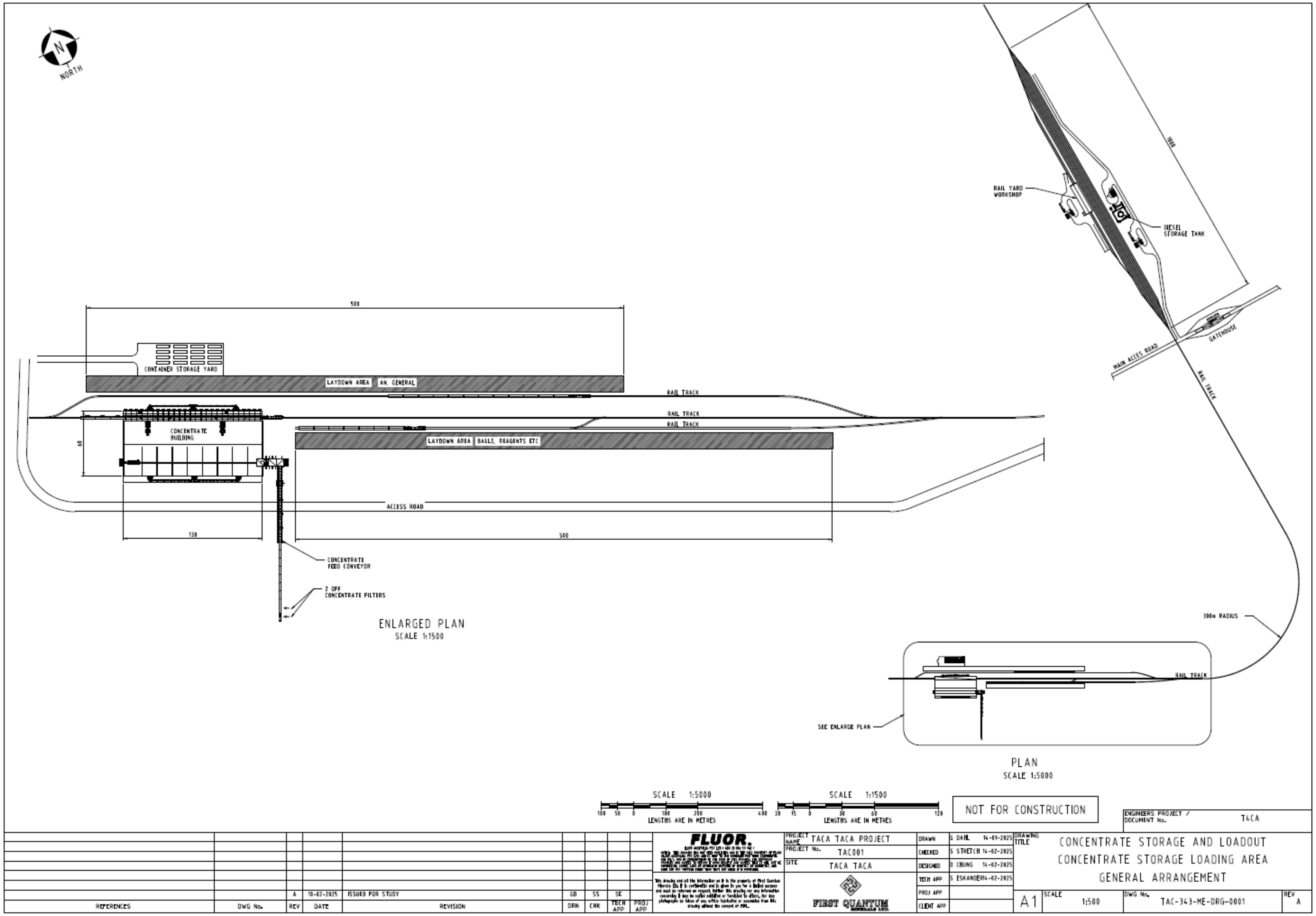


Figure 18-32 Layout of rail sidings and concentrate loadout (source: Fluor, February 2025)



18.11.3 Marshalling yard and construction layout area

The proposed marshalling yard is located on the north side of the plant security gate and is the first rail facility to be encountered in the plant (Figure 18-1). The intention is to limit external access for rail deliveries to this area, where all rail movements from the marshalling yard, back and forth along the spur line, will be managed with shunting locomotives. This facility is also located just north of the main access road, avoiding any disruption of road access by periodic full trains entering or exiting the facilities.

The marshalling yard will be comprised of ten rail sidings. Maintenance of the rolling stock will be undertaken by the rail operator, and conducted in Mejillones, however in the event there are unforeseen maintenance issues, two sidings on the western side of the yard have been allowed to cater for shorter term repairs. The facility is designed at 100 m long, 20 m wide and includes a 20 t overhead crane.

On the far eastern side of the yard, two sidings have been allowed for locomotive fuelling, and if required, fuel unloading in the event of fuel being delivered by rail from either Chile or Salta.

The central six sidings are to allow for:

- one full train awaiting loading,
- one partially filled train,
- one empty train, recently arrived on site,
- one partially empty train in the process of being filled, and
- two sidings for shunting loco movements and any general goods train arrivals or departures.

The overall facility is just over 1 km in length, with the shortest siding being 600 m long.

The area north of the rail easement and confined by a small rise on the salar to the north provides for a 15 ha area, which is around 1,000 m in length. This site will be used during construction for goods delivery and laydown. It is directly accessible from the main access road before trucks are required to enter the construction area. As a site logistics consideration, this laydown area is 1,000 m from the mill building and 1,500 m from the MSA area.

18.11.4 Fuel storage and dispensing

Delivery to site of diesel fuel by both road and rail have been catered for, with the location of unloading facilities positioned just after the site entrance and adjacent to the rail sidings.

Initially, diesel fuel is likely to be delivered by truck to site and will be unloaded to a 1,600 m^3 storage tank. A small adjacent tank will be used to hold and dispense diesel fuel to the locomotives in readiness for their departure to Mejillones.

The balance of fuel will be pumped from this location along the main LV Road to the larger HV storage tank (5,780 m^3 capacity), where a seventeen day supply of diesel will be maintained to satisfy mining, LV and other miscellaneous fuel requirements.

18.11.5 Airstrip

Approval was received in October 2020 from the National Civil Aviation Authority (ANAC) for the construction of a new airstrip located approximately 8 km to the east of the Project site and orientated in a north-west to south-east direction to align with the predominant wind direction. This airstrip would accommodate regular arrival and departure of planes (including medical evacuation planes) to provide services to the Project.

The proposed location of the airstrip is shown in Figure 18-33.

Considering the high altitude and remote location of the Project, personnel will fly into the site using a Dash 8 turboprop plane. Each Dash 8 plane will have the capacity to seat 65 people. The frequency will be four flights per day, arriving from 7:30 am, and will operate seven days per week.

A modest terminal building is to be located at the airstrip with a baggage check-in area and weigh scales, and a baggage screening area and passenger waiting area with ablutions. The terminal building will contain a weather station and communications room for communications with the plane before, during and after flights (Figure 18-34).

Power requirements at the airport will be provided by a standalone solar panel and battery, designed to provide power during the daylight operating hours of the facility.

Pavement markings will be required to ensure visibility for the flight crew, with daily surface maintenance. Occasional reconditioning of the pavement with brine is proposed.

No refuelling facilities are proposed.

The site will include a 1.8 m high chain link fence to ensure no access by fauna. Parking will be provided for a water truck, grader, and temporary parking for six light vehicles and two coaster buses.

Firefighting services will be provided by two foam firefighting trailers/mobile caddies conforming to the requirements of RAAC -153 (*Regulaciones Argentinas de Aviación Civil*).

Geotechnical investigations

The proposed airstrip has been sited at its previously approved location on the Salar de Arizaro. It's geometric design will include a 4.2 km long x 150 m wide runway including shoulders, constructed above the existing salar surface. Structural pavement design of the airstrip is based on a total pavement depth of 500 mm, including 300 mm of sub-base coarse fill, 150 mm of base-coarse, and 50 mm of compacted and graded wearing course. Earthworks will be undertaken during the initial development of the Project to service construction and operations requirements, and to provide crushed and screened mine waste. This material will be hauled to the airstrip site using normal road dump trucks and placed and compacted to specification in situ.

Further specification, parameters and grading for the pavement materials will need to be developed and recommended by a geotechnical pavement designer during detailed design.

18.11.6 Security

The original public road across the Salar de Arizaro will be redirected to avoid close proximity to the operation and inadvertent access into the Project. The security facilities will be located on the salar, along a by-passed length of the existing road. At this location, site access will be controlled by means of a single entry/exit point with a gatehouse for vehicle inspections, a weighbridge and a covered turnstile area for employees.

Figure 18-33 Location of proposed airstrip (source: Fluor, February 2025)

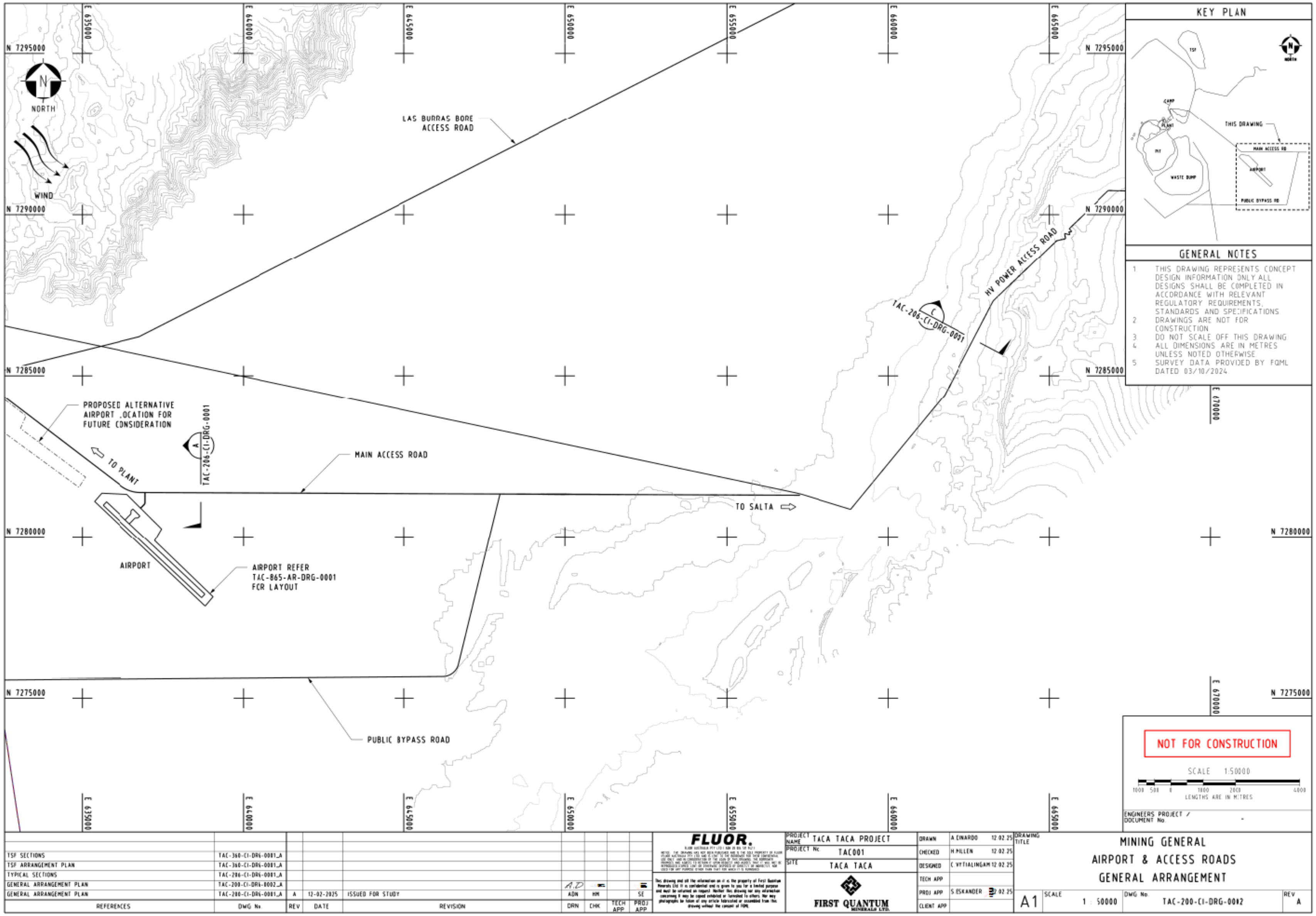
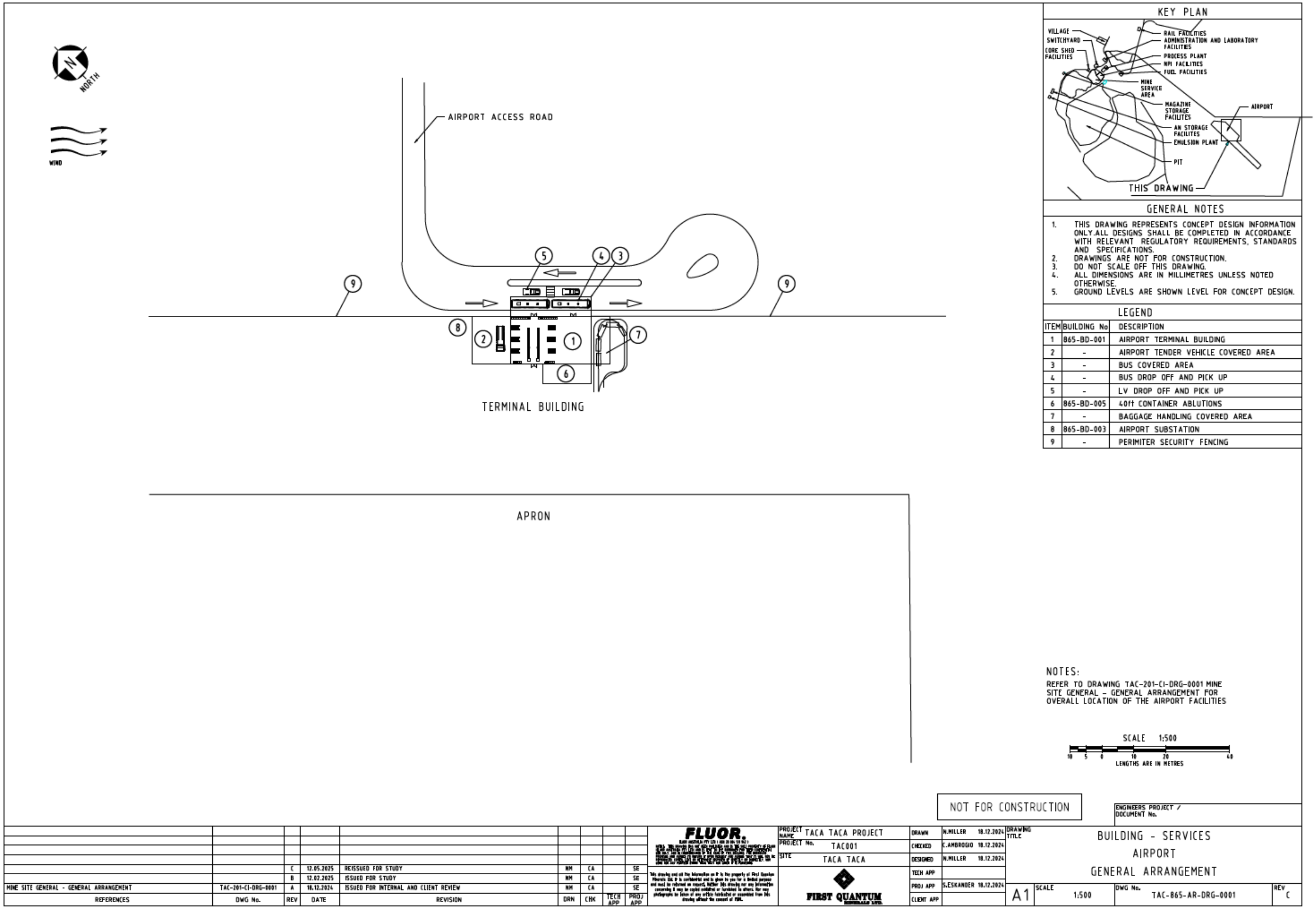


Figure 18-34 General arrangement of the airport facilities (source: Fluor, February 2025)



18.11.7 Construction site utilities and services

The Project execution contemplates the minimisation of specific construction facilities, in lieu of installing permanent infrastructure as soon as possible, and repurposing some of these facilities for use during operations. The intentions are as follows:

- Temporary accommodation will be utilised by existing regional contractors to support off-site remote works associated with the power line and water resources.
- The on-site earthworks, camp and concreting contractors will mobilise small mobile pioneering camp facilities, intended to be replaced with the first phase of the permanent camp rooms and central facilities
- Maintenance workshops and reagents sheds will be utilised as temporary construction workshops for site fabrication and assembly works.
- Initially, it is proposed to install a number of "tensioned skin" or industrial tent structures which would support the early works. These are proposed to be located in positions where they can be utilised as overflow facilities in operations (e.g., surplus copper concentrate storage areas).

18.11.8 Run-off water management and sediment control

The Project area is located in an arid environment characterised by very low rainfall and high evaporation rates. Taca Taca lies on the western margin of the Siete Curvas basin, an extensional basin in which the 80 km x 30 km Salar de Arizaro occupies the western third. Rainfall and snow at higher elevations are the major source of water run-off in the region. This run-off infiltrates the subsurface and flows down into the extensive salars, or in some cases into thick gravel basins where it has accumulated over time.

Rainfall at Taca Taca is very low, with an annual precipitation of approximately 49 mm/year and an evaporation rate of 1,770 mm per year. Consequently, and during construction, building pads will be graded typically at 1% to allow for overland run-off for storm flow, and roads typically with a crossfall of 3% to v-channels to direct runoff.

Detailed design for finished surfaces will need to be undertaken during the detailed design phase to ensure flow away from buildings and with channelling into run-off ponds for sediment control. Erosion and sediment control will from part of the detailed design phase.

18.11.9 Salta facilities

A logistics consolidation facility will be provided at Salta for goods being delivered by road to site (the "Salta Operations Centre", SOC). Goods will be off loaded, preassembled, fabricated and painted as appropriate, and will be delivered just in time to site, where possible using a Taca Taca dedicated truck fleet.

The primary Project warehouse and laydown yard will be located at the SOC, inclusive of the following facilities:

- a large warehouse including air-conditioned offices (one partitioned office and one open plan office with service counter),
- tea preparation area, library, clean room, parts issue area, tool store, ablutions, change room and lockers, and
- parts staging area, electric charging area and electrical repair area.

The rear of the SOC site will have two large, fenced laydown yards, i.e.:

- a Projects laydown yard (approximately 62,950 m^2), and

- an Operations laydown yard (approximately 70,990 m^2).

Both yards will have vehicle access gates for deliveries and pickup of equipment / supplies from the mine, MSA and NPI areas, as required. There will be a connecting vehicle access gate to allow parts, supplies and equipment to be moved efficiently between yards. These yards will be used as staging areas during construction and throughout the operation of the mine and plant.

The primary Project administration and training facility will be located at the SOC. The administration building will contain a reception area, open plan offices, several two-person offices, meeting rooms, a board room and general office facilities. There will be an allowance for Project control room, and a future integrated remote operations centre (IROC) to enable remote operations of HV equipment at the mine site.

The administration space is intended to accommodate all personnel whose roles are not essential to the site operations, including most of the finance, commercial and HR teams, in addition to some mine technical and maintenance planning personnel, amongst others.

The training facility will be equipped with a computer training room, two lecture rooms and workshop training areas. Offices will be provided for the training staff, as well as tea and coffee making facilities. A dining room will be provided with space for seating and ablutions.

The SOC is proposed to be located on the outskirts of the city, in proximity to the city ring road system. This location ought to provide ready access for materials supply and equipment delivery and provide convenient access for Salta based employees. The SOC will also be positioned within easy access to route RN Nº.51, so that goods can be dispatched to site without transiting around and through the city.

ITEM 19 MARKET STUDIES AND CONTRACTS

19.1 Product marketability and demand

The plans for the Taca Taca Project are to produce separate copper and molybdenum concentrates.

The quality of the copper concentrates produced at the Project is assumed to be similar to the qualities of copper concentrates produced at several of the Company's other mining operations. The Company has a high level of confidence regarding the demand for copper concentrates of this quality. In addition to the typical treatment charges, the smelter customer will realise revenues against acid that will be produced as a co-product (as well as margins on the payable metals).

The quality of the molybdenum concentrate is considered as good for sale to regional customers.

No deleterious elements have been identified at this time, in either the copper or molybdenum concentrates, and therefore penalties are not considered.

19.2 Overview of product logistics

Copper and molybdenum concentrates from the Project will be transported by railway to Chile.

The Company assumes that the copper concentrate produced at the Project would be marketed globally to international smelters and exported to those markets by seaborne trade via a port at Mejillones Bay, Chile.

The copper concentrate product would be shipped in bulk, typically in parcels of 10,000 dry metric tonnes. Vessels would be either chartered exclusively for the Company's Taca Taca copper concentrates, or co-chartered with other regional producers. This adds to the flexibility, efficiency and options for delivering the copper concentrate to customers.

Molybdenum concentrate is considered to be potentially saleable within Chile in the first instance, and alternatively could be efficiently shipped in containers from ports in the region, to customers in either western or eastern hemispheres

19.3 Agreements for sale of concentrate

As is the case with products from the Company's other operations, all products will be sold through the Company's internal marketing division, FQM Trading. There are, as yet, no contracts in place for the sale of products from the Project.

FQM Trading has existing sales agreements (in respect to the Company's other operations) and commercial relationships with many of the major international copper smelting companies.

19.4 Other contracts and agreements

No contractual arrangements for concentrate transport, port usage, storage, shipping, smelting, or refining exist at this time.

Contract mining of the substantial pre-strip volume is currently being considered, although no decision has been made at the time of reporting.

The supply of power to the Project will be a major contract and one which is yet to be negotiated.

The supply of diesel fuel into the Project, as another major contract, is also yet to be assessed and negotiated.

ITEM 20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental setting

Further to the Project location and prevailing climatic conditions described in Items 4 and 5, respectively, the environmental setting for the Taca Taca Project can be summarised as follows:

- The site is located in a cold and arid zone, exposed to strong solar radiation and winds.
- The site has hydrographic characteristics typical of the Andean regions, with little permanent surface water, although minor ephemeral and seasonal streams are formed from summer thaw, rainfall, hail and snowfall.
- For the elevations of the site, there are no forms of a glacial or periglacial environment.
- Soils have no agricultural value in the salt flats of Arizaro, although at Taca Taca and the humid area of Valle de Arizaro there are vegetal borders.
- At nearby Caipe there is azonal vegetation which grows in wetlands, associated with water contribution.
- Otherwise, in dry areas, there is zonal vegetation, pedemontes (*pediments)* and alluvial cones, depending on climatic conditions and soil type.
- The Project is located within the "Los Andes Natural Wildlife Reserve." It is one of the three largest protected areas in Argentina and the largest in the Salta province. The Reserve has been classified for multiple use, including for mineral exploration and development activities.
- Although fauna is scarce, certain lizards and birds (yellow winged pigeons, goldfinches, crested duck, peregrine falcon, Baird´s sandpiper, rufous-bellied seedsnipes, puna miner and puna ground tyrant) and mammals (yellow-rumped leaf-eared mouse, Bolivian grass mouse, vicunas) have been found in the area adjacent to the Project:
  - none of these wildlife species have been found in the immediate Project area
- An Integrated Environmental and Social Management Plan for the Project development and future operations will address the overall monitoring and mitigation measures involving all Project components and infrastructures.

20.2 Status of environmental approvals

20.2.1 Background

The 1995 Environmental Protection Mining Code of Argentina requires that each provincial government monitor and enforce the laws pertaining to sustainable development and protection of the environment.

A party that wants to modify or begin any mining related activity as defined by the country's Mining Code (i.e. prospecting, exploration, exploitation, development, preparation, extraction, storage of mineral substances, property abandonment, or mine closure activity) must submit an application to the Provincial Environmental Management Unit (PEMU) and obtain an approved *Informe de Impacto Ambiental* or Environmental and Social Impact Assessment (ESIA) prior to the start of work (Bastida, 2002).

In the Salta Province, the Secretariat of Mining can approve the mining ESIA, thereby acting as PEMU for any mining related activity.

Each ESIA must describe the nature of the proposed work, its potential risk to the environment, and the measures that will be taken to mitigate that risk. The PEMU has a 60-day period to review and either approve or reject the ESIA; however, if PEMU has not responded within 60 days, that does not constitute an approval. If the PEMU deems that the ESIA does not have sufficient content or scope, the party submitting the ESIA is granted 30 days to resubmit their document.

If accepted by the PEMU, the ESIA is used as the basis to create a *Declaración de Impacto Ambiental* or Declaration of Environmental Impact (DEI) to which the party must swear to uphold during the mining-related activity in question. The DEI must be updated at least once every two years. Sanctions and penalties for DEI non-compliance are outlined in the Environmental Protection Mining Code and may include warnings; fines; a suspension of the Environmental Quality Certification; restoration of the environment; temporary or permanent closure of activities; and/or removal of authorisation to conduct mining-related activities.

20.3 Environmental studies

The Project ESIA was submitted in February 2019 (Ausenco, February 2019). Additional ESIA documents have been prepared for the feasibility approval of the 345 kV transmission line connecting the national grid to the Project site (FQM, March 2020). Pre-feasibility for the transmission line was submitted in 2021 to the Mining and Energy Secretariat (SME) and approved in November 2022. The subsequent feasibility ESIA is under evaluation with the authorities after its submission in September 2025.

There is a separate ongoing pre-feasibility ESIA, submitted in 2020, for the proposed access road diversion (FQM, September 2020). There is a new proposed diversion considering a segment of 40 km to avoid the Provincial Route 27 traffic going through the mine. This information is being updated in the permitting application file.

Several Project Description documents were submitted to support these ESIA submissions, including separate descriptions on aspects of the proposed mining and processing plan (FQM, November 2018), the power supply route alternatives and the proposed site access route alternatives. Engineering details are described in these documents in the context of potential environmental and social impacts.

Observations and responses to the Project ESIA submission were received from the Mining Secretariat at the end of Q3 2019, including requests for clarification and further information on some environmental aspects (62 observations). The Company's response document to the Secretariat of Mining was submitted in Q1 2020, including reports on additional studies and aspects such as the conceptual closure plan (Section 20.6), an initial geotechnical investigation of the Salar de Arizaro surface bearing capacity, and an assessment of alternative waste landfill sites, amongst others.

In August 2022, complementary information was submitted by FQM to be incorporated into the ESIA. This information included an SRK Consulting study of alternatives and conceptual designs for the TSF and WRD (SRK, 2022), in addition to a report by Piteau Associates on hydrogeological studies (Piteau, 2022).

A second round of observations (numbering 173) was received from the SME in June 2023. These observations were responded to in October 2023 (FQM, October 2023) including new information on water abstraction from the Socompa, Valle de Arizaro, Valle de Chaschas and Valle de Las Burras basins.

In October 2024, a collaborative workshop was held with SegemAR[14] and the provincial authorities including the SME. This workshop included a site visit, technical briefings and presentations from Company staff, followed by interactive questioning on numerous aspects of the Project.

[14] SegemAR (*Servicio Geológico Minero Argentino) is* a national scientific and technological body responsible for the production of geological, technological, mining and environmental geological knowledge and information on the territory of the Argentine Republic and the continental shelf

The ESIA process requires a final report from the SME on their observations and requests for further information. Once the observations process is satisfactorily concluded, there needs to be a public hearing (*audiencia publica*) prior to the ESIA approval.

In accordance with environmental laws governing the Project, the Project ESIA will need to be updated every two years. Updates to the Project Description will be incorporated into subsequent ESIA submissions accordingly.

20.3.1 Environmental impacts and management requirements

The primary Project environmental impacts and management requirements may be summarised as follows:

1. Emissions of combustion gases (vehicular transit), emissions of particulate material (breathable) by blasting, material loading, tailings sludge and waste deposition:
   - Mitigation measures will include optimisation of heavy equipment, use of equipment with emission control technology, preventive and corrective maintenance of the vehicle fleet, installation of water suppression systems, and spraying of roads to control fugitive dust.
2. Exploitation of groundwater:
   - Mitigation measures will include optimisation of the use of the resource, including the reutilisation of fresh water, respecting the sustainability of the sources, and prospecting for new water supply sites to diversify the use of resources.
   - Additional studies are being undertaken to determine potential risks to the fresh water *vegas* that are characteristic of several of the identified water supply sources.
   - If groundwater monitoring in those environmentally sensitive areas identifies drawdown that may impact vegetation and superficial water systems, then an artifical irrigation scheme may be required.
3. Mining waste disposal considering non-acid forming (NAF) and potentially acid forming (PAF) waste rock laid onto the surface of the Salar de Arizaro:
   - Mitigation of potential acid rock drainage (ARD) conditions will require the placement of a base layer of NAF rock, upon which subsequent lifts of the waste dump will be constructed.
4. Changes in the chemical constitution of the aquifers with respect to Salar de Arizaro and Salar de Taca Taca:
   - Mitigation measures will include a Management Plan for the control of contact and non-contact waters (specifically involving routine water quality monitoring).
   - These measures will include the containment of run-off from structures and facilities within the Project site and the direction and storage of any such run-off such that it could be used for mineral processing purposes.
   - Measures will also include the capture of seepage from the Salar de Arizaro into the mined void. This will serve two purposes, assisting with drained pit slope conditions and providing an additional source of process water.
5. Protection of biodiversity:

   Biodiversity protection measures such as compliance with traffic regulations, the installation of traffic signs to define fauna crossings and fencing of certain areas. Other measures may include fauna protection through dissuasive systems to reduce impacts due to noise and light.

- Additional compensation measures, due to the tailings deposition, include the relocation of Vega Salitrosa and Plumas Verdes to similar environmental areas and other salars close to the Taca Taca Salar.
- Mitigation measures include control and management systems for chemicals and resulting waste. These measures involve the use of secondary containment systems, in accordance with the chemical and waste management plan, to prevent contact in the event of possible spills and contamination of soil and water.

6. Environmental monitoring:
   - The Monitoring Plan is part of the Project's Environmental Management Plan, which allows for the control of mitigation measures and adjustments throughout the life of the Project.
   - Monitoring programmes will be undertaken to allow systematic data collection in order to identify and assess impacts which will be compared with baseline and regulatory parameters, when applicable.
   - Systematic monitoring provides effectiveness of environmental measures. Monitoring will involve internal audits, government and the participation of communities. Since 2020, under the provisions of Resolution 004/18 issued by the SME, FQM has complied with the environmental community monitoring programme which involves Tolar Grande.

The overall environmental management plans and strategy will align with the local requirements regarding site-specific conditions such as The Los Andes Protected Reserve. Social/community impacts and management requirements are listed as follows:

- In 2023, the consultancy firm E&C Asociados carried out the update of the social baseline for the ESIA, as well as the identification of key stakeholders within the Project's area of influence (E&C Asociados, September, 2023). The study included a historical overview of each community, along with an analysis of socioeconomic and cultural aspects, as well as infrastructure, education, health, and employment in the region. A copy of this document is included in Appendix B.
- The communities of the Project's direct influence are mostly composed of members of the Kolla Indigenous Community. In compliance with the legal framework, a Free, Prior and Informed Consultation (FPIC) process was conducted between 2021 and 2024, led by the Secretariat of Indigenous Affairs and the Secretariat of Mining, within the framework of the submission of the ESIAs related to the mine, power line and access road. As a result, in January 2025, a favourable compliance report was issued by the Indigenous Affairs Secretariat for the consultation process (FPIC).
- Between 2016 and 2025, workshops, open meetings, and participatory sessions were held in the towns of Tolar Grande, Pocitos, Olacapato, and San Antonio de los Cobres, with the aim of sharing Project information and to gather concerns and input from the community.
- The main social impacts identified, and the proposed management measures include:
  - migration due to employment opportunities and increased demand for public services
    - technical assistance will be provided through a strategic plan to maximise the benefits of the Taca Taca Project
  - generation of temporary employment associated with the Project:
    - clear and timely communication of recruitment policies and procedures will be promoted.
  - expectations regarding local supplier participation:

- efforts will focus on strengthening community enterprises and promoting local value chains.

Proposals to reduce the footprint of Waste Rock Dump has been submitted the Mining Secretariat for review and comment. A reduced footprint would result in higher final dump elevations, this however would not compromise stability as indicated by preliminary assessments nor have a material impact on haulage costs.

20.3.2 Noise emissions

DAKAR Ingeniería Acústica (DAKAR, March 2020) carried out an environmental noise assessment, the findings from which were submitted to the environmental authorities in response to their ESIA review observations and questions. Year 5 of the operations was modelled, and it was concluded that there would be negligible site generated noise emissions audible in Tolar Grande.

Noise emissions will be considered in the upcoming community monitoring campaigns.

20.3.3 Carbon emissions

An estimate has been made of the annual carbon dioxide emissions from the diesel fuel combusted in the mine and from the power generated to supply the processing plant and site infrastructure. The use of internal carbon pricing in new project evaluation is intended to incentivise the use of lower carbon energy sources and ensure resilience of the Project to transitional climate risk.

The Project will source its electrical energy requirements through a long term power supply agreement with an electricity supplier, to be determined through a competitive tender process. The Company has identified options to source 100% of its electrical energy requirements from renewable sources, in particular from wind and solar power generators. Further alternatives exist, if required, to source a portion of the energy requirements from natural gas power plants in Salta and regionally. Supply negotiations will continue during the Project engineering phase. It is intended to implement trolley assist for medium and long haulage routes, which is expected to result in lower diesel consumption and operational efficiencies.

The annual average diesel fuel related (Scope 1) emission is approximately 125 ktonnes of carbon dioxide. The emissions would be nil if 100% of the supply was supplied from renewable power supply sources. Whereas an estimated annual average of approximately 260 ktonnes of carbon dioxide would be emitted from combined 20% renewable and 80% natural gas generated power supply sources. Table 20-1 lists the annual diesel fuel and power consumption for the Project, in addition to the estimated emission of CO_2. To note from this table:

- the diesel fuel consumed in the plant is as a reagent for molybdenum flotation, and is not combusted
- the calculation of CO_2 emissions relates to the following coefficients:
  - diesel fuel = 2.66 kg CO_2 per litre of fuel
  - explosives = 0.19 kg CO_2 per litre of fuel
  - electric power = 271.5 g CO_2 per kWh from a natural gas fuelled power station
- these coefficients are taken from International Energy Agency (IEA) reference information for Argentina
- a carbon tax has been in place in Argentina since March 2018, however the tax applies to liquid and solid fuels, but not to natural gas (reference OECD website: http://www.oecd.org/ctp/taxing-energy-use-efde7a25-en.htm)
- the fuel price adopted in the operating cost estimates for the Project already accounts for an emissions tax passed on from a supplier

- the proposed power supply for the Project is via an interconnector on the existing 345 kV transmission line, and the generated power could be:
  - from 100% renewable sources, or from a combination of natural gas (which is tax exempt) and renewable sources

Table 20-1 Carbon dioxide emissions

Year	Plant Feed (Mt)	Mining (fleet consumption) Diesel fuel (ML)	Processing (not combusted) Diesel fuel (ML)	Explosives (kt or ML)	Mining (fleet consumption) Electrical Power (MWh)	Processing plant Electrical Power (MWh)	Mining consumption: Diesel coeff. 2.66 tonne CO_{2e}	Bulk explosives: Explosives coeff. 0.19 tonne CO_{2e}	Power consumption: LNG coeff. 271.5 tonne CO_{2e}	Total CO_2 emission tonne CO_{2e}	Scope 1 emission: Total estimated emission tonne CO_{2e}	Power supply emission: Base case 100% renewable tonne CO_{2e}	Power supply emission: Alt case (80% natural gas, 20% renewable) tonne CO_{2e}
1		10.2		5.5	14,363		27,062	1,038	3,899	31,999	28,100	0	3,120
2		41.4		27.2	75,794		110,317	5,137	20,578	136,033	115,455	0	16,462
3		54.7		44.8	113,891		145,490	8,470	30,921	184,881	153,960	0	24,737
4		62.1		54.0	139,724		165,331	10,204	37,935	213,470	175,535	0	30,348
5	**24.9**	76.8	0.01	71.2	174,730	848,432	204,317	13,466	277,788	495,572	217,783	0	222,231
6	**40.0**	76.0	0.01	71.1	168,659	1,131,243	202,273	13,439	352,923	568,635	215,711	0	282,339
7	**40.1**	79.7	0.01	69.1	152,020	1,131,243	212,125	13,063	348,406	573,595	225,189	0	278,725
8	**40.0**	77.4	0.01	70.3	215,552	1,131,243	206,110	13,281	365,655	585,045	219,391	0	292,524
9	**40.0**	83.1	0.01	72.6	282,001	1,131,243	221,203	13,713	383,696	618,612	234,916	0	306,957
10	**40.0**	78.9	0.01	71.3	235,406	1,131,243	210,025	13,469	371,045	594,539	223,493	0	296,836
11	**40.1**	83.7	0.01	74.0	260,363	1,131,243	222,719	13,986	377,821	614,526	236,705	0	302,257
12	**40.0**	84.3	0.01	76.6	255,291	1,131,243	224,283	14,473	376,444	615,200	238,756	0	301,155
13	**40.0**	81.4	0.01	74.2	273,989	1,131,243	216,756	14,022	381,520	612,298	230,777	0	305,216
14	**40.0**	73.2	0.01	73.6	310,000	1,131,243	194,813	13,915	391,297	600,025	208,728	0	313,038
15	**40.1**	76.5	0.01	74.4	282,048	1,131,243	203,657	14,053	383,708	601,419	217,710	0	306,967
16	**40.0**	82.6	0.01	70.5	227,154	1,131,243	219,825	13,318	368,805	601,948	233,143	0	295,044
17	**40.0**	66.1	0.01	64.8	258,446	1,131,243	175,966	12,254	377,301	565,521	188,221	0	301,841
18	**40.0**	55.9	0.01	57.0	256,075	1,131,243	148,715	10,781	376,657	536,153	159,496	0	301,325
19	**40.1**	54.6	0.01	49.0	202,928	1,131,243	145,189	9,265	362,227	516,682	154,454	0	289,782
20	**40.0**	55.6	0.01	47.8	214,280	1,131,243	148,103	9,041	365,309	522,453	157,144	0	292,247
21	**40.0**	55.8	0.01	47.2	239,410	1,131,243	148,416	8,912	372,132	529,460	157,328	0	297,706
22	**40.0**	56.1	0.01	46.1	233,065	1,131,243	149,331	8,720	370,410	528,460	158,050	0	296,328
23	**40.1**	50.3	0.01	42.8	221,444	1,131,243	133,815	8,087	367,254	509,156	141,901	0	293,804
24	**40.0**	52.0	0.01	42.9	190,195	1,131,243	138,488	8,111	358,770	505,369	146,599	0	287,016
25	**40.0**	52.4	0.01	43.8	168,713	1,131,243	139,382	8,276	352,938	500,596	147,658	0	282,350
26	**40.0**	56.4	0.01	43.7	186,706	1,131,243	150,001	8,251	357,823	516,075	158,252	0	286,259
27	**40.1**	61.4	0.01	44.2	193,611	1,131,243	163,524	8,355	359,698	531,577	171,879	0	287,758
28	**40.0**	51.8	0.01	43.4	251,853	1,131,243	137,820	8,210	375,510	521,541	146,030	0	300,408
29	**40.0**	53.0	0.01	45.3	255,190	1,131,243	141,040	8,558	376,417	526,015	149,598	0	301,133
30	**40.0**	49.8	0.01	45.0	345,731	1,131,243	132,553	8,498	400,998	542,049	141,051	0	320,799
31	**40.1**	33.4	0.01	35.4	312,769	1,131,243	88,765	6,688	392,049	487,502	95,453	0	313,639
32	**40.0**	26.7	0.01	29.2	247,907	1,131,243	71,078	5,519	374,439	451,036	76,597	0	299,551
33	**40.0**	32.6	0.01	28.8	249,202	1,131,243	86,849	5,441	374,791	467,080	92,290	0	299,833
34	**40.0**	31.5	0.01	28.5	266,362	1,131,243	83,913	5,378	379,450	468,741	89,292	0	303,560
35	**40.1**	32.0	0.01	28.8	244,299	1,131,243	85,260	5,434	373,460	464,153	90,693	0	298,768
36	**40.0**	33.1	0.01	29.5	72,634	1,131,243	88,099	5,572	326,853	420,524	93,671	0	261,482
37	**40.0**	35.6	0.01	29.9	118,058	1,131,243	94,628	5,653	339,185	439,467	100,282	0	271,348
38	**40.0**	33.6	0.01	29.2	206,649	1,131,243	89,440	5,519	363,238	458,196	94,959	0	290,590
39	**40.1**	27.0	0.01	15.4	122,333	1,131,243	71,962	2,906	340,346	415,214	74,868	0	272,277
40	**40.0**	27.2	0.01	15.2	120,796	1,131,243	72,277	2,881	339,928	415,087	75,158	0	271,943
41	**40.0**	22.4	0.01	16.6	131,947	1,131,243	59,660	3,133	342,956	405,749	62,793	0	274,365
42	**40.0**	13.2	0.01		33,576	1,131,243	35,102		316,248	351,350	35,102	0	252,999
43	**40.1**	13.2	0.01		33,576	1,131,243	35,154		316,248	351,402	35,154	0	252,999
44	**40.0**	13.2	0.01		33,666	1,131,243	35,131		316,273	351,404	35,131	0	253,018
45	**40.0**	13.2	0.01		33,576	1,131,243	35,102		316,248	351,350	35,102	0	252,999
46	**40.0**	13.1	0.01		33,574	1,131,243	34,818		316,248	351,066	34,818	0	252,998
47	**40.1**	13.1	0.01		33,576	1,131,243	34,849		316,248	351,098	34,849	0	252,999
48	**40.0**	13.1	0.01		33,666	1,131,243	34,826		316,273	351,099	34,826	0	253,018
49	**40.0**	13.1	0.01		33,576	1,131,243	34,797		316,248	351,045	34,797	0	252,999
50	**40.0**	13.1	0.01		33,574	1,131,243	34,797		316,248	351,044	34,797	0	252,998
51	**40.1**	13.1	0.01		33,576	1,131,243	34,849		316,248	351,097	34,849	0	252,999
52	**40.0**	13.1	0.01		33,666	1,131,243	34,826		316,273	351,099	34,826	0	253,018
53	**40.0**	13.1	0.01		33,576	1,131,243	34,798		316,248	351,046	34,798	0	252,999
54	**43.9**	12.5	0.01		33,574	1,131,243	33,255		316,248	349,503	33,255	0	252,998
TOTAL	**1,990.1**	**2,398.2**	**0.44**	**1,949.7**	**8,928,338**	**56,279,330**	**6,382,883**	**368,489**	**17,703,882**	**24,455,254**	**6,751,373**	**0**	**14,163,105**
AVERAGE	**39.8**	**44.4**	**0.01**	**47.6**	**165,340**	**1,125,587**	**118,202**	**8,988**	**327,850**	**452,875**	**125,025**	**0**	**262,280**

Considering the above, the impact of a carbon tax is considered to be immaterial to the economics of the Project.

20.3.4 Resettlement

There are no known resettlement requirements for the Project.

20.3.5 Community engagement

The Project's community engagement approach involves ongoing communication with local authorities and community leaders, active participation in a 'Social Round Table', implementation of participatory monitoring, and the development of socio-educational and socio-community programmes. It also includes support for cultural and social events, and continuous collaboration with local communities, as part of a relationship building strategy founded on respect and mutual trust.

20.4 Waste and tailings disposal

Item 16.1.6 provides a detailed account of the Project waste dumping strategy. The major components for mine site water management, including management of waste dump run-off, will include diversion channels, collection and settlement ponds.

Item 18.9 provides a detailed account of the TSF design and operational strategy.

In addition to the ponds and channels, a number of bores will be located on the Salar de Arizaro and Salar de Taca Taca around:

- the periphery of the waste dump, for beneficial use of brine in the process, to support stable compaction of waste on the Salar de Arizaro, and eliminate seepage of brine to the pit and associated pressurisation of pit walls (Figure 18-13), and
- at the entrance to the Salar de Taca Taca (Figure 18-14) to intercept potential seepage from the TSF embankment

20.5 Environmental insurance and restoration fund

According to Law 25,675 (National Environmental Policy (Argentina), ARTICLE 22), any natural or legal person, public or private, who carries out activities that pose risks to the environment, ecosystems, and their constituent elements, must obtain insurance coverage with a sufficient entity to guarantee the financing of the restoration of any damage they may cause; likewise, depending on the case and the possibilities, they may establish an environmental restoration fund to enable the implementation of repair actions.

This is to be calculated according to Project components and activities through a chart of rates.

Whether or not such insurance coverage will be required is yet to be determined.

20.6 Project closure plan and provisions

The Project conceptual closure plan and cost provisioning has considered stages of progressive closure during the operation stage, during final closure and post-closure (requiring on-going maintenance and post-closure monitoring).

The cost estimate components account for dismantling and demolition of infrastructure, longer term surface water management and erosion control, and rehabilitation of the landscape. 20.7 provides a summary of the closure plan, including a list of activities through the various phases, in addition to a provisional closure cost estimate by *GT Ingenieria* (2020). The closure estimate includes a provision for employee severance.

20.7 Environmental liabilities

There are no known environmental liabilities currently existing on the Taca Taca property.

ITEM 21 CAPITAL AND OPERATING COSTS

21.1 Cost estimation basis

21.1.1 Capital costs

FQM commissioned an Engineering Cost Study in Q4 2024 through to late Q2 2025 for the purposes of developing a comprehensive Project infrastructure design and associated capital cost estimate for a 40 Mtpa Stage 1 project. Contributors to this study included:

- Lycopodium for the treatment plant (copper and molybdenum processing), primary crushing to copper concentrate filtration and tails pumping.
  - Lycopodium estimated the cost for the tailings pumping system
- Fluor for the mining and non-process infrastructure including the Salta Operations Centre, the site village, remote water and brine borefields, construction and off-site roads, airstrip, mining service area, copper concentrate conveyor and storage area, plus rail loading facilities.
  - Fluor designed and costed the copper concentrate discharge conveyor, the concentrate storage facilities and the facility for loading to rotainers at the side of the rail spur.
- Process E&I for the HV switchyard, power distribution to mine, plant and infrastructure, and support for the HV powerline from La Puna to site.
- FQM project engineers provided input into the self-perform construction components and the commissioning management costs.

Equipment pricing covering around 90% of the mechanical equipment costs was drawn from vendor quotations or escalated from recent as-purchased equipment awards.

The complete Project development costs were subsequently compiled by Lycopodium from these various engineering information sources. A Basis of Estimate document (Lycopodium, June 2025) provided details of the capital cost estimation process, data input, overall quantities, man hours and a contingency assessment.

The estimate was ultimately developed in US dollars, using an FQM treasury agreed schedule for foreign exchange rates that would be applicable over the life of the Project. To accurately capture the cost of localised engineering, certain components were priced in Australian dollars in the first instance, as were Argentine sourced components priced in Argentine pesos (ARS).

21.1.2 Operating costs

Updated mine operating costs comprising drill, blast, load and haul costs were derived in Q3 2025 by FQM mining engineers. These derivations were estimated from first principles using productivity parameters for the proposed equipment fleet, simulated haul profiles related to the staged pit designs and production schedule, and from corresponding ore/waste haulage destinations.

Key mining cost estimate inputs were as follows:

- diesel fuel = $1.02/litre
- bulk explosives = $850/t
- electrical power = $65/MWh

Other specific cost estimate inputs, such as mining consumable unit costs and consumption rates are elaborated in Item 21.6.1.

Updated process operating costs for the Project were developed from first principles by FQM metallurgists and process engineers. The operating costs were compiled from a variety of sources, including the following:

- Reagent consumption based on laboratory testwork and Cobre Panamá design.
- Modelling by Orway Mineral Consultants (OMC) for crushing and grinding energy and consumables, using ore characteristics measured during the testwork.
- Quoted prices or FQM's database of prices for consumables.
- Power consumption based on the mechanical equipment list and industry standards for absorbed power, with benchmarked allowances for areas not fully defined (i.e., Non Process Infrastructure (NPI)).
- Wages and salaries from in-country FQM, based on current operations.
- Maintenance costs calculated as a percentage of the supply cost for concrete, steel, platework, mechanical equipment, pipework, electrical and instrumentation and building costs.
- Maintenance costs include those for the camp and other NPI such as the power line, bore fields, water and waste treatment.
- Shift rosters based on working hours suited to the high altitude and in-country practices.
- Concentrate (rail) transport and port handling charges based on preliminary enquiries by the Company, in both cases the operating costs cover any additional capital expenditure required by the rail and port operator.
- FQM's database of costs for similar sized operations.

Operating costs were determined for a plant with an annual throughput of 40 Mt of ore at a P_{80} grind size of 180 µm, and assuming a 24 hour per day operation, for 365 days per year. The estimated processing costs were itemised as follows:

- operating consumables
- energy consumption
- maintenance materials and contractors
- labour for process plant operations and maintenance

In addition, G&A (general and administration) costs were estimated for administration and non-process infrastructure, including labour, energy and maintenance costs.

Consumption rates were benchmarked against actual operating data from the company's Sentinel and Cobre Panamá operations.

All operating costs are presented in US dollars and reflect prices for the second quarter of 2025. An expected process plant availability of 91.3% was adopted for the estimate.

Fixed costs are defined as those costs in dollars per annum that are independent of throughput, whilst variable costs increase or decrease with throughput variations. Variable costs thus vary with throughput in dollar terms but are fixed in terms of dollars per tonne.

Between the time of compiling the operating costs for initial mine planning and subsequently producing the cashflow models in the Item 22 Economic Analysis, some adjustments and indirect operating costs were identified. These were included in the cashflow model and their impact cross-checked against the original pit optimisation results (Item 15.4.2).

21.1.3 Metal costs

Updated metal costs for concentrate treatment, refining and freight were adopted from figures provided by the Company's internal metals marketing team.

21.1.4 Estimate status and accuracy

During the design engineering and costing process a workshop was convened to assess the appropriate contingency allowances for the Project, including:

- assessment of the proportion of mechanical equipment, as a percentage of the whole which had been quoted, or using recent FQM and engineer data base pricing
- assessment of engineering maturity regarding risk of quantity growth
- assessment of contractor unit pricing, and recent FQM self-execution personnel for construction tasks

The capital cost estimates developed by Lycopodium (Lycopodium, June 2025) are stated to be accurate to +20% / -10%. Escalation was excluded from the estimate, as were all duties and taxes. All costs were presented in constant Q1 2025 dollars.

21.2 Capital cost estimation

21.2.1 Project development capital costs

Table 21-1 lists the Lycopodium development capital cost estimate by discipline, inclusive of labour costs and varying contingency allowances. Table 21-2 summarises the corresponding total capital cost estimate according to the cost centres described below.

Table 21-1 Capital cost estimate by discipline (Source: Lycopodium)

Discipline	Supply $'000	Installation $'000	Distribution $'000	Ancillary $'000	Freight $'000	Contingency %	Contingency $'000	Total $'000
Architectural	$126,428	$42,913	$350		$2,403	12%	$20,729	$192,823
Concrete	$31,831	$64,380			$78	10%	$9,914	$106,203
Electrical	$316,835	$53,862		$80,514	$15,012	14%	$67,215	$533,438
Instrumentation	$29,796	$4,260			$1,446	16%	$5,611	$41,114
Platework	$33,486	$7,960			$9,111	12%	$6,067	$56,624
Mechanical	$326,222	$49,077			$60,347	11%	$46,625	$482,270
Piping	$152,393	$63,799			$27,111	16%	$39,625	$282,928
Rail	$6,523	$437			$978	13%	$1,032	$8,970
Steelwork	$68,532	$37,084			$46,182	11%	$16,570	$168,368
Owner's costs	$615,779	$27,890		$761,808	$30,408	1%	$10,553	$1,446,438
Earthworks	$1,525	$93,538				12%	$11,408	$106,471
EPCM	$24,231			$307,228		25%	$81,430	$412,889
General	$9	$37	$99,102			15%	$14,872	$114,020
TOTAL	**$1,733,591**	**$445,237**	**$99,452**	**$1,149,550**	**$193,075**	**9%**	**$331,651**	**$3,952,556**

Table 21-2 Summary capital cost estimate by cost centre (Source: Lycopodium)

Cost centre	Materials $M	Direct labour $M	Direct equipment $M	Ancillaries and distributables $M	Freight $M	Contingency $M	Project Total $M
Mining	$684.6	$41.8	$26.4	$669.4	$39.0	$15.6	$1,476.8
Processing plant	$594.1	$185.7	$58.3	$0.0	$118.2	$121.2	$1,077.5
Rail	$4.8	$4.6	$0.5	$0.0	$1.7	$1.2	$12.8
Infrastructure	$405.8	$89.2	$35.1	$80.5	$33.2	$83.2	$727.0
Indirects	$44.2	$3.1	$0.6	$499.1	$1.0	$110.5	$658.5
TOTAL	**$1,733.6**	**$324.3**	**$121.0**	**$1,249.0**	**$193.1**	**$331.7**	**$3,952.6**

21.2.2 Mining capital

Mining pre-strip capital

Table 21-3 lists a Lycopodium (preliminary) estimated amount of $687.3 M for mine establishment. This figure was subsequently superseded by FQM to reflect revised unit mining costs for the three-and-a-half-year pre-strip period, and as listed in Table 21-4 as $764.1 M.

Table 21-3 Itemised mining capital cost estimate (Source: Lycopodium)

Cost centre	Supply Cost $'000	Labour Hours $'000	Direct Labour $'000	Direct Equipment $'000	Construction Distributables $'000	Project Ancillary $'000	Freight Cost $'000	Subtotal Cost $'000	Contingency Cost $'000	Escalation Cost $'000	Taxation & Duties $'000	Project Total $'000
Mining												
Mine mobile equipment												
Primary mining fleet	$558,724	$516	$13,945	$13,945			$30,408	**$617,022**				$617,022
Ancillary mining fleet	$38,369							**$38,369**				$38,369
Mining support	$18,686							**$18,686**				$18,686
subtotal	**$615,779**	**$516**	**$13,945**	**$13,945**			**$30,408**	**$674,078**				**$674,078**
Mining general	$11,650	$89	$2,241	$180			$722	**$14,792**	$2,029			$16,822
Mine establishment	$11,573	$79	$1,465	$2,484		$669,400	$124	**$685,046**	$2,229			$687,275
Mining facilities	$37,786	$903	$23,176	$9,730			$7,316	**$78,009**	$9,886			$87,895
Mine mobile equipment - electrical	$7,777	$39	$984	$79			$445	**$9,284**	$1,433			$10,717
subtotal	**$68,786**	**$1,111**	**$27,865**	**$12,473**		**$669,400**	**$8,608**	**$787,132**	**$15,577**			**$802,709**
Total mining	**$684,565**	**$1,627**	**$41,810**	**$26,418**		**$669,400**	**$39,016**	**$1,461,209**	**$15,577**			**$1,476,786**

Table 21-4 Mining pre-strip cost estimate

	UNITS	Year 1	Year 2	Year 3	Year 4	TOTAL
Mining pre-strip ore and waste						
Mined tonnage	Mt	19.3	94.3	146.7	161.7	**422.1**
Unit mining cost	$/t	$3.01	$1.91	$1.75	$1.61	**$1.79**
LCC	$'000	$11,721	$49,243	$79,392	$89,400	**$229,756**
Diesel	$'000	$8,254	$43,065	$55,462	$53,869	**$160,649**
Electricity	$'000	$932	$4,925	$7,375	$7,786	**$21,019**
Tyres	$'000	$2,656	$11,549	$14,544	$14,140	**$42,888**
Drill & blast	$'000	$6,202	$30,501	$46,843	$48,309	**$131,855**
Labour	$'000	$17,511	$29,377	$41,787	$37,433	**$126,108**
Equipment hire	$'000	$4,000	$4,000	$3,986	$3,430	**$15,416**
Other mining costs	$'000	$7,000	$7,000	$6,975	$6,003	**$26,978**
Total	**$'000**	**$58,275**	**$179,660**	**$256,364**	**$260,370**	**$754,669**

Mining mobile equipment capital

FQM mining engineers developed the mining equipment related capital costs listed in Table 21-6 founded on the scheduled LOM mining plan, the derived primary equipment requirements (Table 21-5), and from reference quotes provided by OEMs (original equipment manufacturers).

The FQM estimated capital costs for this equipment, incurred over the four year period prior to first ore processing, is listed in Table 21-6 as $672.2M as opposed to the $674.1M in the Lycopodium compilation in Table 21-3. In the Project economic analysis (Item 22), the development capital mining equipment purchases are carried over into Year 5, hence the $749.8 M figure shown in Table 21-6.

Table 21-5 Mining mobile equipment requirements

	UNITS	Year 1	Year 2	Year 3	Year 4	Year 5
Shovels						
Rope shovel	#	1	2.25	4	5	5
Face shovel	#	0	1	1	1	2
Loaders						
Front end loader	#	1	1	1	1	1
Trucks						
Large dump trucks	#	9	25	32	36	39
Small dump trucks	#	6	8	8	8	8
Drills						
Large rig	#	1	2	4	4	4
Small rig	#	1	1	2	3	5
Pre-split	#	1	2	4	4	4
Major ancillaries						
Dozer	#	1	5	8	9	11
Grader	#	1	3	4	5	6
Water cart	#	1	3	5	6	7
Wheel dozer	#	1	2	4	4	5

Table 21-6 Mining mobile equipment cost estimate

	UNITS	Year 1	Year 2	Year 3	Year 4	S'TOTAL	Year 5	TOTAL
Shovels								
Rope shovel	$'000	$39,922	$79,844	$39,922	$39,922	**$199,610**		**$199,610**
Face shovel	$'000		$17,871			**$17,871**	$19,235	**$37,106**
Loaders								
Front end loader	$'000	$14,321				**$14,321**	$15,415	**$29,736**
Trucks								
Large dump trucks	$'000	$69,204	$123,029	$53,825	$30,757	**$276,815**	$24,829	**$301,644**
Small dump trucks	$'000	$21,000	$7,000			**$28,000**		**$28,000**
Drills								
Large rig	$'000	$7,113	$7,113	$14,226		**$28,451**		**$28,451**
Small rig	$'000	$5,714		$5,714	$5,714	**$17,141**	$12,299	**$29,440**
Pre-split	$'000	$1,866	$1,866	$3,731		**$7,463**		**$7,463**
Major ancillaries								
Dozer	$'000	$1,400	$5,600	$4,200	$1,400	**$12,600**	$3,014	**$15,614**
Grader	$'000	$850	$1,700	$850	$850	**$4,250**	$915	**$5,165**
Water cart	$'000	$3,500	$1,750	$3,500	$1,750	**$10,500**	$1,884	**$12,384**
Wheel dozer	$'000	$1,400	$1,400	$2,800		**$5,600**	$1,507	**$7,107**
Ancillaries/support equipment								
Hydraulic Excavator	$'000	$2,850				**$2,850**		**$2,850**
Hydraulic hammer/imp	$'000	$144			$144	**$288**		**$288**
Backhoe/loader	$'000	$129				**$129**		**$129**
Transport truck	$'000		$5,633	$5,633		**$11,267**		**$11,267**
Cable reeler	$'000	$240	$481	$481		**$1,202**		**$1,202**
Lighting plant	$'000	$44	$44	$44		**$132**		**$132**
Vibratory compactor	$'000	$367	$183	$183		**$734**		**$734**
Crawler crane	$'000		$1,586			**$1,586**		**$1,586**
80t rough terrain crane	$'000					**$0**		**$0**
Rough terrain 5t forklif	$'000		$62	$62		**$124**		**$124**
Rough terrain 16.3t for	$'000	$255		$255	$255	**$765**		**$765**
30t tyre handling forkli	$'000	$321				**$321**		**$321**
Fuel/lube truck	$'000	$3,248	$1,624			**$4,872**		**$4,872**
Fuel/lube truck (small)	$'000	$231		$231	$231	**$693**		**$693**
Mechanics/welding tru	$'000	$307	$307	$307		**$920**		**$920**
Aerial boom truck	$'000	$313	$313	$313		**$940**		**$940**
Utility truck w/crane	$'000	$136	$136	$136		**$408**		**$408**
Tyre handler (3)	$'000	$225	$225	$225		**$675**		**$675**
4x4 diesel pickup; sing	$'000	$87	$87	$87		**$260**		**$260**
4x4 diesel pickup; dual	$'000	$151	$151	$151		**$452**		**$452**
Crew bus; diesel (30 s	$'000	$1,037	$1,037	$2,075		**$4,150**		**$4,150**
Systems								
Geotechnical monitori	$'000	$1,800				**$1,800**		**$1,800**
Trolley	$'000		$900	$3,600	$3,600	**$8,100**	$3,875	**$11,975**
Mining software	$'000	$1,000				**$1,000**		**$1,000**
Dispatch system	$'000	$3,836				**$3,836**		**$3,836**
Others								
Geology	$'000	$2,600				**$2,600**		**$2,600**
TOTAL	**$'000**	**$185,612**	**$259,942**	**$142,551**	**$84,623**	**$672,728**	**$82,971**	**$755,699**

Adjusted mining development capital

After finalising the pre-strip mining and mobile equipment cost estimates, some further additions were identified for inclusion in the Project economic analysis (Item 22). These adjustments are listed in Table 21-7, and totalled alongside the corresponding (and superseded) Lycopodium compilation.

Table 21-7 Adjusted mining development capital cost estimate

	UNITS	Year 1	Year 2	Year 3	Year 4	Year 5	TOTAL
Mining pre-strip ore and waste	$'000	$58,275	$179,660	$256,364	$260,370		**$754,669**
Mining - general	$'000	$4,132	$5,786	$3,173	$1,884	$1,847	**$16,822**
Mine establishment	$'000	$4,390	$6,149	$3,372	$2,002	$1,963	**$17,875**
Mining facilities	$'000	$21,588	$30,234	$16,580	$9,842	$9,650	**$87,895**
Mining equipment and ancillaries - startup	$'000	$185,612	$259,942	$142,551	$84,623	$82,971	**$755,699**
Mining fleet component change-outs	$'000	$1,210	$12,517	$29,771	$68,776	$0	**$112,274**
Mining electrical equipment	$'000	$2,632	$3,686	$2,022	$1,200	$1,177	**$10,717**
Total	**$'000**	**$277,840**	**$497,974**	**$453,832**	**$428,697**	**$97,608**	**$1,755,950**

21.2.3 Processing capital

The Lycopodium estimated capital cost for the processing plant (Table 21-8) accounts for:

- plant site earthworks and roads
- primary crushing, crushed ore stockpiling, stockpile feed conveying and stockpile reclaim
- milling, pebble crushing and grinding media
- rougher and cleaner flotation, plus concentrate regrind
- copper concentrate thickening, filtration and loadout
- molybdenum (prefloat) thickening and flotation
- molybdenum concentrate filtration, drying and bagging
- tails thickening and pumping
- tailings dam
- all reagents
- plant services including process water and air

Table 21-8 Processing plant capital cost estimate (source: Lycopodium)

Cost centre	Supply Cost $'000	Labour Hours $'000	Direct Labour $'000	Direct Equipment $'000	Construction Distributables $'000	Project Ancillary $'000	Freight Cost $'000	Subtotal Cost $'000	Contingency Cost $'000	Escalation Cost $'000	Taxation & Duties $'000	Project Total $'000
Processing plant												
Plant site general	$33,994	$602	$12,762	$11,312			$2,113	**$60,181**	$8,791			**$68,972**
Crushing and stockpiling	$38,561	$502	$13,439	$1,365			$5,263	**$58,628**	$7,205			**$65,833**
Stockpile reclaim and milling	$263,557	$2,491	$85,759	$8,084			$56,621	**$414,021**	$49,927			**$463,948**
Flotation	$113,899	$943	$27,124	$3,431			$25,999	**$170,453**	$21,473			**$191,927**
Concentrate handling	$35,004	$284	$7,649	$999			$5,607	**$49,259**	$6,329			**$55,588**
Tailings disposal	$68,952	$1,496	$27,225	$31,599			$15,351	**$143,128**	$19,089			**$162,217**
Metals recovery	$17,893	$209	$5,572	$694			$3,758	**$27,918**	$3,648			**$31,566**
Reagents	$9,826	$122	$3,139	$353			$1,039	**$14,357**	$2,097			**$16,453**
Plant services	$12,422	$115	$2,993	$483			$2,441	**$18,339**	$2,640			**$20,979**
Total processing plant	**$594,109**	**$6,764**	**$185,663**	**$58,320**	**$0**	**$0**	**$118,192**	**$956,284**	**$121,198**	**$0**	**$0**	**$1,077,482**

21.2.4 Rail capital

The Lycopodium estimated capital cost for rail (Table 21-9) accounts for development of the earthworks embankment on the rail spur, the locomotive site maintenance and fuelling facilities, in addition to the copper concentrate storage and rotainer loading facility, plus the general goods hardstand area alongside the rail spur line.

Otherwise, no other capital costs are allowed for rail infrastructure between the site and the Mejillones port, nor for the rolling stock. This exclusion is on basis that a proposed operational contract between FQM and a rail operations joint venture will entirely fund all such capital costs as part of a long term contract for the delivery of concentrate to the port.

Table 21-9 Rail capital cost estimate (source: Lycopodium)

Cost centre	Supply Cost $'000	Labour Hours $'000	Direct Labour $'000	Direct Equipment $'000	Construction Distributables $'000	Project Ancillary $'000	Freight Cost $'000	Subtotal Cost $'000	Contingency Cost $'000	Escalation Cost $'000	Taxation & Duties $'000	Project Total $'000
Rail												
Concentrate handling	$4,068	$132	$3,910	$405			$1,539	**$9,921**	$992			**$10,913**
Site utilities and services	$759	$25	$654	$112			$147	**$1,672**	$191			**$1,862**
Total rail	**$4,828**	**$156**	**$4,564**	**$516**	**$0**	**$0**	**$1,686**	**$11,593**	**$1,183**	**$0**	**$0**	**$12,776**

21.2.5 Infrastructure capital

The Lycopodium cost estimate for Project infrastructure (Table 21-10) accounts for the development of the following:

- the 345 kV overhead power line from La Puna to the Project switchyard
  - based on the preliminary design and pricing (provided by Tecnolatina)
  - with more detailed assessment, including development of the construction access road (by Fluor)
  - additional detailed design of the HV switchyard , incoporating power factor correction facilities (by Process E&I)
- the layout and facilities at the Salta Operations Centre (by Fluor)
- the layout and facilities at the Pocitos truck stop (by Fluor)
- off-site roads including the public road redirection to the site in addition to route transport improvements (by Fluor)
- the layout and facilities at the proposed airstrip, specifically the runway and land-side air services requirements (by Fluor, with technical support from Flytec in Salta)
- design of the water and brine catchments, including sizing of bore pumps, catchment pipelines, catchment storage tank, overland gravity line to the Project water tanks, easements overland and across the salar, together with the overhead power line distribution systems and substations (by Fluor and Process E&I, respectively)
  - this work was supported by the water catchment assessments by Piteau and vendor pricing for tanks, pumps and piping costs from Enviropipes (in Perth)
- design of the village facilities including accommodation blocks, kitchen, dining, laundry, recreation and central services (by Fluor and supported by vendor pricing from Argentine and Chilean suppliers)

Table 21-10 Infrastructure capital cost estimate (source: Lycopodium)

Cost centre	Supply Cost $'000	Labour Hours $'000	Direct Labour $'000	Direct Equipment $'000	Construction Distributables $'000	Project Ancillary $'000	Freight Cost $'000	Subtotal Cost $'000	Contingency Cost $'000	Escalation Cost $'000	Taxation & Duties $'000	Project Total $'000
Infrastructure												
Infrastructure - general (construction)	$18,874	$268	$6,862	$3,065			$3,197	**$31,998**	$4,049			**$36,047**
Infrastructure - general (roads/rail)	$9,096	$259	$4,648	$9,124			$1,626	**$24,494**	$2,996			**$27,490**
Environmental	$413	$12	$351	$41			$116	**$921**	$95			**$1,017**
Utilities and services	$148,166	$1,554	$38,310	$9,706			$24,367	**$220,548**	$30,003			**$250,551**
Power supply	$99,106	$93	$2,387	$201		$80,514	$1,446	**$183,654**	$23,918			**$207,572**
Buildings - admin and security	$14,595	$159	$3,797	$1,359			$304	**$20,055**	$2,599			**$22,653**
Buildings - services	$22,895	$336	$7,912	$5,360			$1,597	**$37,764**	$4,538			**$42,302**
Village	$91,083	$1,005	$24,844	$6,260			$441	**$122,628**	$14,735			**$137,363**
Golf course	$1,614	$2	$58	$5			$97	**$1,774**	$274			**$2,047**
Total infrastructure	**$405,841**	**$3,686**	**$89,169**	**$35,120**	**$0**	**$80,514**	**$33,191**	**$643,836**	**$83,207**	**$0**	**$0**	**$727,043**

21.2.6 Indirect capital

Indirect capital costs cover the owner's costs (i.e., FQM or the Company), EPCM (for engineering, procurement, construction and management) costs, spares and first fill costs, in addition to the cost of temporary facilities and construction accommodation (Table 21-11 and Table 21-12). Owner's costs also include the construction management team, construction equipment costs, the Perth office owner's team, pre-operational readiness costs, the commissioning management team costs, IT costs and insurances. For this estimate, the relevant costs have been compiled by FQM.

Construction and management costs were estimated also by FQM, on a self-perform basis referencing data from previous Company projects.

Spares were estimated as a percentage of the mechanical equipment supply value. First fill costs were calculated according to the first fill requirements for oils and greases, recognising the reagent storage capacity on site and the cost of reagents in the supply chain.

Table 21-11 Indirect capital cost estimate (source: Lycopodium)

Cost centre	Supply Cost $'000	Labour Hours $'000	Direct Labour $'000	Direct Equipment $'000	Construction Distributables $'000	Project Ancillary $'000	Freight Cost $'000	Subtotal Cost $'000	Contingency Cost $'000	Escalation Cost $'000	Taxation & Duties $'000	Project Total $'000
Indirects												
Owners costs						$256,106		**$256,106**	$67,425			**$323,531**
EPCM	$38,503					$103,057	$39	**$141,598**	$21,406			**$163,005**
Contractor indirects	$73	$5,993	$57	$5			$4	**$139**	$21			**$160**
Spares						$26,241		**$26,241**	$3,936			**$30,177**
First fills						$14,232		**$14,232**	$1,423			**$15,656**
Temporary facilities and services	$5,672	$119	$3,000	$595	$19,043		$948	**$29,258**	$4,213			**$33,471**
Construction accommodation					$80,408			**$80,408**	$12,061			**$92,470**
Total indirects	**$44,248**	**$6,112**	**$3,058**	**$600**	**$99,452**	**$399,636**	**$990**	**$547,983**	**$110,486**	**$0**	**$0**	**$658,469**

Table 21-12 Indirect capital cost estimate details (source: Lycopodium)

Cost centre	Subtotal cost $'000	Contingency rate %	Contingency cost $'000	Total cost $'000
Pre-operations				
Commissioning team	$21,605	15%	$3,241	$24,845
Construction team	$118,985	15%	$17,848	$136,833
Construction equipment	$51,428	15%	$7,714	$59,142
Labour rate contingency			$31,606	$31,606
FQM Perth team	$12,154	15%	$1,823	$13,977
Operational readiness; G&A	$12,154	10%	$1,215	$13,370
Operational readiness; processing	$14,984	10%	$1,498	$16,482
IT	$9,296	10%	$930	$10,226
Insurances	$15,500	10%	$1,550	$17,050
Subtotal	**$256,106**		**$67,425**	**$323,531**
Spares and first fills				
Spares	$26,241	15%	$3,936	$30,177
First fills	$14,232	10%	$1,423	$15,656
Subtotal	**$40,473**		**$5,359**	**$45,833**
Other				
EPCM	$141,598	15%	$21,406	**$163,005**
Contractor indirects	$139	15%	$21	**$160**
Temporary facilities and services	$29,258	14%	$4,213	**$33,471**
Construction accommodation	$80,408	15%	$12,061	**$92,470**
Subtotal	**$251,403**	15%	**$37,702**	**$289,105**
TOTAL	**$547,982**		**$110,486**	**$658,469**

21.3 Expansion capital costs

An internal capital cost estimate has been completed for expansion of the Project from 40 Mtpa to 60 Mtpa processing, as itemised in Table 21-13. The 60 Mtpa expansion estimate allows for:

- increased crushing, grinding and rougher flotation facilities associated with a 50% increase in plant feed
- tails thickening and pumping increases by 50% associated with the production of more tails
- additional process water handling facilities

Other parts of the processing plant including regrind, cleaner flotation and reagents remain unchanged given there is a corresponding drop in annual feed grade with the increase in throughput.

Table 21-13 Expansion capital cost estimate

	UNITS	
3.0 - Plant Site - General	$'000	$35,536
3.1 - Crushing & Stockpiling	$'000	$58,221
3.2 - Stockpile Reclaim & Milling	$'000	$305,197
3.3 - Flotation	$'000	$53,973
3.6 - Tailings Disposal	$'000	$57,247
3.8 - Reagents	$'000	$10,688
3.9 - Plant Services	$'000	$7,865
8.0 - Infrastructure - General	$'000	$12,902
8.1 - Infrastructure - General	$'000	$52,595
8.3 - Utilities & Services	$'000	$149,676
8.4 - Power Supply	$'000	$3,418
8.5 - Buildings - Admin & Security	$'000	$197
9.2 - EPCM	$'000	$99,450
9.4 - Contractor Indirects	$'000	$19,681
9.5 - Spares	$'000	$13,832
9.6 - First Fill	$'000	$4,910
9.8 - Temporary Facilities & Services	$'000	$19,465
9.9 - Construction Accommodation	$'000	$8,624
	$'000	**$913,476**

In relation to site infrastructure:

- all mining infrastructure remains the same
- The camp is sized for the peak of construction, pre-strip and operational ramp-up so will not need to be augmented
- There is no change in roads, power line or for the air strip

In addition to the $913.5 M listed in Table 21-13, there is an additional estimated provision of $105.2 M for purchasing additional mining fleet. The timeframe for expenditure of these expansion costs is tabled in Item 22.2.3.

21.4 Sustaining capital costs

Sustaining costs in relation to ongoing mining equipment purchases were estimated explicitly according to the scheduled LOM mining plan, the derived equipment requirements (Table 21-14), and from reference quotes provided by OEMs. This was completed as an extension of the estimate for related costs during the four-year Project development period. Table 21-15 lists the annual mining equipment sustaining cost provisions. Similarly, Table 21-16 lists the corresponding cost provisions for the change-out of mining equipment (maintenance) components.

Annual sustaining costs for the Project in general were estimated as a simple allowance of 4.5% of the corresponding annual process operating costs as listed in the cashflow model of Item 22.

Table 21-14 Mining mobile equipment requirements

	UNITS	Shovels	Loaders	Trucks	Drills	Major ancillaries
Yr 5	#	7	1	47	13	29
Yr 6	#	7	1	47	13	29
Yr 7	#	7	1	47	13	29
Yr 8	#	7	1	48	14	29
Yr 9	#	7	1	56	14	29
Yr 10	#	7	1	56	14	29
Yr 11	#	7	1	58	14	29
Yr 12	#	7	1	58	14	29
Yr 13	#	7	1	58	14	29
Yr 14	#	7	1	56	14	29
Yr 15	#	7	1	56	14	29
Yr 16	#	7	1	54	14	25
Yr 17	#	6	1	49	13	23
Yr 18	#	5	1	44	11	20
Yr 19	#	4	1	42	10	18
Yr 20	#	4	1	42	10	18
Yr 21	#	4	1	42	9.5	18
Yr 22	#	4	1	42	9	18
Yr 23	#	3	1	38	9	16
Yr 24	#	3	1	38	9	16
Yr 25	#	3	1	38	9	16
Yr 26	#	3	1	39	9	16
Yr 27	#	3	1	42	8.5	16
Yr 28	#	3	1	43	8	16
Yr 29	#	3	1	45	8	16
Yr 30	#	3	1	45	8	16
Yr 31	#	3		34	7	13
Yr 32	#	2		29	6	11
Yr 33	#	2		29	6	11
Yr 34	#	2		29	6	11
Yr 35	#	2		29	6	11
Yr 36	#	2		31	6	10
Yr 37	#	2		32	6	10
Yr 38	#	2		32	6	10
Yr 39	#	2		19	4	8
Yr 40	#	2		19	4	8
Yr 41	#	2		19	3	8
Yr 42	#	2		8		8
Yr 43	#	2		8		8
Yr 44	#	2		8		8
Yr 45	#	2		8		8
Yr 46	#	2		8		8
Yr 47	#	2		8		8
Yr 48	#	2		8		8
Yr 49	#	2		8		8
Yr 50	#	2		8		8
Yr 51	#	2		8		8
Yr 52	#	2		8		8
Yr 53	#	2		8		8
Yr 54	#	2		8		8

Table 21-15 Mining equipment, annual new and replacement costs summary

	UNITS	Shovels	Loaders	Trucks	Drills	Major ancillaries	Other ancillaries	Systems	Others	Total
Yr 5	$'000	$17,871	$14,321	$23,068	$11,427	$6,800	$5,884	$3,600		**$82,971**
Yr 6	$'000				$3,731		$5,756	$2,700		**$12,187**
Yr 7	$'000				$3,731		$5,911	$4,500		**$14,143**
Yr 8	$'000			$7,689	$5,714		$5,854	$4,050		**$23,307**
Yr 9	$'000			$61,514		$1,400	$5,719	$5,400		**$74,033**
Yr 10	$'000				$3,731	$10,300	$5,687	$2,700		**$22,418**
Yr 11	$'000			$7,000	$3,731	$10,800	$5,671	$4,050		**$31,252**
Yr 12	$'000					$6,800	$5,655	$6,750		**$19,205**
Yr 13	$'000		$14,321		$28,451	$6,800	$5,765	$10,800		**$66,138**
Yr 14	$'000				$9,445		$5,763	$4,500		**$19,708**
Yr 15	$'000	$17,871			$9,445		$5,726	$11,051		**$44,092**
Yr 16	$'000			$24,500	$5,714		$5,405	$6,300		**$41,919**
Yr 17	$'000				$11,427		$4,521	$2,700		**$18,648**
Yr 18	$'000				$1,866	$1,400	$3,972	$1,350		**$8,588**
Yr 19	$'000				$3,731	$7,700	$3,586	$3,150		**$18,168**
Yr 20	$'000					$1,700	$3,623	$2,700		**$8,023**
Yr 21	$'000			$76,893		$2,600	$3,623	$5,400		**$88,516**
Yr 22	$'000				$1,866	$5,600	$3,618	$1,800		**$12,884**
Yr 23	$'000		$14,321	$30,757	$3,731	$3,150	$3,102	$5,400		**$60,462**
Yr 24	$'000	$39,922		$15,379			$3,083	$900		**$59,283**
Yr 25	$'000			$11,189			$2,987	$7,200		**$21,377**
Yr 26	$'000			$30,757	$7,579	$1,400	$3,094	$900		**$43,730**
Yr 27	$'000			$26,568	$3,731	$8,550	$3,102	$2,700		**$44,651**
Yr 28	$'000			$7,689	$5,714		$3,094			**$16,497**
Yr 29	$'000	$39,922		$15,379	$5,714	$2,600	$2,930	$2,700		**$69,244**
Yr 30	$'000				$7,579		$2,851	$1,350		**$11,780**
Yr 31	$'000			$7,000	$1,866		$1,212			**$10,077**
Yr 32	$'000					$2,800	$980	$900		**$4,680**
Yr 33	$'000						$1,019	$2,250		**$3,269**
Yr 34	$'000				$1,866	$1,400	$1,004			**$4,269**
Yr 35	$'000				$1,866	$6,300	$993	$1,800		**$10,958**
Yr 36	$'000					$2,250	$917	$1,350		**$4,517**
Yr 37	$'000			$7,689			$917			**$8,606**
Yr 38	$'000					$850	$887	$900		**$2,637**
Yr 39	$'000						$711			**$711**
Yr 40	$'000						$683			**$683**
Yr 41	$'000						$658			**$658**
Yr 42	$'000					$1,400	$610			**$2,010**
Yr 43	$'000					$4,350	$611			**$4,961**
Yr 44	$'000					$1,400	$610			**$2,010**
Yr 45	$'000			$30,757			$610			**$31,367**
Yr 46	$'000	$39,922		$7,689		$1,400	$610			**$49,621**
Yr 47	$'000					$2,250	$611			**$2,861**
Yr 48	$'000						$610			**$610**
Yr 49	$'000						$610			**$610**
Yr 50	$'000						$610			**$610**
Yr 51	$'000						$611			**$611**
Yr 52	$'000						$610			**$610**
Yr 53	$'000						$610			**$610**
Yr 54	$'000						$669			**$669**
TOTAL	**$'000**	**$155,508**	**$42,964**	**$391,519**	**$143,656**	**$102,000**	**$133,953**	**$111,851**	**$0**	**$1,081,451**

Table 21-16 Mining equipment components, annual change-out costs summary

	UNITS	Shovels	Loaders	Trucks	Drills	Major ancillaries	Total
Yr 5	$'000	$27,017	$2,003	$1,192	$4,701	$7,340	**$42,253**
Yr 6	$'000	$21,688	$3,647	$20,748	$5,534	$9,013	**$60,629**
Yr 7	$'000	$52,517	$3,647	$20,194	$2,904	$10,942	**$90,205**
Yr 8	$'000	$15,171	$8,688	$28,759	$5,643	$8,065	**$66,326**
Yr 9	$'000	$50,473	$5,908	$8,557	$4,199	$5,887	**$75,023**
Yr 10	$'000	$58,154	$7,815	$16,575	$5,725	$7,687	**$95,956**
Yr 11	$'000	$13,581	$2,999	$9,010	$5,477	$5,255	**$36,322**
Yr 12	$'000	$35,152	$1,918	$41,785	$3,298	$10,387	**$92,539**
Yr 13	$'000	$27,786	$2,650	$1,968	$3,334	$9,601	**$45,338**
Yr 14	$'000	$36,922	$3,647	$19,039	$2,016	$9,939	**$71,563**
Yr 15	$'000	$38,404	$6,905	$17,010	$3,442	$13,346	**$79,107**
Yr 16	$'000	$27,262	$8,038	$18,635	$7,580	$6,804	**$68,319**
Yr 17	$'000	$14,127	$2,003	$14,231	$2,559	$5,065	**$37,984**
Yr 18	$'000	$11,351	$4,237	$19,659	$3,173	$3,892	**$42,312**
Yr 19	$'000	$4,559	$4,238	$9,919	$2,866	$2,680	**$24,262**
Yr 20	$'000	$5,956	$8,686	$10,950	$5,453	$5,290	**$36,335**
Yr 21	$'000	$3,768	$2,224	$13,086	$3,083	$6,547	**$28,709**
Yr 22	$'000	$7,046	$2,336	$7,610	$2,559	$2,722	**$22,273**
Yr 23	$'000	$1,792	$2,224	$6,535	$2,547	$5,915	**$19,014**
Yr 24	$'000	$2,074	$1,750	$16,655	$2,972	$6,231	**$29,683**
Yr 25	$'000	$6,314	$1,745	$4,408	$1,735	$4,985	**$19,187**
Yr 26	$'000	$2,233	$1,745	$10,169	$927	$3,226	**$18,301**
Yr 27	$'000	$2,940	$5,003	$12,018	$1,156	$4,092	**$25,209**
Yr 28	$'000	$1,198	$5,016	$15,207	$605	$4,956	**$26,982**
Yr 29	$'000	$3,070	$788	$13,333	$1,956	$5,676	**$24,824**
Yr 30	$'000	$2,103	$1,436	$13,751	$568	$4,375	**$22,233**
Yr 31	$'000	$6,444	$1,436	$5,263	$1,991	$4,718	**$19,851**
Yr 32	$'000	$4,526	$1,440	$16,277	$1,961	$2,919	**$27,123**
Yr 33	$'000	$1,471	$4,693	$8,631	$1,541	$1,996	**$18,332**
Yr 34	$'000	$18,976	$788	$12,503	$1,586	$2,270	**$36,124**
Yr 35	$'000	$1,636	$788	$8,066	$2,628	$2,533	**$15,651**
Yr 36	$'000	$2,349	$0	$9,017	$1,997	$3,432	**$16,794**
Yr 37	$'000	$16,446	$0	$10,664	$1,018	$5,144	**$33,273**
Yr 38	$'000	$1,341	$0	$9,990	$568	$3,212	**$15,111**
Yr 39	$'000	$2,472	$0	$6,904	$1,275	$1,325	**$11,975**
Yr 40	$'000	$1,510	$0	$6,268	$718	$2,819	**$11,315**
Yr 41	$'000	$19,106	$0	$5,649	$0	$935	**$25,691**
Yr 42	$'000	$4,513	$0	$2,443	$0	$1,389	**$8,344**
Yr 43	$'000	$1,471	$0	$2,443	$0	$1,037	**$4,951**
Yr 44	$'000	$5,732	$0	$2,450	$0	$1,138	**$9,320**
Yr 45	$'000	$1,636	$0	$782	$0	$3,550	**$5,967**
Yr 46	$'000	$4,814	$0	$2,947	$0	$1,604	**$9,365**
Yr 47	$'000	$2,103	$0	$785	$0	$1,645	**$4,533**
Yr 48	$'000	$6,461	$0	$5,783	$0	$1,594	**$13,838**
Yr 49	$'000	$5,111	$0	$1,778	$0	$1,798	**$8,688**
Yr 50	$'000	$2,068	$0	$2,262	$0	$2,248	**$6,579**
Yr 51	$'000	$19,574	$0	$2,634	$0	$1,617	**$23,824**
Yr 52	$'000	$2,239	$0	$4,891	$0	$1,538	**$8,668**
Yr 53	$'000	$2,940	$0	$2,406	$0	$789	**$6,134**
Yr 54	$'000	$17,044	$0	$2,482	$0	$461	**$19,987**
TOTAL	**$'000**	**$624,642**	**$110,442**	**$504,319**	**$101,293**	**$221,630**	**$1,562,327**

21.5 Closure costs

GT Ingeniería (2020) produced a conceptual cost estimate as listed in Table 21-17. This was a comprehensive estimate, itemised into seventeen closure components, plus post-closure activities such as on-going monitoring at specifically identified sites across the Project area.

The estimate components accounted for dismantling and demolition of infrastructure, longer term surface water management and erosion control, and rehabilitation of the landscape.

The 2020 estimate was inflated to $71.9 M for the purposes of cashflow modelling. This figure also accounts for employee severance.

Table 21-17 Project closure cost estimate (source: GT Ingeniería)

Item	Description	Subtotal $	Proportion (%)
1	Provisional and preliminary works	$223,000	1.4%
2	CM1: Filling Taco Chico Pit	$872,036	5.3%
3	CM2: Taca Taca Open Pit	$3,666,959	22.2%
4	CM5: Tailings storage facility	$2,533,734	15.3%
5	CM3: East waste dump	$5,007,600	30.3%
6	CM4: West waste dump	$963,000	5.8%
7	CM6: Ore stockpile	$18,523	0.1%
8	CM8: Solid waste landfill	$223,904	1.4%
9	CM7: Process plant	$2,305,682	14.0%
10	CM7: Primary crushing plants	$20,000	0.1%
11	CM7: Camp	$78,830	0.5%
12	CM7: Mining services area	$52,412	0.3%
13	CM7: Power lines	$20,000	0.1%
14	CM7: Explosives magazine	$245,824	1.5%
15	CM9: Roads	$58,000	0.4%
16	CM10: Railway	$20,000	0.1%
17	Post closure activities	$206,760	1.3%
	Total direct costs	**$16,516,264**	**100.0%**
	General expenses (25%)	$4,129,066	
	Profit (10%)	$1,651,626	
	Subtotal	**$5,780,692**	
	Supervision (7%)	$1,560,787	
	Owners costs (3%)	$668,909	
	Total cost	**$2,229,696**	
	Allowance for complentary studies e.g., ARD, salt formaiton, hydrology, geotechnics (0.1%)	$24,527	
	Engineering cost (0.1%)	$24,527	
	Contingency (40%)	$9,830,282	
	Total costs	**$9,879,335**	
	Grand total cost	**$34,405,987**	
	Inflated to 2025	**$71,979,663**	

21.6 Operating costs

21.6.1 Mining costs

Mine operating costs comprising drill, blast, load and haul costs were derived by FQM mining engineers in Q3 2025. These derivations were estimated from first principles using productivity parameters for the proposed equipment fleet, simulated haul profiles related to the phased pit designs and production schedule, and from corresponding ore/waste haulage destinations.

The following commentary provides information on the estimates for specific variable and fixed mining costs. An overall summary of the estimated annual mining costs is provided in Table 21-18. A related annual summary in terms of unit mining costs is provided in Table 21-19.

Table 21-18 Estimated annual mining costs, LOM total dollars

Year	Stage	Annual mining costs						
		Drill & Blast ($M)	LCC ($M)	Diesel ($M)	Electricity ($M)	Tyres ($M)	Labour ($M)	Total ($M)
1	Pre-strip	$7.2	$11.6	$8.3	$0.9	$2.6	$17.5	**$48.1**
2		$35.5	$47.9	$39.1	$4.9	$11.4	$29.4	**$168.1**
3		$54.6	$76.4	$51.6	$7.4	$14.3	$41.9	**$246.3**
4		$64.7	$96.6	$58.8	$9.1	$16.2	$43.6	**$289.0**
5	Production	$79.8	$103.5	$72.6	$11.4	$17.7	$46.4	**$331.4**
6		$80.6	$114.5	$71.8	$11.0	$17.5	$46.4	**$341.7**
7		$80.2	$111.5	$75.7	$9.9	$17.4	$46.4	**$341.0**
8		$80.0	$105.1	$73.3	$14.0	$18.0	$46.9	**$337.4**
9		$82.1	$104.0	$79.0	$18.3	$20.7	$48.7	**$352.8**
10		$81.9	$110.5	$74.8	$15.3	$19.8	$48.7	**$351.0**
11		$83.0	$105.0	$79.3	$16.9	$20.8	$48.4	**$353.4**
12		$84.4	$110.5	$79.7	$16.6	$20.3	$48.4	**$359.9**
13		$82.2	$100.8	$77.0	$17.8	$21.0	$48.4	**$347.1**
14		$81.1	$103.2	$68.6	$20.1	$19.9	$47.9	**$340.8**
15		$82.3	$99.6	$71.9	$18.3	$20.4	$47.9	**$340.4**
16		$77.5	$98.0	$78.3	$14.8	$19.4	$46.7	**$334.7**
17		$69.9	$80.9	$61.8	$16.8	$17.4	$44.7	**$291.5**
18		$60.3	$75.3	$51.8	$16.6	$15.5	$40.3	**$259.9**
19		$52.5	$60.7	$50.8	$13.2	$14.1	$39.1	**$230.4**
20		$51.6	$65.6	$52.0	$13.9	$14.0	$39.1	**$236.2**
21		$50.9	$63.3	$52.1	$15.6	$14.5	$39.0	**$235.5**
22		$50.5	$63.5	$52.5	$15.1	$15.0	$38.9	**$235.5**
23		$45.8	$56.8	$46.8	$14.4	$13.4	$37.4	**$214.6**
24		$46.2	$61.0	$48.6	$12.4	$13.3	$37.4	**$218.8**
25		$46.9	$56.8	$48.9	$11.0	$13.1	$37.4	**$214.0**
26		$46.4	$59.0	$52.9	$12.1	$13.9	$37.7	**$222.0**
27		$46.6	$60.9	$58.1	$12.6	$14.9	$38.2	**$231.2**
28		$45.7	$60.6	$48.3	$16.4	$15.2	$38.1	**$224.2**
29		$47.1	$61.4	$49.4	$16.6	$15.6	$38.5	**$228.6**
30		$46.6	$63.2	$46.2	$22.5	$16.0	$38.4	**$232.9**
31		$36.9	$50.8	$29.9	$20.3	$11.8	$25.8	**$175.5**
32		$30.6	$48.8	$23.4	$16.1	$9.7	$24.0	**$152.6**
33		$30.0	$43.6	$29.5	$16.2	$10.2	$24.0	**$153.5**
34		$29.8	$44.9	$28.4	$17.3	$10.6	$24.0	**$155.0**
35		$30.1	$43.4	$28.9	$15.9	$10.1	$24.0	**$152.4**
36		$30.9	$45.1	$29.9	$4.7	$10.5	$24.1	**$145.3**
37		$31.3	$44.4	$32.4	$7.7	$11.3	$24.4	**$151.4**
38		$30.5	$40.5	$30.5	$13.4	$11.1	$24.4	**$150.4**
39		$16.1	$26.7	$24.4	$7.9	$6.6	$20.7	**$102.5**
40		$15.8	$29.6	$24.6	$7.8	$6.6	$20.7	**$105.1**
41		$17.2	$26.9	$19.7	$8.6	$6.6	$20.5	**$99.4**
42			$18.3	$11.4	$2.2	$3.1	$13.4	**$48.4**
43			$13.3	$11.4	$2.2	$3.1	$13.5	**$43.5**
44			$15.4	$11.4	$2.2	$3.1	$13.5	**$45.6**
45			$14.7	$11.4	$2.2	$3.1	$13.5	**$44.9**
46			$15.9	$11.3	$2.2	$3.1	$13.6	**$46.0**
47			$16.1	$11.3	$2.2	$3.1	$13.6	**$46.2**
48			$19.3	$11.3	$2.2	$3.1	$13.6	**$49.4**
49			$17.0	$11.3	$2.2	$3.1	$13.6	**$47.2**
50			$18.2	$11.3	$2.2	$3.1	$13.7	**$48.4**
51			$16.3	$11.3	$2.2	$3.1	$13.7	**$46.5**
52			$16.0	$11.3	$2.2	$3.1	$13.7	**$46.3**
53			$13.4	$11.3	$2.2	$3.1	$13.7	**$43.7**
54			$15.1	$10.7	$2.2	$2.8	$13.8	**$44.6**
Total		**$2,143.6**	**$3,041.1**	**$2,228.1**	**$580.0**	**$628.2**	**$1,681.4**	**$10,302.3**

Table 21-19 Estimated average unit mining costs

Year	Stage	MINING & RECLAIM PYHSICALS					UNIT MINING COSTS					TOTAL
		Ore (Mt)	Waste (Mt)	Total Mined (Mt)	HG/LG Reclaim (Mt)	MW Reclaim (Mt)	Drill & blast ($/t)	Load & haul ($/t)	Fixed costs ($/t)	MW reclaim ($/t)	Total costs ($/t)	COST ($M)
1	**Pre-strip**		19.3	19.3			$0.37	$1.21	$0.91		**$2.49**	**$48.1**
2			94.3	94.3			$0.38	$1.09	$0.31		**$1.78**	**$168.1**
3		0.5	146.7	147.3			$0.37	$1.02	$0.28		**$1.67**	**$246.3**
4		9.9	161.7	171.6			$0.38	$1.05	$0.25		**$1.68**	**$289.0**
5	**Production**	52.3	143.0	195.2	7.9		$0.41	$1.05	$0.24		**$1.70**	**$331.4**
6		51.5	143.8	195.2	3.5		$0.41	$1.10	$0.24		**$1.75**	**$341.7**
7		45.5	150.3	195.8	0.0		$0.41	$1.10	$0.24		**$1.74**	**$341.0**
8		42.4	152.8	195.2	3.9		$0.41	$1.08	$0.24		**$1.73**	**$337.4**
9		52.7	142.5	195.2	2.2		$0.42	$1.14	$0.25		**$1.81**	**$352.8**
10		46.7	148.5	195.2	1.1		$0.42	$1.13	$0.25		**$1.80**	**$351.0**
11		56.2	139.5	195.8			$0.42	$1.13	$0.25		**$1.81**	**$353.4**
12		69.9	125.4	195.2			$0.43	$1.16	$0.25		**$1.84**	**$359.9**
13		57.8	137.4	195.2	3.8		$0.42	$1.11	$0.25		**$1.78**	**$347.1**
14		55.0	140.2	195.2	3.7		$0.42	$1.08	$0.25		**$1.75**	**$340.8**
15		57.8	137.9	195.8	1.9		$0.42	$1.07	$0.24		**$1.74**	**$340.4**
16		63.4	116.8	180.2	6.4		$0.43	$1.17	$0.26		**$1.86**	**$334.7**
17		74.0	82.0	156.0	0.1		$0.45	$1.13	$0.29		**$1.87**	**$291.5**
18		73.8	58.0	131.8	5.3		$0.46	$1.21	$0.31		**$1.97**	**$259.9**
19		57.3	59.8	117.1	6.7		$0.45	$1.19	$0.33		**$1.97**	**$230.4**
20		51.9	64.9	116.8	8.3		$0.44	$1.25	$0.34		**$2.02**	**$236.2**
21		48.5	68.3	116.8	8.3		$0.44	$1.25	$0.33		**$2.02**	**$235.5**
22		43.6	73.0	116.6	8.3		$0.43	$1.25	$0.33		**$2.02**	**$235.5**
23		55.5	43.3	98.8	8.3		$0.46	$1.33	$0.38		**$2.17**	**$214.6**
24		56.6	41.9	98.5	7.9		$0.47	$1.37	$0.38		**$2.22**	**$218.8**
25		60.9	37.6	98.5	4.6		$0.48	$1.32	$0.38		**$2.17**	**$214.0**
26		60.3	38.2	98.5	8.3		$0.47	$1.40	$0.38		**$2.25**	**$222.0**
27		62.6	36.2	98.8	8.3		$0.47	$1.48	$0.39		**$2.34**	**$231.2**
28		59.2	39.3	98.5	8.3		$0.46	$1.43	$0.39		**$2.28**	**$224.2**
29		68.4	30.1	98.5	2.6		$0.48	$1.45	$0.39		**$2.32**	**$228.6**
30		67.0	31.5	98.4			$0.47	$1.50	$0.39		**$2.37**	**$232.9**
31		56.0	19.4	75.4	4.1		$0.49	$1.50	$0.34		**$2.33**	**$175.5**
32		49.5	10.6	60.2	4.1		$0.51	$1.63	$0.40		**$2.54**	**$152.6**
33		47.5	12.7	60.2	6.7		$0.50	$1.65	$0.40		**$2.55**	**$153.5**
34		45.8	14.4	60.2	5.7		$0.50	$1.68	$0.40		**$2.58**	**$155.0**
35		47.0	13.3	60.3	4.8		$0.50	$1.63	$0.40		**$2.53**	**$152.4**
36		50.9	9.2	60.2			$0.51	$1.50	$0.40		**$2.41**	**$145.3**
37		53.1	7.1	60.2			$0.52	$1.59	$0.41		**$2.52**	**$151.4**
38		52.7	5.5	58.2			$0.52	$1.64	$0.42		**$2.58**	**$150.4**
39		27.8	2.8	30.6	16.1		$0.53	$2.15	$0.68		**$3.35**	**$102.5**
40		27.7	2.5	30.2	14.6		$0.52	$2.27	$0.68		**$3.48**	**$105.1**
41		30.9	1.5	32.4	10.8		$0.53	$1.91	$0.63		**$3.07**	**$99.4**
42					40.0			$0.87	$0.34		**$1.21**	**$48.4**
43					40.1			$0.75	$0.34		**$1.08**	**$43.5**
44					40.0			$0.80	$0.34		**$1.14**	**$45.6**
45					40.0			$0.78	$0.34		**$1.12**	**$44.9**
46					2.0	38.0				$1.15	**$1.15**	**$46.0**
47						40.1				$1.15	**$1.15**	**$46.2**
48						40.0				$1.24	**$1.24**	**$49.4**
49						40.0				$1.18	**$1.18**	**$47.2**
50						40.0				$1.21	**$1.21**	**$48.4**
51						40.1				$1.16	**$1.16**	**$46.5**
52						40.0				$1.16	**$1.16**	**$46.3**
53						40.0				$1.09	**$1.09**	**$43.7**
54						43.9				$1.01	**$1.01**	**$44.6**
Total / Average		**1,990.1**	**2,903.4**	**4,893.5**	**348.6**	**362.1**	**$0.44**	**$1.24**	**$0.32**	**$1.15**	**$1.94**	**$10,302.3**

Drill and blast costs

Table 21-20 lists the estimated annual drill and blast costs. The required annual bulk explosive quantities were determined from the respective ore, waste and wall control (e.g., pre-split blasting) powder factors as detailed in Item 16. The annual drill and blast costs were estimated from the following unit costs and a Kuz-Ram blast fragmentation model used to calculate explosives consumables:

- ore drill and blast = $1.58/bcm
- waste drill and blast = $0.99/bcm
- wall control drill and blast = $0.82/bcm

The above unit costs are derived from the following consumable items:

- $850/t for bulk explosive
- electric detonators at $25 each
- primers at $6 each
- detonating cord at $0.25/m
- stemming at $30/m^2
- drilling at $7.50/m
- explosive contractor allowance of $0.10/m^2

Life cycle equipment costs

Table 21-21 lists the annual equipment life cycle total dollar cost estimates. This table draws upon the hourly cost information in Table 21-22, along with the following explanations and inputs:

- the listed equipment hourly costs for the primary mining fleet account for component replacements and programmed maintenance parts
- the listed equipment hourly costs exclude labour and capital costs
- the equipment hours for each fleet item are factored for availability and utilisation, respectively, as follows:
  - large and small production drills: 89.03% and 85%
  - pre-split drills: 82.76% and 60%
  - rope shovels: 85% and 90.2%
  - face shovel: 87.2% and 92.5%
  - front end loader: 73.7% and 60%
  - large trucks: 85% and 82.53%
  - small trucks: 83% and 85.3%
- estimated hourly costs for the ancillary equipment, whilst not listed in Table 21-22, have been estimated on the same basis

Diesel fuel costs

Table 21-23 lists the annual equipment hours, fleet diesel fuel consumption and estimated total fuel cost.

The consumption rates for each fleet item are taken to be:

- pre-split drill = 30 litres/hour
- front end loader = 160.1 litres/hour
- large truck = LOM average of 155.6 litres/hour

- small truck = LOM average of 141.4 litres/hour
- combined ancillaries = LOM average of 75.7 litres/hour

Electrical power costs

Table 21-24 lists the annual equipment hours, fleet electric power consumption and estimated total power cost.

The consumption rates for each fleet item are taken to be:

- large drill = 1,237 kW/hour
- small drill = 600 kW/hour
- rope shovel = 2,500 kW/hour
- trolley lines = 3,300 kW/hour

Tyre costs

Table 21-25 lists the relevant annual equipment hours and estimated total tyre consumption cost.

The unit cost rates for each fleet item are taken to be:

- front end loader = $15.40/hour (tyre cost of $154 k and 10,000 hour life)
- large truck = $8.40/hour (tyre cost of $42 k and 5,000 hour life)
- small truck = $6.36/hour (tyre cost of $35 k and 5,500 hour life)
- grader = $0.80/hour (tyre cost of $2.4 k and 3,000 hour life)
- water cart = $5.73/hour (tyre cost of $22.9 k and 4,000 hour life)
- wheel dozer = $5.73/hour (tyre cost of $22.9 k and 4,000 hour life)

Labour costs

Table 21-26 lists the annual manning numbers and estimated total labour cost.

In this table:

- the management contingent includes mining department senior staff, shift supervisors and technical services personnel
- the operators include production drillers, shovel/FEL operators, truck drivers and others
- the maintenance personnel include technicians and supervisors

The labour costs were estimated from the annual salary/wage and on-costs associated with each personnel designation, as per the figures in Table 21-27. These figures include a base annual salary or wage, plus a 30% overhead allowance and travel costs, to and from site.

Table 21-20 Estimated annual drill and blast costs

Year	Stage	Drill and blast volumes			Blasting				Drill and blast costs			Total
		Ore (Mbcm)	Waste (Mbcm)	Wall control (Mbcm)	Ore (t @ PF 1.38kg/bcm)	Waste (t @ PF 0.85kg/bcm)	Wall control (t @ PF 0.60kg/bcm)	Total explosives (t)	Ore @ $1.58/bcm ($M)	Waste @ $0.99/bcm ($M)	Wall control @ $0.82/bcm ($M)	D&B Cost ($M)
1			6.1	1.5		5,177.0	914.7	6,091.7		$6.0	$1.2	**$7.2**
2	Pre-strip		29.8	7.5		25,459.9	4,498.3	29,958.3		$29.4	$6.1	**$35.5**
3		0.2	45.7	11.5	217.7	39,006.4	6,915.5	46,139.7	$0.2	$45.0	$9.4	**$54.6**
4		2.9	49.9	13.2	4,062.7	42,648.2	7,978.3	54,689.2	$4.6	$49.2	$10.8	**$64.7**
5		15.5	43.9	14.8	21,398.8	37,485.6	8,957.0	67,841.4	$24.4	$43.3	$12.1	**$79.8**
6		15.3	44.8	15.0	21,117.7	38,261.6	9,063.4	68,442.7	$24.1	$44.2	$12.3	**$80.6**
7		13.4	47.2	15.2	18,601.9	40,324.5	9,153.5	68,080.0	$21.2	$46.5	$12.4	**$80.2**
8		12.7	48.3	15.2	17,502.9	41,217.6	9,191.4	67,911.9	$20.0	$47.6	$12.5	**$80.0**
9		15.9	45.1	15.3	22,027.5	38,527.9	9,209.7	69,765.0	$25.1	$44.5	$12.5	**$82.1**
10		14.0	47.8	15.5	19,417.2	40,815.6	9,329.2	69,562.0	$22.2	$47.1	$12.7	**$81.9**
11		17.1	44.1	15.3	23,669.0	37,668.4	9,236.8	70,574.2	$27.0	$43.5	$12.5	**$83.0**
12		21.2	39.2	15.1	29,328.3	33,434.6	9,106.0	71,869.0	$33.5	$38.6	$12.3	**$84.4**
13		17.5	42.8	15.1	24,187.0	36,581.5	9,101.3	69,869.8	$27.6	$42.2	$12.3	**$82.2**
14		16.6	43.3	15.0	22,919.7	36,976.8	9,032.9	68,929.4	$26.2	$42.7	$12.2	**$81.1**
15		17.3	43.2	15.1	23,902.2	36,912.3	9,128.7	69,943.2	$27.3	$42.6	$12.4	**$82.3**
16		19.0	36.7	13.9	26,222.0	31,355.0	8,399.8	65,976.8	$29.9	$36.2	$11.4	**$77.5**
17		22.3	25.4	11.9	30,801.1	21,662.0	7,186.7	59,649.8	$35.2	$25.0	$9.7	**$69.9**
18		22.0	17.8	9.9	30,383.5	15,187.0	5,997.1	51,567.6	$34.7	$17.5	$8.1	**$60.3**
19		17.2	18.4	8.9	23,782.1	15,670.7	5,362.6	44,815.4	$27.1	$18.1	$7.3	**$52.5**
20		15.5	20.1	8.9	21,419.3	17,202.0	5,375.4	43,996.7	$24.5	$19.9	$7.3	**$51.6**
21		14.5	21.1	8.9	20,052.0	17,995.6	5,366.5	43,414.1	$22.9	$20.8	$7.3	**$50.9**
22		13.2	22.6	9.0	18,264.6	19,309.3	5,403.7	42,977.6	$20.8	$22.3	$7.3	**$50.5**
23		16.9	13.2	7.5	23,312.6	11,268.4	4,533.5	39,114.5	$26.6	$13.0	$6.1	**$45.8**
24	Production	17.3	12.8	7.5	23,992.7	10,959.2	4,553.1	39,504.9	$27.4	$12.6	$6.2	**$46.2**
25		18.7	11.4	7.5	25,840.8	9,716.9	4,535.2	40,092.9	$29.5	$11.2	$6.2	**$46.9**
26		18.4	11.5	7.5	25,397.6	9,811.6	4,503.5	39,712.7	$29.0	$11.3	$6.1	**$46.4**
27		18.9	10.8	7.4	26,131.5	9,261.3	4,486.4	39,879.1	$29.8	$10.7	$6.1	**$46.6**
28		17.8	11.8	7.4	24,613.1	10,033.4	4,457.2	39,103.6	$28.1	$11.6	$6.0	**$45.7**
29		20.5	8.9	7.3	28,308.9	7,623.1	4,434.4	40,366.4	$32.3	$8.8	$6.0	**$47.1**
30		20.0	9.3	7.3	27,598.4	7,906.7	4,407.0	39,912.1	$31.5	$9.1	$6.0	**$46.6**
31		16.8	5.8	5.7	23,295.6	4,947.1	3,414.8	31,657.5	$26.6	$5.7	$4.6	**$36.9**
32		15.0	3.2	4.6	20,774.2	2,750.6	2,751.7	26,276.6	$23.7	$3.2	$3.7	**$30.6**
33		14.3	3.8	4.5	19,751.0	3,257.6	2,729.7	25,738.3	$22.5	$3.8	$3.7	**$30.0**
34		13.8	4.3	4.5	19,158.5	3,682.6	2,740.2	25,581.3	$21.9	$4.3	$3.7	**$29.8**
35		14.2	4.0	4.6	19,688.0	3,412.7	2,750.2	25,850.9	$22.5	$3.9	$3.7	**$30.1**
36		15.5	2.8	4.6	21,409.9	2,367.5	2,753.4	26,530.8	$24.4	$2.7	$3.7	**$30.9**
37		16.1	2.1	4.6	22,279.0	1,829.3	2,753.1	26,861.4	$25.4	$2.1	$3.7	**$31.3**
38		16.0	1.7	4.4	22,099.3	1,436.5	2,664.1	26,199.8	$25.2	$1.7	$3.6	**$30.5**
39		8.5	0.8	2.3	11,697.0	717.1	1,402.4	13,816.5	$13.4	$0.8	$1.9	**$16.1**
40		8.4	0.8	2.3	11,583.3	642.1	1,376.8	13,602.2	$13.2	$0.7	$1.9	**$15.8**
41		9.4	0.4	2.4	12,940.4	378.3	1,478.2	14,796.9	$14.8	$0.4	$2.0	**$17.2**
42												
43												
44												
Total		**599.4**	**902.8**	**375.5**	**829,149.2**	**770,881.3**	**226,633.4**	**1,826,663.8**	**$946.5**	**$889.8**	**$307.3**	**$2,143.6**

Table 21-21 Estimated annual equipment life cycle costs

Year	Stage	Drills LCC Large ($M)	Drills LCC Small ($M)	Drills LCC Pre-split ($M)	Shovel LCC Rope ($M)	Shovel LCC Face ($M)	Loader LCC FEL ($M)	Trucks LCC Large ($M)	Trucks LCC Small ($M)	Ancillaries LCC Dozer, grader, w/cart ($M)	Ancillaries LCC Support ($M)	Miscellaneous Systems ($M)	Miscellaneous Others ($M)	Total LCC ($M)
1	Pre-strip	$0.1	$0.1	$0.2	$0.3	$0.0	$0.5	$0.9	$0.7	$0.7	$1.4	$2.3	$4.3	**$11.6**
2		$1.2	$0.6	$0.8	$3.2	$1.2	$1.8	$9.7	$4.5	$9.8	$6.7	$2.3	$6.3	**$47.9**
3		$3.4	$1.6	$1.7	$4.4	$1.7	$1.2	$16.1	$7.0	$19.3	$10.4	$2.1	$7.7	**$76.4**
4		$3.7	$2.3	$1.7	$6.8	$1.3	$2.3	$30.1	$4.6	$21.3	$12.1	$2.1	$8.3	**$96.6**
5	Production	$4.3	$4.1	$1.7	$7.5	$3.7	$2.0	$23.2	$7.6	$24.0	$14.3	$2.1	$9.0	**$103.5**
6		$4.0	$4.4	$1.7	$6.0	$2.5	$3.0	$34.4	$6.9	$26.4	$14.0	$2.1	$9.0	**$114.5**
7		$2.7	$3.8	$1.7	$8.7	$3.0	$2.3	$31.5	$6.8	$25.5	$14.4	$2.1	$9.0	**$111.5**
8		$3.4	$5.4	$1.7	$5.4	$4.1	$3.1	$28.0	$4.5	$24.4	$14.3	$2.1	$9.0	**$105.1**
9		$4.4	$4.1	$1.7	$8.5	$3.1	$2.7	$31.1	$3.8	$19.6	$13.9	$2.1	$9.0	**$104.0**
10		$3.2	$5.2	$1.7	$7.8	$2.4	$3.9	$38.1	$2.1	$21.2	$13.9	$2.1	$9.0	**$110.5**
11		$4.1	$5.8	$1.7	$5.6	$3.0	$2.1	$35.0	$2.7	$20.2	$13.8	$2.1	$9.0	**$105.0**
12		$1.5	$3.9	$1.7	$6.8	$4.2	$1.4	$39.2	$3.2	$23.8	$13.8	$2.1	$9.0	**$110.5**
13		$2.4	$4.6	$1.7	$7.0	$0.9	$2.2	$28.9	$3.4	$24.7	$14.1	$2.1	$9.0	**$100.8**
14		$4.5	$3.0	$1.7	$8.5	$1.7	$2.5	$31.0	$2.0	$23.3	$14.0	$2.1	$9.0	**$103.2**
15		$2.9	$3.5	$1.7	$7.6	$3.1	$2.7	$27.9	$1.9	$23.2	$14.0	$2.1	$9.0	**$99.6**
16		$5.7	$3.8	$1.7	$6.8	$3.1	$3.1	$24.1	$6.0	$20.0	$13.2	$2.1	$8.6	**$98.0**
17		$1.2	$4.5	$1.7	$5.0	$1.5	$2.1	$20.3	$7.2	$16.5	$11.0	$2.1	$7.9	**$80.9**
18		$1.6	$5.1	$1.3	$4.2	$2.5	$3.4	$17.7	$7.3	$13.3	$9.7	$2.1	$7.3	**$75.3**
19		$2.3	$3.8	$1.3	$1.4	$0.9	$2.8	$15.5	$4.2	$10.9	$8.7	$2.1	$6.9	**$60.7**
20		$1.3	$4.8	$1.3	$1.4	$1.7	$3.1	$15.8	$2.8	$15.7	$8.8	$2.1	$6.9	**$65.6**
21		$3.0	$3.1	$1.3	$1.4	$1.5	$1.5	$12.9	$7.6	$13.1	$8.8	$2.1	$6.9	**$63.3**
22		$0.4	$4.5	$1.3	$1.4	$2.3	$1.7	$15.8	$5.9	$12.4	$8.8	$2.1	$6.9	**$63.5**
23		$0.7	$4.5	$1.3	$1.4		$1.4	$14.4	$3.8	$13.4	$7.6	$2.1	$6.4	**$56.8**
24		$0.0	$3.6	$1.3	$2.5		$1.3	$18.8	$3.7	$13.9	$7.5	$2.1	$6.4	**$61.0**
25		$0.0	$3.8	$1.3	$3.2		$1.3	$15.5	$2.1	$13.8	$7.3	$2.1	$6.4	**$56.8**
26		$0.8	$1.9	$1.3	$3.0		$0.9	$21.1	$2.3	$11.7	$7.5	$2.1	$6.4	**$59.0**
27		$1.1	$2.4	$1.3	$3.4		$1.6	$21.0	$2.8	$11.2	$7.6	$2.1	$6.4	**$60.9**
28			$1.4	$1.3	$1.9		$1.6	$20.9	$2.6	$15.0	$7.5	$2.1	$6.4	**$60.6**
29			$3.0	$1.3	$4.1		$0.9	$23.4	$2.6	$10.6	$7.1	$2.1	$6.4	**$61.4**
30			$2.1	$1.3	$2.6		$1.2	$23.3	$2.9	$14.5	$6.9	$2.1	$6.4	**$63.2**
31			$2.7	$0.8	$3.3		$1.2	$20.4	$2.3	$9.6	$2.6	$2.1	$5.8	**$50.8**
32			$3.3	$0.8	$2.7		$0.8	$19.7	$3.1	$8.9	$2.1	$2.1	$5.4	**$48.8**
33			$2.7	$0.8	$2.2		$1.5	$16.2	$3.1	$7.5	$2.2	$2.1	$5.4	**$43.6**
34			$3.9	$0.8	$3.6		$0.8	$16.9	$1.6	$7.7	$2.1	$2.1	$5.4	**$44.9**
35			$3.3	$0.8	$3.5		$0.8	$15.7	$1.4	$8.3	$2.1	$2.1	$5.4	**$43.4**
36			$3.0	$0.8	$3.0			$16.6	$2.5	$9.8	$2.0	$2.1	$5.4	**$45.1**
37			$2.5	$0.8	$2.9			$17.9	$2.0	$8.8	$2.0	$2.1	$5.4	**$44.4**
38			$2.1	$0.8	$2.6			$15.9	$1.4	$8.4	$1.9	$2.1	$5.3	**$40.5**
39			$1.3	$0.4	$2.3			$9.8		$4.8	$1.5	$2.1	$4.6	**$26.7**
40			$1.5	$0.4	$2.4			$10.1		$7.1	$1.5	$2.1	$4.6	**$29.6**
41			$0.4		$3.0			$10.5		$5.0	$1.4	$2.1	$4.6	**$26.9**
42					$2.1			$3.8		$5.2	$1.3	$2.1	$3.8	**$18.3**
43					$1.4			$3.8		$3.9	$1.3	$1.0	$1.9	**$13.3**
44					$2.1			$3.8		$5.4	$1.3	$1.0	$1.9	**$15.4**
45					$1.5			$3.6		$5.4	$1.3	$1.0	$1.9	**$14.7**
46					$2.4			$3.7		$5.7	$1.3	$1.0	$1.9	**$15.9**
47					$1.6			$4.0		$6.4	$1.3	$1.0	$1.9	**$16.1**
48					$2.0			$6.4		$6.8	$1.3	$1.0	$1.9	**$19.3**
49					$2.1			$5.1		$5.6	$1.3	$1.0	$1.9	**$17.0**
50					$1.4			$5.9		$6.8	$1.3	$1.0	$1.9	**$18.2**
51					$3.0			$5.4		$3.6	$1.3	$1.0	$1.9	**$16.3**
52					$1.5			$4.5		$5.9	$1.3	$1.0	$1.9	**$16.0**
53					$1.8			$4.4		$3.0	$1.3	$1.0	$1.9	**$13.4**
54					$2.4			$6.4		$2.0	$1.4	$1.0	$1.9	**$15.1**
Total		**$63.9**	**$131.2**	**$50.1**	**$194.9**	**$49.3**	**$66.8**	**$916.0**	**$143.0**	**$664.9**	**$351.6**	**$98.7**	**$310.8**	**$3,041.1**

Table 21-22 Hourly life cycle cost estimates

SMU per interval	Drills LCC Large ($/h)	Drills LCC Small ($/h)	Drills LCC Pre-split ($/h)	Shovel LCC Rope ($/h)	Shovel LCC Face ($/h)	Loader LCC FEL ($/h)	Trucks LCC Large ($/h)	Trucks LCC Small ($/h)
6,000	$31.23	$34.62	$96.33	$100.61	$52.04	$171.28	$34.10	$39.56
12,000	$109.64	$89.72	$96.33	$159.71	$165.41	$296.89	$60.79	$90.50
18,000	$246.83	$168.28	$96.33	$162.96	$237.79	$198.52	$93.32	$128.59
24,000	$124.10	$111.23	$96.33	$227.01	$183.86	$382.59	$164.64	$146.11
30,000	$288.94	$214.15	$96.33	$244.58	$357.45	$224.29	$104.72	$75.41
36,000	$83.88	$58.96	$96.33	$140.93	$120.90	$310.42	$151.24	$179.23
42,000	$95.88	$119.77	$96.33	$379.47	$190.88	$200.83	$135.88	$55.25
48,000	$275.05	$170.31	$96.33	$153.28	$236.59	$389.33	$94.60	$166.88
54,000	$87.81	$96.21	$96.33	$197.73	$216.82	$200.83	$104.65	$127.90
60,000	$325.22	$229.17	$96.33	$286.16	$323.18	$410.55	$162.21	$79.23
66,000	$55.66	$56.94	$96.33	$150.61	$97.03	$198.52	$124.98	$76.10
72,000	$120.47	$100.55	$96.33	$188.04	$210.24	$33.92	$127.73	$31.25
78,000	$6.86	$6.26	$96.33	$162.96	$273.60	$33.92	$68.77	$31.25
84,000				$369.79	$193.89	$33.92	$79.42	$31.25
90,000				$244.58	$28.69	$33.92		$31.25
96,000				$140.93	$28.69	$33.92		$31.25
102,000				$236.70	$28.69			$31.25
108,000				$153.28				$31.25
114,000				$197.73				$31.25
120,000				$70.84				

.

Table 21-23 Estimated annual diesel fuel consumption and costs

Year	Stage	Primary Equipment P-S drill (hours)	Loader (hours)	Large truck (hours)	Small truck (hours)	Total Ancillaries (hours)	P-S drill 30 L/h (kL)	Loader 160.1 L/h (kL)	Large truck Fuel burn (kL)	Small truck Fuel burn (kL)	Total Ancillaries (kL)	Total Fuel Consumption (kL)	Total Fuel Cost ($M)
1	Pre-strip	2,175	2,983	27,657	18,540	16,849	65.3	477.6	4,179.3	2,068.5	1,313.0	**8,103.7**	**$8.3**
2		8,700	5,965	159,025	49,408	87,186	261.0	955.0	24,866.6	5,767.4	6,491.2	**38,341.2**	**$39.1**
3		17,400	5,966	199,904	49,392	141,537	522.0	955.2	32,784.9	5,583.4	10,862.5	**50,708.0**	**$51.6**
4		17,447	5,981	231,804	49,752	161,766	523.4	957.6	38,207.7	5,757.3	12,276.0	**57,721.9**	**$58.8**
5	Production	17,399	9,840	249,779	49,616	195,021	522.0	1,575.4	47,259.2	7,135.1	14,797.9	**71,289.6**	**$72.6**
6		17,398	9,838	244,741	49,616	195,027	521.9	1,575.1	47,843.5	5,789.4	14,798.3	**70,528.2**	**$71.8**
7		17,400	9,840	242,596	49,616	195,024	522.0	1,575.4	49,824.9	7,605.3	14,798.1	**74,325.6**	**$75.7**
8		17,448	9,866	255,863	49,752	195,556	523.4	1,579.5	48,539.7	6,529.0	14,838.5	**72,010.2**	**$73.3**
9		17,398	9,839	309,584	49,616	195,026	521.9	1,575.2	53,116.6	7,558.9	14,798.2	**77,570.9**	**$79.0**
10		17,398	9,840	290,825	49,616	195,016	521.9	1,575.4	50,407.4	6,131.1	14,797.4	**73,433.3**	**$74.8**
11		17,400	9,838	302,363	62,020	195,022	522.0	1,575.1	53,211.3	7,714.7	14,798.0	**77,821.0**	**$79.3**
12		17,448	9,867	291,548	62,190	195,560	523.4	1,579.7	53,398.4	7,944.6	14,838.7	**78,284.9**	**$79.7**
13		17,398	9,838	304,800	62,020	195,019	521.9	1,575.1	51,192.5	7,484.3	14,797.7	**75,571.6**	**$77.0**
14		17,398	9,840	287,657	55,818	195,026	521.9	1,575.4	43,520.4	6,938.2	14,798.3	**67,354.2**	**$68.6**
15		17,400	9,839	298,637	55,818	195,025	522.0	1,575.2	47,325.6	6,421.0	14,798.1	**70,642.0**	**$72.0**
16		17,448	9,866	286,897	55,971	168,513	523.4	1,579.5	53,447.3	8,507.2	12,846.3	**76,903.8**	**$78.3**
17		17,398	9,840	251,623	55,818	154,594	521.9	1,575.4	39,343.5	7,652.0	11,603.0	**60,695.8**	**$61.8**
18		13,048	9,838	222,266	55,818	127,619	391.4	1,575.1	31,574.5	7,675.3	9,615.7	**50,832.0**	**$51.8**
19		13,050	9,840	194,882	55,818	120,887	391.5	1,575.4	30,574.7	8,304.5	9,046.9	**49,892.9**	**$50.8**
20		13,086	9,865	192,508	55,971	121,221	392.6	1,579.4	31,155.2	8,845.4	9,071.9	**51,044.5**	**$52.0**
21		13,049	7,748	204,872	55,818	120,887	391.5	1,240.5	31,703.8	8,812.6	9,046.9	**51,195.3**	**$52.1**
22		13,048	7,748	214,545	55,818	120,887	391.4	1,240.5	32,109.7	8,799.1	9,046.9	**51,587.7**	**$52.5**
23		13,050	7,747	184,150	55,818	107,737	391.5	1,240.3	27,229.1	8,759.8	8,298.4	**45,919.1**	**$46.8**
24		13,086	7,769	181,901	55,971	108,031	392.6	1,243.8	29,062.3	8,649.3	8,321.1	**47,669.0**	**$48.6**
25		13,049	7,747	178,267	55,818	107,737	391.5	1,240.3	30,119.2	7,913.4	8,298.4	**47,962.8**	**$48.9**
26		13,048	7,747	193,770	55,818	107,735	391.4	1,240.3	31,126.0	10,903.0	8,298.1	**51,958.9**	**$52.9**
27		13,050	7,748	212,936	55,818	107,736	391.5	1,240.5	35,369.1	11,714.0	8,298.4	**57,013.5**	**$58.1**
28		13,086	7,768	219,564	55,971	108,032	392.6	1,243.7	28,218.2	9,217.4	8,321.1	**47,392.8**	**$48.3**
29		13,049	7,748	227,526	55,818	107,736	391.5	1,240.5	30,738.8	7,845.1	8,298.4	**48,514.1**	**$49.4**
30		13,048	7,747	236,335	55,818	107,733	391.4	1,240.3	27,164.5	8,246.2	8,298.1	**45,340.5**	**$46.2**
31		8,700	3,873	187,607	24,808	87,188	261.0	620.1	19,405.6	2,571.0	6,491.3	**29,348.9**	**$29.9**
32		8,724	3,885	147,664	24,876	74,236	261.7	622.0	13,762.9	2,596.0	5,758.3	**23,000.9**	**$23.4**
33		8,699	3,873	158,186	24,808	74,035	261.0	620.1	17,001.0	5,321.9	5,742.5	**28,946.5**	**$29.5**
34		8,699	3,874	165,274	24,808	74,032	261.0	620.2	17,658.3	3,577.6	5,742.4	**27,859.4**	**$28.4**
35		8,700	3,874	156,215	24,808	74,034	261.0	620.2	18,707.5	3,019.8	5,742.5	**28,351.1**	**$28.9**
36		8,724		167,682	24,876	74,237	261.7		20,351.1	3,011.6	5,758.2	**29,382.6**	**$29.9**
37		8,699		184,480	24,808	74,035	261.0		22,800.1	3,011.6	5,742.6	**31,815.3**	**$32.4**
38		8,699		180,554	24,808	74,033	261.0		20,884.9	3,011.6	5,742.4	**29,899.9**	**$30.5**
39		4,350		118,240		53,485	130.5		19,932.2		3,935.5	**23,998.2**	**$24.4**
40		4,362		117,217		53,632	130.9		20,045.8		3,946.3	**24,122.9**	**$24.6**
41				117,158		53,481			15,382.8		3,935.1	**19,318.0**	**$19.7**
42				47,992		53,486			7,253.1		3,935.4	**11,188.5**	**$11.4**
43				47,993		53,482			7,272.9		3,935.2	**11,208.1**	**$11.4**
44				48,127		53,633			7,253.1		3,946.3	**11,199.4**	**$11.4**
45				47,993		53,484			7,253.1		3,935.4	**11,188.5**	**$11.4**
46				47,288		53,484			7,146.5		3,935.4	**11,081.8**	**$11.3**
47				47,238		53,486			7,158.1		3,935.5	**11,093.6**	**$11.3**
48				47,364		53,631			7,138.6		3,946.2	**11,084.8**	**$11.3**
49				47,238		53,484			7,138.6		3,935.3	**11,073.9**	**$11.3**
50				47,239		53,483			7,138.6		3,935.3	**11,073.9**	**$11.3**
51				47,235		53,484			7,158.1		3,935.3	**11,093.4**	**$11.3**
52				47,366		53,631			7,138.6		3,946.2	**11,084.8**	**$11.3**
53				47,240		53,487			7,138.6		3,935.6	**11,074.2**	**$11.3**
54				51,864		40,024			7,838.0		2,656.7	**10,494.6**	**$10.7**
Total		**520,156**	**275,325**	**9,221,279**	**1,799,272**	**5,799,452**	**15,605**	**44,080**	**1,434,572**	**254,393**	**438,963**	**2,187,612**	**$2,228.2**

Table 21-24 Estimated annual electric power consumption and costs

Year	Stage	Equipment hours: Large drill (hours)	Equipment hours: Small drill (hours)	Equipment hours: Rope shovel (hours)	Equipment hours: Trolley lines (hours)	Large drill 1,237 kW/h (MW)	Small drill 600 kW/h (MW)	Rope shovel 2,500 kW/h (MW)	Trolley lines 3,300 kW/h (MW)	Total Power Consumption (MW)	Total Power Cost ($M)
1	Pre-strip	3,315	3,081	3,358	0	4,100.7	1,848.6	8,393.4		**14,342.7**	$0.9
2		13,258	6,162	20,148	1,611	16,400.1	3,697.2	50,360.4	5,316.3	**75,774.1**	$4.9
3		26,516	12,325	26,867	1,976	32,800.3	7,395.0	67,154.7	6,520.8	**113,870.8**	$7.4
4		26,591	18,538	33,672	3,483	32,893.1	11,122.8	84,164.0	11,493.9	**139,673.8**	$9.1
5	Production	26,516	30,811	33,583	11,955	32,800.3	18,486.6	83,941.5	39,451.5	**174,679.9**	$11.4
6		26,517	30,813	33,581	10,116	32,801.5	18,487.8	83,936.5	33,382.8	**168,608.7**	$11.0
7		26,516	30,810	33,581	5,075	32,800.3	18,486.0	83,936.5	16,747.5	**151,970.3**	$9.9
8		26,589	37,075	33,674	23,090	32,890.6	22,245.0	84,169.0	76,197.0	**215,501.6**	$14.0
9		26,518	36,974	33,581	43,334	32,802.8	22,184.4	83,936.5	143,002.2	**281,925.9**	$18.3
10		26,516	36,975	33,581	29,215	32,800.3	22,185.0	83,936.5	96,409.5	**235,331.3**	$15.3
11		26,517	36,973	33,582	36,777	32,801.5	22,183.8	83,939.0	121,364.1	**260,288.5**	$16.9
12		26,588	37,074	33,673	35,126	32,889.4	22,244.4	84,166.5	115,915.8	**255,216.1**	$16.6
13		26,516	36,977	33,582	40,898	32,800.3	22,186.2	83,939.0	134,963.4	**273,888.9**	$17.8
14		26,516	36,972	33,581	51,812	32,800.3	22,183.2	83,936.5	170,979.6	**309,899.6**	$20.1
15		26,520	36,973	33,581	43,340	32,805.2	22,183.8	83,936.5	143,022.0	**281,947.6**	$18.3
16		26,588	37,075	33,674	26,591	32,889.4	22,245.0	84,169.0	87,750.3	**227,053.7**	$14.8
17		19,887	36,974	26,864	43,762	24,600.2	22,184.4	67,147.2	144,414.6	**258,346.5**	$16.8
18		13,258	36,974	26,867	45,526	16,400.1	22,184.4	67,154.7	150,235.8	**255,975.1**	$16.6
19		13,258	30,811	20,148	35,623	16,400.1	18,486.6	50,360.4	117,555.9	**202,803.1**	$13.2
20		13,296	30,896	20,205	38,990	16,447.2	18,537.6	50,502.9	128,667.0	**214,154.7**	$13.9
21		13,258	30,811	20,148	46,678	16,400.1	18,486.6	50,360.4	154,037.4	**239,284.6**	$15.6
22		6,629	30,813	20,148	47,240	8,200.1	18,487.8	50,360.4	155,892.0	**232,940.3**	$15.1
23		6,629	30,811	20,149	43,718	8,200.1	18,486.6	50,362.9	144,269.4	**221,319.0**	$14.4
24		6,647	30,895	20,204	34,185	8,222.3	18,537.0	50,500.4	112,810.5	**190,070.2**	$12.4
25		6,629	30,813	20,149	27,731	8,200.1	18,487.8	50,362.9	91,512.3	**168,563.1**	$11.0
26		6,629	30,810	20,149	33,184	8,200.1	18,486.0	50,362.9	109,507.2	**186,556.2**	$12.1
27		6,629	30,812	20,149	35,276	8,200.1	18,487.2	50,362.9	116,410.8	**193,461.0**	$12.6
28			30,896	20,204	55,353		18,537.6	50,500.4	182,664.9	**251,702.9**	$16.4
29			30,813	20,149	56,421		18,487.8	50,362.9	186,189.3	**255,040.0**	$16.6
30			30,811	20,149	83,858		18,486.6	50,362.9	276,731.4	**345,580.9**	$22.5
31			30,811	20,149	73,862		18,486.6	50,362.9	243,744.6	**312,594.1**	$20.3
32			24,716	13,470	60,374		14,829.6	33,668.6	199,234.2	**247,732.4**	$16.1
33			24,650	13,432	60,807		14,790.0	33,573.6	200,663.1	**249,026.7**	$16.2
34			24,650	13,432	66,007		14,790.0	33,573.6	217,823.1	**266,186.7**	$17.3
35			24,648	13,433	59,321		14,788.8	33,576.1	195,759.3	**244,124.2**	$15.9
36			24,716	13,469	7,254		14,829.6	33,666.1	23,938.2	**72,433.9**	$4.7
37			24,651	13,433	21,058		14,790.6	33,576.1	69,491.4	**117,858.1**	$7.7
38			24,649	13,433	47,904		14,789.4	33,576.1	158,083.2	**206,448.7**	$13.4
39			18,486	13,432	23,475		11,091.6	33,573.6	77,467.5	**122,132.7**	$7.9
40			18,538	13,470	22,971		11,122.8	33,668.6	75,804.3	**120,595.7**	$7.8
41			18,488	13,432	26,388		11,092.8	33,573.6	87,080.4	**131,746.8**	$8.6
42				13,433				33,576.1		**33,576.1**	$2.2
43				13,433				33,576.1		**33,576.1**	$2.2
44				13,469				33,666.1		**33,666.1**	$2.2
45				13,433				33,576.1		**33,576.1**	$2.2
46				13,432				33,573.6		**33,573.6**	$2.2
47				13,433				33,576.1		**33,576.1**	$2.2
48				13,469				33,666.1		**33,666.1**	$2.2
49				13,433				33,576.1		**33,576.1**	$2.2
50				13,432				33,573.6		**33,573.6**	$2.2
51				13,433				33,576.1		**33,576.1**	$2.2
52				13,469				33,666.1		**33,666.1**	$2.2
53				13,433				33,576.1		**33,576.1**	$2.2
54				13,432				33,573.6		**33,573.6**	$2.2
Total		**500,846.0**	**1,168,551.0**	**1,112,300.0**	**1,461,365.0**	619,546.5	**701,130.6**	**2,780,221.7**	**4,822,504.5**	**8,923,403.3**	**$580.0**

Table 21-25 Estimated annual tyre consumption costs

Year	Stage	Equipment hours Loader (hours)	Large truck (hours)	Small truck (hours)	Grader (hours)	Water Cart (hours)	Wheel dozer (hours)	Loader ($M) 4 tyres $15.4/h	Large truck ($M) 6 tyres $8.4/h	Small truck ($M) 6 tyres $6.4/h	Grader ($M) 6 tyres $0.8/h	Water Cart ($M) 6 tyres $5.7/h	Wheel dozer ($M) 4 tyres $5.728/h	Total Tyre Cost ($M)
1		2,983	27,657	18,540	3,210	6,730	3,593	$0.2	$1.4	$0.7	$0.0	$0.2	$0.1	**$2.6**
2	Pre-strip	5,965	159,025	49,408	19,257	20,192	14,372	$0.4	$8.0	$1.9	$0.1	$0.7	$0.3	**$11.4**
3		5,966	199,904	49,392	25,676	33,651	28,744	$0.4	$10.1	$1.9	$0.1	$1.2	$0.7	**$14.3**
4		5,981	231,804	49,752	32,185	40,494	28,824	$0.4	$11.7	$1.9	$0.2	$1.4	$0.7	**$16.2**
5		9,840	249,779	49,616	38,514	47,112	35,930	$0.6	$12.6	$1.9	$0.2	$1.6	$0.8	**$17.7**
6		9,838	244,741	49,616	38,516	47,115	35,930	$0.6	$12.3	$1.9	$0.2	$1.6	$0.8	**$17.5**
7		9,840	242,596	49,616	38,516	47,112	35,931	$0.6	$12.2	$1.9	$0.2	$1.6	$0.8	**$17.4**
8		9,866	255,863	49,752	38,620	47,243	36,029	$0.6	$12.9	$1.9	$0.2	$1.6	$0.8	**$18.0**
9		9,839	309,584	49,616	38,516	47,114	35,930	$0.6	$15.6	$1.9	$0.2	$1.6	$0.8	**$20.7**
10		9,840	290,825	49,616	38,515	47,110	35,930	$0.6	$14.7	$1.9	$0.2	$1.6	$0.8	**$19.8**
11		9,838	302,363	62,020	38,514	47,114	35,931	$0.6	$15.2	$2.4	$0.2	$1.6	$0.8	**$20.8**
12		9,867	291,548	62,190	38,622	47,242	36,029	$0.6	$14.7	$2.4	$0.2	$1.6	$0.8	**$20.3**
13		9,838	304,800	62,020	38,514	47,112	35,930	$0.6	$15.4	$2.4	$0.2	$1.6	$0.8	**$21.0**
14		9,840	287,657	55,818	38,515	47,115	35,930	$0.6	$14.5	$2.1	$0.2	$1.6	$0.8	**$19.9**
15		9,839	298,637	55,818	38,517	47,112	35,931	$0.6	$15.1	$2.1	$0.2	$1.6	$0.8	**$20.4**
16		9,866	286,897	55,971	32,182	40,494	28,824	$0.6	$14.5	$2.1	$0.2	$1.4	$0.7	**$19.4**
17		9,840	251,623	55,818	32,098	33,652	28,744	$0.6	$12.7	$2.1	$0.2	$1.2	$0.7	**$17.4**
18		9,838	222,266	55,818	25,677	26,920	21,558	$0.6	$11.2	$2.1	$0.1	$0.9	$0.5	**$15.5**
19		9,840	194,882	55,818	25,676	26,921	21,559	$0.6	$9.8	$2.1	$0.1	$0.9	$0.5	**$14.1**
20		9,865	192,508	55,971	25,748	26,995	21,617	$0.6	$9.7	$2.1	$0.1	$0.9	$0.5	**$14.0**
21		7,748	204,872	55,818	25,676	26,923	21,559	$0.5	$10.3	$2.1	$0.1	$0.9	$0.5	**$14.5**
22		7,748	214,545	55,818	25,676	26,921	21,558	$0.5	$10.8	$2.1	$0.1	$0.9	$0.5	**$15.0**
23		7,747	184,150	55,818	19,258	26,922	21,558	$0.5	$9.3	$2.1	$0.1	$0.9	$0.5	**$13.4**
24		7,769	181,901	55,971	19,310	26,996	21,618	$0.5	$9.2	$2.1	$0.1	$0.9	$0.5	**$13.3**
25		7,747	178,267	55,818	19,258	26,922	21,558	$0.5	$9.0	$2.1	$0.1	$0.9	$0.5	**$13.1**
26		7,747	193,770	55,818	19,259	26,920	21,558	$0.5	$9.8	$2.1	$0.1	$0.9	$0.5	**$13.9**
27		7,748	212,936	55,818	19,257	26,922	21,558	$0.5	$10.7	$2.1	$0.1	$0.9	$0.5	**$14.9**
28		7,768	219,564	55,971	19,311	26,995	21,618	$0.5	$11.1	$2.1	$0.1	$0.9	$0.5	**$15.2**
29	Production	7,748	227,526	55,818	19,257	26,922	21,558	$0.5	$11.5	$2.1	$0.1	$0.9	$0.5	**$15.6**
30		7,747	236,335	55,818	19,257	26,921	21,559	$0.5	$11.9	$2.1	$0.1	$0.9	$0.5	**$16.0**
31		3,873	187,607	24,808	19,259	20,192	14,372	$0.2	$9.5	$0.9	$0.1	$0.7	$0.3	**$11.8**
32		3,885	147,664	24,876	12,872	20,247	14,412	$0.2	$7.4	$0.9	$0.1	$0.7	$0.3	**$9.7**
33		3,873	158,186	24,808	12,840	20,191	14,372	$0.2	$8.0	$0.9	$0.1	$0.7	$0.3	**$10.2**
34		3,874	165,274	24,808	12,838	20,190	14,372	$0.2	$8.3	$0.9	$0.1	$0.7	$0.3	**$10.6**
35		3,874	156,215	24,808	12,838	20,191	14,372	$0.2	$7.9	$0.9	$0.1	$0.7	$0.3	**$10.1**
36			167,682	24,876	12,874	20,246	14,412		$8.5	$0.9	$0.1	$0.7	$0.3	**$10.5**
37			184,480	24,808	12,838	20,193	14,372		$9.3	$0.9	$0.1	$0.7	$0.3	**$11.3**
38			180,554	24,808	12,838	20,190	14,372		$9.1	$0.9	$0.1	$0.7	$0.3	**$11.1**
39			118,240		12,838	13,462	7,186		$6.0	$0.0	$0.1	$0.5	$0.2	**$6.6**
40			117,217		12,874	13,498	7,206		$5.9	$0.0	$0.1	$0.5	$0.2	**$6.6**
41			117,158		12,838	13,460	7,186		$5.9	$0.0	$0.1	$0.5	$0.2	**$6.6**
42			47,992		12,840	13,460	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
43			47,993		12,838	13,460	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
44			48,127		12,874	13,498	7,206		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
45			47,993		12,838	13,462	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
46			47,288		12,838	13,460	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
47			47,238		12,839	13,462	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
48			47,364		12,873	13,498	7,206		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
49			47,238		12,839	13,460	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
50			47,239		12,838	13,460	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
51			47,235		12,839	13,460	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
52			47,366		12,873	13,498	7,206		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
53			47,240		12,839	13,462	7,186		$2.4	$0.0	$0.1	$0.5	$0.2	**$3.1**
54			51,864		12,838		7,187		$2.6	$0.0	$0.1	$0.0	$0.2	**$2.8**
Total		**275,325**	**9,221,279**	**1,799,272**	**1,165,860**	**1,407,614**	**1,039,081**	**$17.0**	**$464.8**	**$68.7**	**$5.6**	**$48.4**	**$23.8**	**$628.2**

Table 21-26 Estimated annual manning numbers and labour costs

Year	Stage	Head count				Manning costs			
		Mg't #	Operator #	Maintenance #	Total #	Mg't ($M)	Operator ($M)	Maintenance ($M)	Total ($M)
1	Pre-strip	18	169	166	**353**	$6.8	$5.9	$4.8	**$17.5**
2		33	302	214	**550**	$12.3	$11.0	$6.1	**$29.4**
3		53	403	260	**716**	$19.9	$14.6	$7.4	**$41.9**
4		53	445	267	**765**	$19.9	$16.1	$7.6	**$43.6**
5	Production	53	495	301	**849**	$19.9	$18.0	$8.5	**$46.4**
6		53	495	301	**849**	$19.9	$18.0	$8.5	**$46.4**
7		53	495	301	**849**	$19.9	$18.0	$8.5	**$46.4**
8		53	504	305	**862**	$19.9	$18.3	$8.6	**$46.9**
9		53	536	321	**910**	$19.9	$19.7	$9.0	**$48.7**
10		53	536	321	**910**	$19.9	$19.7	$9.0	**$48.7**
11		53	515	325	**893**	$19.9	$19.3	$9.1	**$48.4**
12		53	515	325	**893**	$19.9	$19.3	$9.1	**$48.4**
13		53	515	325	**893**	$19.9	$19.3	$9.1	**$48.4**
14		53	507	321	**881**	$19.9	$19.0	$9.0	**$47.9**
15		53	507	321	**881**	$19.9	$19.0	$9.0	**$47.9**
16		53	479	309	**840**	$19.9	$18.0	$8.7	**$46.7**
17		53	440	292	**785**	$19.9	$16.6	$8.3	**$44.7**
18		48	391	271	**710**	$17.9	$14.7	$7.7	**$40.3**
19		48	365	261	**674**	$17.9	$13.7	$7.5	**$39.1**
20		48	365	261	**674**	$17.9	$13.7	$7.5	**$39.1**
21		48	363	260	**670**	$17.9	$13.6	$7.4	**$39.0**
22		48	360	259	**667**	$17.9	$13.5	$7.4	**$38.9**
23		48	332	246	**626**	$17.9	$12.4	$7.1	**$37.4**
24		48	332	246	**626**	$17.9	$12.4	$7.1	**$37.4**
25		48	332	246	**626**	$17.9	$12.4	$7.1	**$37.4**
26		48	336	248	**632**	$17.9	$12.6	$7.1	**$37.7**
27		48	346	253	**646**	$17.9	$13.1	$7.2	**$38.2**
28		48	347	243	**638**	$17.9	$13.1	$7.0	**$38.1**
29		48	355	247	**650**	$17.9	$13.5	$7.1	**$38.5**
30		48	350	247	**645**	$17.9	$13.3	$7.1	**$38.4**
31		25	266	216	**506**	$9.4	$10.2	$6.3	**$25.8**
32		25	228	200	**453**	$9.4	$8.7	$5.8	**$24.0**
33		25	228	201	**454**	$9.4	$8.7	$5.9	**$24.0**
34		25	228	202	**455**	$9.4	$8.7	$5.9	**$24.0**
35		25	228	203	**456**	$9.4	$8.7	$5.9	**$24.0**
36		25	228	204	**457**	$9.4	$8.8	$6.0	**$24.1**
37		25	232	207	**464**	$9.4	$9.0	$6.0	**$24.4**
38		25	232	208	**465**	$9.4	$9.0	$6.1	**$24.4**
39		25	162	176	**363**	$9.4	$6.0	$5.2	**$20.7**
40		25	162	177	**364**	$9.4	$6.0	$5.2	**$20.7**
41		25	157	176	**358**	$9.4	$5.8	$5.2	**$20.5**
42		15	98	147	**260**	$5.8	$3.3	$4.3	**$13.4**
43		15	98	148	**261**	$5.8	$3.3	$4.4	**$13.5**
44		15	98	149	**262**	$5.8	$3.3	$4.4	**$13.5**
45		15	98	150	**263**	$5.8	$3.3	$4.4	**$13.5**
46		15	98	151	**264**	$5.8	$3.3	$4.4	**$13.6**
47		15	98	152	**265**	$5.8	$3.3	$4.5	**$13.6**
48		15	98	153	**266**	$5.8	$3.3	$4.5	**$13.6**
49		15	98	154	**267**	$5.8	$3.3	$4.5	**$13.6**
50		15	98	155	**268**	$5.8	$3.3	$4.6	**$13.7**
51		15	98	156	**269**	$5.8	$3.3	$4.6	**$13.7**
52		15	98	157	**270**	$5.8	$3.3	$4.6	**$13.7**
53		15	98	158	**271**	$5.8	$3.3	$4.6	**$13.7**
54		15	98	159	**272**	$5.8	$3.3	$4.7	**$13.8**
Total						**$729.9**	**$594.6**	**$356.9**	**$1,681.4**

Table 21-27 Annual departmental salary and on-costs

	$'000 pa
Management	
Mining senior staff	$441.0
Shift supervision	$393.5
Techncial Services (planning, geology, geotechncial, dewatering)	$327.0
Operators	
Production driller	$44.1
Primary loading unit	$44.1
Haul truck driver	$44.1
Ancillary plant	$26.4
Other support personnel	$26.4
Maintenance	
Electrical management	$64.1
Electrical technician	$26.4
Mobile plant management	$64.1
Mobile plant technician	$26.4

21.6.2 Processing, general and administration costs

Operating costs were calculated for life of mine and peak concentrate production periods. In peak periods the feed grade will be higher than the LOM average and concentrate production will be maximised. Table 21-28 and Table 21-29 provide a summary of the process operating costs for a plant throughput of 40 Mtpa at peak and LOM concentrate production levels.

Table 21-28 Process and G&A cost summary – peak years

Cost Centre	Total Cost		Fixed Costs		Variable Cost	
	US$/year	US$/t ore	%	US$/year	US$/year	US$/t ore
Labour	33,536,694	0.838	100%	33,536,694	0	0.000
Operating Consumables	95,375,065	2.384	14%	13,778,705	81,596,361	2.040
Power	86,134,491	2.153	34%	29,440,515	56,693,975	1.417
Maintenance Materials and contractors	42,127,343	1.053	81%	33,921,901	8,205,442	0.205
TOTAL PROCESS COSTS	**257,173,593**	**6.429**	**43%**	**110,677,815**	**146,495,778**	**3.662**
General & Administration	57,618,147	1.440	100%	57,618,147	0	0

Table 21-29 Process and G&A cost summary –life of mine average

Cost Centre	Total Cost		Fixed Costs		Variable Cost	
	US$/year	US$/t ore	%	US$/year	US$/year	US$/t ore
Labour	33,536,694	0.838	100%	33,536,694	0	0.000
Operating Consumables	93,633,815	2.341	15%	13,778,705	79,855,110	1.996
Power	83,116,671	2.078	33%	27,026,259	56,090,411	1.402
Maintenance Materials and contractors	42,127,343	1.053	81%	33,921,901	8,205,442	0.205
TOTAL PROCESS COSTS	**252,414,522**	**6.310**	**43%**	**108,263,559**	**144,150,963**	**3.604**
General & Administration	57,618,147	1.440	100%	57,618,147	0	0.000

The costs in the above tables include costs incurred in the production of molybdenum concentrates. As per the Cobre Panamá project, molybdenum production may be delayed for a time until the copper circuit has been successfully commissioned. The costs for molybdenum production have been separated out from the total process operating costs, so that the economics of molybdenum production could be evaluated.

The operating costs are summarised in Table 21-30 and Table 21-31 and are considered to have an accuracy of ±15%.

Table 21-30 Process and GA operating cost summary for copper production alone – peak years

Cost Centre	Total Cost		Fixed Costs		Variable Cost	
	US$/year	US$/t ore	%	US$/year	US$/year	US$/t ore
Labour	32,697,794	0.817	100%	32,697,794	0	0.000
Operating Consumables	91,160,959	2.279	15%	13,778,705	77,382,254	1.935
Power	84,195,662	2.105	33%	27,889,453	56,306,210	1.408
Maintenance Materials and contractors	41,288,526	1.032	81%	33,250,848	8,037,679	0.201
TOTAL PROCESS COSTS	**249,342,942**	**6.234**	**43%**	**107,616,799**	**141,726,143**	**3.543**
General & Administration	57,618,147	1.440	100%	57,618,147	0	0.000

Table 21-31 Molybdenum production cost summary – peak years

Cost Centre	Total Cost		Fixed Costs		Variable Cost	
	US$/year	US$/t ore	%	US$/year	US$/year	US$/t ore
Labour	838,900	0.021	100%	838,900	0	0.000
Operating Consumables	4,214,106	0.105	0%	0	4,214,106	0.105
Power	1,938,828	0.048	80%	1,551,063	387,766	0.010
Maintenance Materials and contractors	838,817	0.021	80%	671,054	167,763	0.004
TOTAL PROCESS COSTS	**7,830,652**	**0.196**	**39%**	**3,061,016**	**4,769,635**	**0.119**

Table 21-32 Process and GA operating cost summary for copper production alone – LOM

Cost Centre	Total Cost		Fixed Costs		Variable Cost	
	US$/year	US$/t ore	%	US$/year	US$/year	US$/t ore
Labour	32,697,794	0.817	100%	32,697,794	0	0.000
Operating Consumables	91,160,959	2.279	15%	13,778,705	77,382,254	1.935
Power	81,177,842	2.029	31%	25,475,197	55,702,646	1.393
Maintenance Materials and contractors	41,288,526	1.032	81%	33,250,848	8,037,679	0.201
TOTAL PROCESS COSTS	**246,325,122**	**6.158**	**43%**	**105,202,543**	**141,122,579**	**3.528**
General & Administration	57,618,147	1.440	100%	0	57,618,147	1.440

Table 21-33 Molybdenum production cost summary – life of mine average

Cost Centre	Total Cost		Fixed Costs		Variable Cost	
	US$/year	US$/t ore	%	US$/year	US$/year	US$/t ore
Labour	838,900	0.021	100%	838,900	0	0.000
Operating Consumables	2,472,855	0.062	0%	0	2,472,855	0.062
Power	1,938,828	0.048	80%	1,551,063	387,766	0.010
Maintenance Materials and contractors	838,817	0.021	80%	671,054	167,763	0.004
TOTAL PROCESS COSTS	**6,089,401**	**0.152**	**50%**	**3,061,016**	**3,028,384**	**0.076**

Consumables costs

Estimated consumables costs are itemised in Table 21-34.

Reagent consumptions have been calculated from the laboratory testwork results, from first principles or have been taken from FQM's database of similar projects. Reagent costs have been sourced from budget quotations and in-house data relating to similar projects in the region.

Crusher liner consumption rates and costs were supplied from the Cobre Panamá operations. A diesel price, delivered to site, of US$1.02 per litre has been adopted.

Mill liner and steel ball consumptions were derived from comminution testwork on ten composite samples of the various ore types at Taca Taca, conducted by ALS laboratories in Kamloops, Canada. Each composite was subjected to comminution SMC testwork to estimate SAG mill sizing and power demand. The results were analysed by OMC, who provided mill liner wear rates, and steel ball consumption rates.

Table 21-34 Consumable costs

Item		Cost to Site	Unit	Consumption	Annual Consumption		TOTAL LOM		TOTAL PEAK	
		US$/Unit		Rate	LOM	Peak	US$/a	US$/t	US$/a	US$/t
Crusher liners										
Gyratory crusher	Concave	192,930	set		8.0 set(s)/y	8.0 set(s)/y	1,543,440	0.039	1,543,440	0.039
Gyratory crusher	Mantle	158,050	set		8.0 set(s)/y	8.0 set(s)/y	1,264,400	0.032	1,264,400	0.032
Pebble crushers	MP 1250	77,500			4.0 set(s)/y	4.0 set(s)/y	310,000	0.008	310,000	0.008
Mill liners										
SAG mill	Steel	2,435,400	set		2.0 set(s)/y	2.0 set(s)/y	4,870,800	0.122	4,870,800	0.122
Ball mill	Steel	1,063,800	set		4.0 set(s)/y	4.0 set(s)/y	4,255,200	0.106	4,255,200	0.106
Regrind mill	Steel	501,400	set		1.0 set(s)/y	1.0 set(s)/y	501,400	0.013	501,400	0.013
Lime ball mill	Steel	50,000	set		1.0 set(s)/y	1.0 set(s)/y	50,000	0.001	50,000	0.001
Subtotal liners							**12,795,240**	**0.320**	**12,795,240**	**0.320**
Grinding media										
SAG mill balls	140 mm	1,560	t	417 g/t ore	16,680 t/y	16,680 t/y	26,020,800	0.651	26,020,800	0.651
Ball mill balls	65 mm	1,410	t	431 g/t ore	17,240 t/y	17,240 t/y	24,308,400	0.608	24,308,400	0.608
Regrind balls	Ceramic beads	5,100	t	0.38 kg/t rough cons	1,520 t/y	1,520 t/y	7,752,000	0.194	7,752,000	0.194
Lime ball mill	50 mm	1,500	t	0.80 kg/t lime	22 t/y	22 t/y	32,784	0.001	32,784	0.001
Subtotal Grinding Media							**58,113,984**	**1.453**	**58,113,984**	**1.453**
Reagents - copper										
Collector	SEX or PAX	5,900	t	16 g/t ore	640.0 t/y	640.0 t/y	3,776,000	0.094	3,776,000	0.094
Frother	MIBC	2,890	t	21 g/t ore	840.0 t/y	840.0 t/y	2,427,600	0.061	2,427,600	0.061
Lime	pH modifier	301	t	683 g/t ore	27,320.0 t/y	27,320.0 t/y	8,223,320	0.206	8,223,320	0.206
Flocculant	Flot tails and conc	3,518	t	25 g/t ore	1,000.0 t/y	1,000.0 t/y	3,518,000	0.088	3,518,000	0.088
Subtotal Copper Reagents							**17,944,920**	**0.449**	**17,944,920**	**0.449**
Reagents - molybdenum										
Collector	Diesel oil	1,019	/kL	28 g/t bulk conc	16 t/y	29 t/y	12,090	0.000	21,705	0.001
Cons dryer fuel	Diesel oil	1,019	/kL	40 g/t bulk conc	319 t/y	319 t/y	238,950	0.006	238,950	0.006
Frother	Polyglycol	2,890	/t	37 g/t bulk conc	21 t/y	38 t/y	61,561	0.002	110,520	0.003
Surfactant	NP 10	42,000	/t	17 g/t bulk conc	10 t/y	18 t/y	411,061	0.010	737,974	0.018
NaHS	NaHS	1,073	/t	15 g/t bulk conc	9 t/y	16 t/y	9,266	0.000	16,635	0.000
pH modifier	Carbon dioxide	783	/t	75 g/t bulk conc	43 t/y	78 t/y	33,809	0.001	60,697	0.002
Depressant	Hexafluoro polymer	2,000	/t	1,021 g/t bulk conc	588 t/y	1,055 t/y	1,175,611	0.029	2,110,564	0.053
Dispersant	Sodium silicate	425	/t	130 g/t bulk conc	75 t/y	134 t/y	31,808	0.001	57,105	0.001
Scrubber solution	Sodium hydroxide	1,600	/t	330 g/t bulk conc	190 t/y	341 t/y	303,978	0.008	545,728	0.014
Flocculant	Concentrate	3,518	/t	25 g/t bulk conc	55 t/y	89 t/y	194,721	0.005	314,228	0.008
Subtotal Molybdenum Reagents							**2,472,855**	**0.062**	**4,214,106**	**0.105**
Filter Consumables										
Filter Cloths		9,809	set		16.0 set(s)/y	16.0 set(s)/y	156,944	0.004	156,944	0.004
Fuel										
Diesel	Mobile equipment	1,019	/kL		1,219 kL/y	1,219 kL/y	1,241,621	0.031	1,241,621	0.031
Water Treatment and Supply										
Water supply cost		0	/year		0.0 lot/y	0.0 lot/y	0	0.000	0	0.000
Potable water		0	kL		500.0 lot/y	500 kL/d	18,250	0.000	18,250	0.000
Waste Treatment										
Allowance		470	/person		1,500 people	1,500 people	705,000	0.018	705,000	0.018
General										
Mill lubricants	Allowance	70,000	lot		1.0 lot/y	1.0 lot/y	70,000	0.002	70,000	0.002
Tools and equipment	Allowance	50,000	lot		1.0 lot/y	1.0 lot/y	50,000	0.001	50,000	0.001
General supplies	Allowance	15,000	lot		1.0 lot/y	1.0 lot/y	15,000	0.000	15,000	0.000
Operator supplies	Allowance	50,000	lot		1.0 lot/y	1.0 lot/y	50,000	0.001	50,000	0.001
Subtotal Miscellaneous							**2,306,815**	**0.058**	**2,306,815**	**0.058**
TOTAL							**93,633,814**	**2.341**	**95,375,065**	**2.384**

Mill liner consumptions were calculated using original liner weights, an assumption that 30% would be scrap at the end of the liner life, and wear rates in terms of kg/t milled as estimated by OMC. These calculations, however, showed unrealistic liner life (typically three months) relative to industry norms. From operating practice at the Company's similar Sentinel operation, a full liner set per mill, per annum was allowed for Taca Taca. This is considered to be conservative. Liner weights were taken from drawings for the Sentinel mill liners (same size mills) and costs for these liners were taken as $2,700/t of liner mass.

Steel ball consumption was estimated by OMC, using costs from Sentinel. The regrind mill ceramic media costs were provided by Cobre Panamá.

Costs at Sentinel for 2019 were $0.82/t for grinding media and $0.48/t for reagents. Reagent costs at Taca Taca are higher because of the high consumption of frother, indicated by the testwork, and the inclusion of reagents (e.g. NaHS) for molybdenum recovery.

Energy costs

Energy will be supplied to the processing plant from the local power grid, via a new transmission line installed for the Project. Unit costs of $65/MWh have been used in this estimate, following recent discussions with CAMMESA.

Energy consumption was calculated from the electrical load list generated as part of the mechanical equipment specification developed by Lycopodium. Where power consumption estimates for NPI facilities was not available, allowances were made based on FQM database information.

The updated energy cost estimate is listed in Table 21-35 and Table 21-36.

Table 21-35 Energy cost estimate breakdown for peak years

	Installed Power	Continuous Draw	Annual Energy Consumption	Total Annual Power Cost	
	kW	kW	kWh/year	US$/year	US$/t ore
Crushing, Screening & Stockpiling	6,646	4,715	41,300,159	2,684,510	0.067
Stockpile Reclaim & Milling	130,865	106,802	935,587,870	60,813,212	1.520
Flotation	28,725	21,002	183,978,834	11,958,624	0.299
Concentrate Handling	3,157	1,757	15,387,816	1,000,208	0.025
Tailings Disposal	17,010	8,227	72,064,140	4,684,169	0.117
Mo Flot and Cons Handling	5,561	3,405	29,828,129	1,938,828	0.048
Reagents	1,552	987	8,648,660	562,163	0.014
Plant Services	7,858	4,378	38,350,404	2,492,776	0.062
TOTAL PROCESS ENERGY COSTS	**201,373**	**151,272**	**1,325,146,012**	**86,134,491**	**2.153**
Village	0	0	3,000,000	195,000	0.005
Workshops / Plant Buildings	2,000	1,000	8,760,000	569,400	0.014
Airport / other infrastructure	200	100	876,000	56,940	0.001
Miscellaneous	500	500	4380000	284700	0.007
G&A ENERGY COSTS	**2,700**	**1,600**	**17,016,000**	**1,106,040**	**0.028**

Table 21-36 Energy (power) cost estimate breakdown for LOM average

	Installed Power	Continuous Draw	Annual Energy Consumption	Total Annual Power Cost	
	kW	kW	kWh/year	US$/year	US$/t ore
Crushing, Screening & Stockpiling	6,646	4,715	41,300,159	2,684,510	0.067
Stockpile Reclaim & Milling	130,865	106,802	935,587,870	60,813,212	1.520
Flotation	28,725	15,702	137,550,834	8,940,804	0.224
Concentrate Handling	3,157	1,757	15,387,816	1,000,208	0.025
Tailings Disposal	17,010	8,227	72,064,140	4,684,169	0.117
Mo Flot and Cons Handling)	5,561	3,405	29,828,129	1,938,828	0.048
Reagents	1,552	987.29	8,648,660	562,163	0.014
Plant Services	7,858	4,378	38,350,404	2,492,776	0.062
TOTAL PROCESS ENERGY COSTS	**201,373**	**145,972**	**1,278,718,012**	**83,116,671**	**2.078**
Village			3,000,000	195,000	0.005
Workshops / Plant Buildings	2,000	1,000	8,760,000	569,400	0.014
Airport / other infrastructure	200	100	876,000	56,940	0.001
Miscellaneous	500	500	4,380,000	284,700	0.007
G&A ENERGY COSTS	2,700	1,600	17,016,000	1,106,040	0.028

Maintenance costs

Maintenance materials costs for the operation have been factored from the capital cost estimate, using specific factors from the FQM database and industry standards.

The allowance covers mechanical spares and wear parts, but excludes crushing and grinding wear components, media and general consumables. The maintenance costs exclude all payroll maintenance labour (covered in plant labour) but does include a cost for contract labour taken from the Sentinel operating costs. The estimated costs are listed in Table 21-37.

Mobile equipment costs

Mobile equipment costs provide for the fuel and lease of the light vehicles, portable generators and other mobile equipment for the site. It has been assumed that all mobile equipment is leased, and that the lease costs include major maintenance items.

The fuel costs have been included in the consumables cost estimate. Sentinel maintenance costs for the process plant are $1.15/t, which compares with the estimated cost of $1.05/t listed in Table 21-37.

Table 21-37 Maintenance cost estimate

Area	Capital Cost (supply)	Maintenance factor	Maintenance Cost	
	US$	%	US$/year	US$/t ore
Plant Maintenance				
Plant general	34,524,987	2.98%	1,028,306	0.026
Crushing, Screening & Stockpiling	38,780,409	4.82%	1,868,839	0.047
Stockpile Reclaim & Milling	265,615,958	6.14%	16,300,476	0.408
Flotation	116,451,807	4.10%	4,769,368	0.119
Concentrate Handling	36,621,002	3.83%	1,402,050	0.035
Tailings Disposal	68,968,125	4.38%	3,017,804	0.075
Molybdenum Recovery	17,893,374	4.69%	838,817	0.021
Reagents	8,840,784	3.21%	284,070	0.007
Plant Services	13,377,491	3.25%	434,225	0.011
Rail head	10,965,956	1.67%	183,388	0.005
Contractors			12,000,000	0.300
Mobile Equipment		Included in lease costs		
TOTAL PLANT MAINTENANCE COSTS	**612,039,893**		**42,127,343**	**1.053**
G&A Maintenance				
Non-process infrastructure	584,035,987	1.76%	10,277,773	0.257
Maintenance software (SAP etc)			250,000	0.006
Maintenance manuals			150,000	0.004
Maintenance training			100,000	0.003
TOTAL G&A MAINTENANCE COSTS		**3.40%**	**10,777,773**	**0.269**

Labour costs

Administration, process plant management, supervisor, operator, maintenance and NPI personnel requirements were estimated according to the personnel numbers at similar sized FQM projects, i.e. Sentinel and Cobre Panamá.

Labour rates were supplied by the Administration and Finance Superintendent for FQM in Salta, and on-costs of 30% were assumed.

Workers will be flown from Salta to site and bussed to and from the accommodation village. Workers will work a two week on, two week off schedule. Expatriate staff will work a six week on, two weeks off roster.

The labour cost for the Project excludes all head office personnel. Laboratory personnel have been included.

All workers will be housed in the accommodation village at site. The village will be constructed and owned by FQM, and allowance was made for the staff to operate and maintain it, included under administration labour costs. Camp operating costs were assumed to be $35/per person/per day. For expatriate staff, the labour costs also include thirteen economy class return flights per year between Argentina and the place of hiring at US$5,000 per return flight. Table 21-38 summarises the labour complement and the associated operating cost estimate.

Table 21-38 Labour cost summary

Area	Employees	Labour US$/year	Camp US$/year	Flights US$/year	FIFO US$/year	Total US$/year
Process Plant Operations – copper plant	254	13,516,000	2,433,638	660,400	162,500	16,772,538
Process Plant Operations – molybdenum plan	16	644,000	153,300	41,600	0	838,900
Process Plant Maintenance	189	13,493,000	1,810,856	491,400	130,000	15,925,256
TOTAL PLANT LABOUR COST	**459**	**27,653,000**	**4,397,794**	**1,193,400**	**292,500**	**33,536,694**
Administration	286	14,017,300	2,740,238	743,600	195,000	17,696,138
Non-process infrastructure	142	7,683,800	1,360,538	369,200	130,000	9,543,538
TOTAL G&A LABOUR COSTS	**428**	**21,701,100**	**4,100,775**	**1,112,800**	**325,000**	**27,239,675**

Administration labour costs

The estimated administration labour requirements relate to a day shift team, with assistant-level staff supporting senior positions to ensure continuous coverage during breaks.

The following disciplines have been included in the administration department:

- management
- Salta office
- safety, health & environment (SHE)
- human resources
- security
- finance and administration
- IT
- camp and aerodrome
- community relations

It is assumed that the Salta office (i.e., SOC) will have a small staff contingent. The SHE department will include an on-site nurse working day shift and first aid officers working on shift. The accommodation and aerodrome facilities will cater for a 2,000 bed camp, with 1,500 people on site at any one time[15].

Plant labour costs

The estimate of the plant labour contingent was determined for a four-panel operation (two shifts working 12 hours per day, two rotation shifts), to provide continuous coverage for the plant operation with allowance for leave and absenteeism coverage.

For cost estimation purposes the plant was divided into five main operating areas:

- crushing
- grinding
- flotation
- services (reagents)
- tailings management

[15] Cobre Panamá had a 10,000 bed camp and 1,000 camp staff, with 195 staff for Taca Taca, hence the ratio of staff to beds is appropriate.

Each area would be managed by an area superintendent on day shift, with operators, attendants and labourers in each section on shift. The shifts would be controlled by a supervisor, whilst an additional supervisor would be assigned to the tailings management facility.

Maintenance labour costs

The maintenance labour estimate has been built-up in a similar manner to the plant complement estimate, with maintenance supervisors on dayshift in charge of the maintenance for individual sections of the plant.

These supervisors would each have a team of mechanics, boilermakers, electricians and instrumentation technicians who would mainly work on day shift. However, a sole mechanic, electrician and instrument technician would each be assigned to every shift to deal with any unplanned maintenance issues.

The total process and maintenance complement is estimated to be 459. By comparison, the Sentinel labour complement was 365 in 2019 (including 43 positions filled by contractors). Taca Taca has additional labour to allow for the molybdenum plant and for the equal time roster.

Non-Process Infrastructure (NPI) labour costs

The estimated NPI labour requirements relate primarily to a day shift team. The following disciplines have been included in the NPI department:

- NPI management
- rail
- water and waste treatment (including water intake)
- NPI electrical
- NPI mechanical
- port operations

The rail team will focus on operation of the rail loadout for concentrate export and reagent import.

The water team will be responsible for water intake from the various borefields, and for the water and sewage treatment plants.

NPI electrical and mechanical personnel will be responsible for operation of the supply power lines, installations on the borefields and light vehicle maintenance.

General and administration costs

Administration cost estimates were developed from first principal calculations and historical in-house data for similar sized operations. These costs do not include administration labour, royalty or treatment/refining charges. The former are included in labour and the latter are included in the metal costs estimate.

Table 21-39 shows the contribution of the cost categories to the total administration cost. An allowance for laboratory assays for plant control and concentrate export samples was made.

Table 21-39 Administration expenses cost estimate

	$/year	$/t
Site office		
Telecommunications	600,000	0.015
Telecommunications maintenance	90,000	0.002
Stationery	90,000	0.002
Postage, courier and light freight	85,000	0.002
Computer supplies	1,000,000	0.025
Insurances		
Light vehicles	130,000	0.003
Financial		
Banking charges	550,000	0.014
Legal fees	45,000	0.001
Auditing costs	90,000	0.002
Fees		
Tenement maintenance charges	70,000	0.002
Tenement fees	100,000	0.003
Community relations expenses	1,000,000	0.025
Consultants		
Environmental consultants	150,000	0.004
Other consultants	220,000	0.006
Metallurgical testing	500,000	0.013
Environmental compliance testing	40,000	0.001
Personnel		
First aid and medical costs	443,500	0.011
Medicals, visas, passports	1,200,000	0.030
Recreational and local facilities	300,000	0.008
Entertainment	500,000	0.013
Safety clothing	443,500	0.011
Travel and accommodation	217,000	0.005
Recruiting/relocation	740,312	0.019
Training	200,000	0.005
General		
Site laboratory	1,112,348	0.028
Mobile equipment lease costs	7,878,000	0.197
Services	500,000	0.013
Miscellaneous	200,000	0.005
TOTAL	**18,494,660**	**0.462**

21.7 Metal costs

In addition to royalties, metal costs for each of the copper and molybdenum concentrates comprise:

- concentrate treatment and refining charges
- concentrate transport charges
- port handling charges

Information on applicable treatment and refining charges was provided by the Company's internal metals marketing group.

Estimated concentrate (rail) transport charges were adopted from preliminary tenders submitted by rail operators, whilst port handling and sea freight charges were averaged estimates referencing preliminary enquiries by the Company.

21.7.1 Concentrate treatment and refining charges

The following advised treatment and refining charges have been adopted:

- copper concentrate:
  - treatment = $85/dmt of concentrate
  - refining = $0.08/lb of contained copper metal
- molybdenum concentrate:
  - treatment = $69.19/dmt of concentrate
- gold (in copper concentrate):
  - refining = $5.10/(t)oz

21.7.2 Concentrate transport charges

Argentina based personnel obtained preliminary cost estimates from operators within Argentina and Chile. This information was adopted for the estimate of transport costs for concentrate from site to a port located on the coast in Chile. These costs are provided in Table 21-40. The railways tariff is based on external consultant assessments and modelling for the freight of 1 Mt/a concentrate.

Table 21-40 Operating cost estimate for rail transport of copper concentrate

Railway	Costs	Fixed	Annual Cost	
	US$	US$	US$	US$/t
Fixed component	25,454,045		25,454,045	
Variable fee	39.4864	1,000,000	39,486,400	
Total			**64,940,445**	**64.94**

An approximate cost of $78/t for molybdenum concentrate has been adopted assuming that the product would be freighted in two tonne bulk bags.

21.7.3 Port and shipping costs

Benchmarked prices for port logistics and concentrate handling charges is in the order of $13 to $17/dmt of concentrate.

Initial discussions with shipping operators indicated that sea freight charges would be in the order of $35.00 to $39.00/t for shipment to Korea, Japan or China. As such, an average cost of $38.00/t of concentrate was adopted to provide the consolidated shipping costs shown in Table 21-41.

Table 21-41 Concentrate transport costs

	Units	Peak years	LOM
Cu concentrate produced	t/a	1,033,577	575,715
Mo concentrate produced	t/a	3,573	2,214
Total concentrate produced		1,037,149	577,929
Cost per tonne copper concentrate for rail	USD/t	65.241	65.241
Cost per tonne molybdenum concentrate for rail	USD/t	78.289	78.289
Cost per tonne for port	USD/t	15.00	15.00
Cost per tonne for shipping (to China)		38.00	38.00
Transport cost – copper	**USD/annum**	**122,211,316**	**68,073,267**
Transport cost - molybdenum	**USD/annum**	**469,071**	**290,675**

ITEM 22 ECONOMIC ANALYISIS

In accordance with Part 2.3 (1) (c) of the Rules and Policies of Canadian National Instrument (NI) 43-101, the economic analysis set out below does not include Inferred Mineral Resources.

The economic analysis in the form of a basic cashflow model is intended to support the Mineral Reserve estimate, and to demonstrate a positive cashflow for mining and processing. The Project development, ongoing capital and sustaining costs are included in the analysis for completeness. The model is provided pre-tax and post-tax.

22.1 Methodology and key assumptions

The basic methodology adopted for the economic analysis was to tabulate the detailed 40 Mtpa Stage 1 production schedule (ore and waste mined, ore processed, stockpiles reclaimed) and the recovered metal profiles for the production schedule described in Item 16. To reflect the gradual attainment of the designed plant recovery during plant ramp-up in the first two years of the Project, downgrade factors were adopted as per similar allowances applied to the Cobre Panamá ramp-up period. Linked to the recovered (and down-rated) metal profile were the payability assumptions and the metal prices, to then arrive at annual gross revenues.

Treatment charges and refining charges (metal costs for copper, molybdenum and gold realisation) were next calculated, inclusive of royalties, to then arrive at annual net revenues. Initial development and subsequent capital costs, in addition to sustaining capital costs and operating costs, were deducted from the net revenue to finally arrive at an annual cashflow.

Key assumptions for the Stage 1 40 Mtpa economic analysis are as follows:

- the Project commences with a mining pre-strip from Year 1 until Year 4, with copper (and gold) production commencing in Year 5, to be followed by molybdenum production commencing in Year 6
- the plant feed tonnes and grade account for mining dilution and mining recovery factors
- the initial Project development capital is expended in Years 1 to 5
  - the mining spend during this period includes related infrastructure and initial fleet purchases
  - the assumed spending intensity on the processing plant, rail facilities and other Project infrastucture is 13% in Year 1, 18% in Year 2, 30% in Year 3, 28% in Year 4 and 11% in Year 5
- initial production starts in Year 5 at 24.9 Mt, ramping up to 40 Mtpa from Year 6 and continuing at that level until Year 54
- the first two years of processing feature down-rated processing recoveries
- costs associated with pre-strip mining volumes in Years 1 to 4 are assumed to be capitalised
- capital costs incurred during the life of operations account for:
  - new and replacement mining equipment, including mining equipment change-out costs
  - an additional sustaining cost allowance for other Project infrastructure
  - deferred mine stripping costs
- the annual mine operating costs reflect the mining production schedule, inclusive of stockpile reclaim, less deferred stripping costs
- the other annual operating costs (i.e., processing and G&A unit costs) were not profiled for each year
- the treatment and product freight (i.e., metal) costs were profiled against varying concentrate grade and payability factors

- the cashflows consider royalties, notional capital depreciation schedules and the applicable corporate tax rate

An economic analysis has been provided for a preliminary Stage 2 60 Mtpa production schedule (Item 16.4). The capital and operating costs associated with this expansion schedule are also preliminary and have not been estimated to the same level of detail as those reported under 20.7.

22.1.1 Metal pricing

The annual revenues in the two cashflow models were calculated referencing late 2025 consensus pricing information. As such, a consistent long term price of $4.50/lb copper was adopted for modelling, as opposed to $3.50/lb that was adopted for the pit optimisation (Item 15.3.4).

A long term price of $1,200/oz gold was adopted for the pit optimisation (Item 15.3.4), whereas $3,000/oz gold was adopted subsequently for the cashflow model. A long term price of $18.00/lb for molybdenum was adopted for the cashflow model, compared with $12.00/lb for the earlier optimisation.

The impact of these pricing changes relative to the original pit optimisation is discussed in Item 15.

22.1.2 Regulatory and fiscal environment (including RIGI incentive regime)

In July 2024, Argentina enacted the “Bases Law,” which includes the Incentive Regime for Large Investments (RIGI). This framework establishes a series of tax, customs, and foreign-exchange incentives for new investment projects exceeding USD 200 million, applicable to mining, energy, oil and gas, forestry, steel, technology, tourism, and infrastructure sectors. Eligible projects must meet a minimum computable investment of USD 200 million, of which at least USD 80 million (40%) must be disbursed within the first two years following adhesion to the regime. Projects admitted to the RIGI receive 30 years of stability across tax, customs, and foreign-exchange matters. Applications may be submitted until July 2026, with the possibility of a one-year extension if deemed necessary by the government. The province of Salta has formally adhered to the RIGI, aligning its provincial tax framework with the national regime.

Key incentives include the reduction of the Corporate Income Tax rate to 25%, accelerated refund of VAT through transferable Tax Credit Certificates, the option to apply accelerated depreciation for movable assets and mining infrastructure, and the ability to carry forward tax losses indefinitely. In addition, the Bank Debits and Credits Tax becomes 100% creditable against Corporate Income Tax.

The regime provides an exemption from import duties on capital goods, equipment, spare parts, and inputs required during the construction phase. Export duties are eliminated starting in the third year following adhesion to the RIGI. It is worth noting, however, that for copper concentrate specifically, the National Government has already reduced the export duty rate to 0% as from January 2025, pursuant to Decree 674/2024, regardless of RIGI’s phased export-duty exemption schedule.

Regarding foreign-exchange rules, the RIGI grants a progressive liberalisation of export proceeds, which may be retained abroad without an obligation to repatriate them: 20% from Year 2, 40% from Year 3, and 100% from Year 4. It also authorises access to foreign currency for the repayment of loans and the distribution of dividends, provided that the funds corresponding to capital contributions and loan disbursements have been previously brought into Argentina through the official foreign exchange market.

Taxes not specifically covered or exempted under the RIGI remain subject to the general Argentine tax regime. In particular, the 3% mining royalty on pit-head value remains in force, along with other federal, provincial, or municipal taxes not explicitly exempted.

The regime also includes a specific provision aimed at promoting local supplier development, requiring that at least 20% of goods and works contracted involve qualified local suppliers.

22.1.3 Private royalties

The applicable Taca Taca Mining Group concessions are subject to a contractual royalty of 1.5% of net smelter return (the Taca Taca royalty). In modelling these royalty charges, revenue was assumed to be calculated on payable metal. An export levy, as a net revenue deduction, was not included in the cashflow model. Dividend withholding tax was also excluded.

22.1.4 Payable metal factors

The payable metal factors adopted in the cashflow models were assumed to be:

- recovered copper = 96.1%
- recovered molybdenum = 86.0%
- recovered gold = 90.0%

22.2 Cashflow model inputs

22.2.1 Production schedule

The Stage 1 40 Mtpa production schedule forming the basis of the Mineral Reserve cashflow model is the same as that listed in Item 16.2.5, apart from a minor adjustment in the final years of processing. The modified 40 Mtpa production schedule is listed in Table 22-1.

The listed metal recovered in Years 5 and 6 is after the application of downrated recovery factors for those particular years.

The preliminary Stage 2 60 Mtpa production schedule (plant feed tonnes and grade) is listed within the related cashflow model in Table 22-11.

22.2.2 Processing recoveries

From the metallurgical testwork results and the mine production schedules; the following average life of mine recoveries are evident in the cashflow modelling:

- copper recovery of 87.1% to a concentrate grade of 25.7 %Cu
- molybdenum recovery of 44.3% to a molybdenum concentrate grade of 47% Mo
- gold recovery to the copper concentrate of 61.4%

To reflect the modelling of initial cashflows as per the Cobre Panamá project, copper recovery ramp-up factors were also applied to the Taca Taca model, as follows:

- Year 1 = 85% of the modelled recovery; Year 2 = 95% of the modelled recovery and Year 3 = 100% of the modelled recovery.

The impact of this factoring is a reduction in the overall average life of mine copper recovery from 87.1% to 86.7%.

Table 22-1 Life of mine production schedule for Stage 1 40 Mtpa processing (cashflow modelling)

Year	Stage	Mining						Processing				Metal Recovered		
		Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)	Waste (Mt)	Total Mined (Mt)	Ore (Mt)	Cu (%)	Mo (ppm)	Au (g/t)	Cu (kt)	Mo (kt)	Au (k(t)oz)
1						19.3	19.3							
2	Pre-strip					94.3	94.3							
3		0.5	0.18	20.34	0.06	146.7	147.3							
4		9.9	0.28	34.41	0.06	161.7	171.6							
5		52.3	0.37	52.72	0.07	143.0	195.2	24.9	0.61	71.60	0.09	112.4	0.0	38.5
6		51.5	0.59	80.74	0.10	143.8	195.2	40.0	0.73	92.31	0.11	240.6	1.4	84.1
7		45.5	0.77	110.81	0.14	150.3	195.8	40.1	0.78	110.56	0.14	271.5	1.8	109.6
8		42.4	0.69	136.20	0.12	152.8	195.2	40.0	0.78	119.50	0.13	271.9	2.0	97.9
9		52.7	0.67	182.71	0.14	142.5	195.2	40.0	0.80	178.74	0.15	283.1	3.0	113.8
10		46.7	0.67	196.76	0.17	148.5	195.2	40.0	0.79	206.76	0.18	282.8	3.7	137.9
11		56.2	0.56	118.68	0.11	139.5	195.8	40.1	0.71	147.23	0.13	246.8	2.7	102.1
12		69.9	0.42	87.35	0.09	125.4	195.2	40.0	0.60	127.53	0.12	206.9	2.3	96.8
13		57.8	0.40	79.91	0.08	137.4	195.2	40.0	0.50	102.32	0.10	170.7	2.0	79.3
14		55.0	0.37	84.69	0.08	140.2	195.2	40.0	0.46	103.02	0.10	160.0	2.0	81.4
15		57.8	0.39	114.85	0.10	137.9	195.8	40.1	0.50	132.91	0.11	172.3	2.3	88.0
16		63.4	0.40	108.27	0.09	116.8	180.2	40.0	0.52	128.16	0.12	181.7	2.4	92.2
17		74.0	0.43	118.59	0.11	82.0	156.0	40.0	0.59	136.43	0.13	207.1	2.4	103.2
18		73.8	0.44	111.62	0.10	58.0	131.8	40.0	0.59	129.65	0.12	207.4	2.4	97.7
19		57.3	0.42	110.48	0.11	59.8	117.1	40.1	0.59	132.26	0.14	207.4	2.5	112.4
20		51.9	0.42	96.65	0.09	64.9	116.8	40.0	0.50	110.42	0.11	173.6	2.1	87.3
21		48.5	0.42	98.30	0.09	68.3	116.8	40.0	0.51	109.82	0.10	174.7	2.1	79.9
22		43.6	0.37	149.35	0.10	73.0	116.6	40.0	0.41	132.48	0.10	142.7	2.5	78.8
23		55.5	0.40	130.07	0.09	43.3	98.8	40.1	0.47	141.05	0.11	163.4	2.6	84.7
24		56.6	0.35	139.17	0.08	41.9	98.5	40.0	0.46	133.11	0.09	157.9	2.3	67.2
25		60.9	0.34	106.43	0.07	37.6	98.5	40.0	0.45	110.43	0.08	154.1	1.8	62.2
26		60.3	0.38	112.94	0.07	38.2	98.5	40.0	0.47	126.75	0.09	162.2	2.2	67.9
27		62.6	0.31	135.46	0.07	36.2	98.8	40.1	0.42	148.16	0.08	148.2	2.6	66.2
28		59.2	0.27	107.75	0.06	39.3	98.5	40.0	0.34	118.42	0.08	120.2	2.1	60.9
29		68.4	0.31	110.86	0.06	30.1	98.5	40.0	0.42	106.08	0.07	149.0	2.1	55.8
30	Production	67.0	0.33	115.16	0.05	31.5	98.4	40.0	0.45	108.85	0.06	158.6	2.2	49.6
31		56.0	0.32	129.17	0.07	19.4	75.4	40.1	0.39	134.14	0.08	138.6	2.6	65.3
32		49.5	0.33	145.94	0.07	10.6	60.2	40.0	0.38	146.25	0.08	134.8	2.8	64.6
33		47.5	0.36	132.79	0.07	12.7	60.2	40.0	0.40	128.82	0.08	141.7	2.4	63.3
34		45.8	0.37	134.65	0.06	14.4	60.2	40.0	0.42	134.36	0.07	146.0	2.4	59.0
35		47.0	0.37	146.59	0.07	13.3	60.3	40.1	0.42	146.18	0.08	148.9	2.7	61.2
36		50.9	0.43	166.00	0.07	9.2	60.2	40.0	0.51	172.74	0.08	181.5	3.0	61.1
37		53.1	0.42	170.87	0.07	7.1	60.2	40.0	0.50	175.16	0.08	176.9	2.9	65.6
38		52.7	0.41	155.48	0.08	5.5	58.2	40.0	0.47	161.40	0.09	167.2	2.7	68.5
39		27.8	0.44	166.93	0.07	2.8	30.6	40.1	0.40	156.62	0.08	141.8	2.7	59.2
40		27.7	0.54	163.31	0.08	2.5	30.2	40.0	0.44	151.99	0.08	155.1	2.6	62.7
41		30.9	0.56	162.92	0.07	1.5	32.4	40.0	0.48	154.74	0.08	168.5	2.7	59.1
42								40.0	0.21	130.66	0.08	71.6	2.3	62.3
43								40.1	0.21	130.41	0.08	71.5	2.3	61.3
44								40.0	0.21	130.11	0.08	71.0	2.3	59.7
45								40.0	0.21	130.11	0.08	71.0	2.3	59.7
46								40.0	0.13	77.70	0.03	45.5	1.4	28.0
47								40.1	0.13	74.91	0.03	44.2	1.3	26.4
48								40.0	0.13	74.91	0.03	44.1	1.3	26.3
49								40.0	0.13	74.91	0.03	44.1	1.3	26.3
50								40.0	0.13	74.91	0.03	44.1	1.3	26.3
51								40.1	0.13	74.91	0.03	44.2	1.3	26.4
52								40.0	0.13	74.91	0.03	44.1	1.3	26.3
53								40.0	0.13	74.91	0.03	44.1	1.3	26.3
54								40.0	0.13	74.91	0.03	39.8	1.2	23.7
55								3.9	0.13	74.91	0.03	4.3	0.1	2.6
Total		**1,990.1**	**0.42**	**122.8**	**0.09**	**2,903.4**	**4,893.5**	**1,990.1**	**0.42**	**122.8**	**0.09**	**7,311.7**	**108.2**	**3,367.0**

22.2.3 Capital and sustaining costs

40 Mtpa operation

The total estimated capital costs for the Project were spilt into initial (i.e., development) capital expenditure and subsequent operational expenditure, as listed in Table 22-2. The totals in Table 22-2 are captured in the cashflow model and are consistent with the itemisation of costs listed in Item 21.

As additional input to the cashflow model, Table 22-2 also summarises the mining equipment and processing/infrastructure sustaining costs listed in Item 21.

60 Mtpa operation

Table 22-3 lists the additional capital costs incurred in Years 6 to 8 of $1,018.7 M, associated with an expansion to 60 Mtpa processing. This table also lists the estimated operational expenditure and sustaining costs associated with the expanded production scenario.

22.2.4 Operating costs

Between the time of estimating in detail the mining, processing and G&A costs as set out in Item 21, additional costs were identified by the Company's finance team to account for specific operating and departmental costs which are typical at other FQM operations of comparable size and scale. These 'adjustments and indirects' include such as processing labour and maintenance material costs, departmental administration, equipment hire, freight on procurement, training and other costs. These additional costs listed in Table 22-4, Table 22-5 and Table 22-6 have been included in the cashflow models.

40 Mtpa operation

The overall average unit operating costs evident in the cashflow model for the 40 Mtpa Stage 1 are:

- pre-strip mining ore and waste:
  - $1.74/t average before adjustments and indirects (range: $1.67/t to $2.44/t)
  - $1.84/t average after adjustments and indirects (range: $1.75/t to $3.01/t)
- operations mining of ore and waste (including stockpile reclaim):
  - $1.83/t average before adjustments and indirects (range: $0.91/t to $2.61/t)
  - $1.93/t average after adjustments and indirects (range: $0.97/t to $2.80/t)
- processing:
  - $6.87/t before indirects
  - $7.25/t average after indirects
- G&A:
  - $1.52/t processed, average after indirects
- water supply tariff and sundry taxes:
  - effectively $0.05/t processed on average, based on current tarff pricing
- C1 costs and AISC:
  - $1.14/lb Cu and $1.59/lb Cu for the first 20 years of operation
  - $1.39/lb Cu and $1.74/lb Cu for the life of mine

Table 22-2 Project capital and sustaining cost estimates for Stage 1 40 Mtpa cashflow modelling

	UNITS	Y1	Y2	Y3	Y4	Y5	Subtotal	Y5 - Y9	Y10 -Y14	Y15 - Y19	Y20 - Y24	Y25 - Y29	Y30 - Y34	Y35 - Y39	Y40 - Y44	Y45 - Y49	Y50 - Y54	Y55	TOTAL
Project development capital																			
Mining																			
Mining pre-strip ore and waste	$'000	$58,275	$179,660	$256,364	$260,370		**$754,669**												**$754,669**
Mining infrastructure	$'000	$219,565	$318,314	$197,468	$168,327	$97,608	**$1,001,281**												**$1,001,281**
Subtotal mining	**$'000**	**$277,840**	**$497,974**	**$453,832**	**$428,697**	**$97,608**	**$1,755,950**												**$1,755,950**
Processing	$'000	$145,238	$191,291	$325,802	$297,867	$117,283	**$1,077,482**												**$1,077,482**
Rail	$'000	$1,722	$2,268	$3,863	$3,532	$1,391	**$12,776**												**$12,776**
Infrastructure	$'000	$98,001	$129,076	$219,838	$200,989	$79,138	**$727,043**												**$727,043**
Indirects	$'000	$88,758	$116,902	$199,104	$182,032	$71,674	**$658,469**												**$658,469**
Total development capital	**$'000**	**$611,560**	**$937,511**	**$1,202,439**	**$1,113,117**	**$367,093**	**$4,231,720**												**$4,231,720**
Operational capital																			
Deferred stripping	$'000							$667,890	$506,345	$140,551	$18,291	$0	$0	$0	$0	$0	$0	$0	**$1,333,076**
Closure costs	$'000							$0	$0	$0	$0	$0	$0	$0	$0	$0	$61,933	$9,763	**$71,696**
Total operational capital	$'000							**$667,890**	**$506,345**	**$140,551**	**$18,291**	**$0**	**$0**	**$0**	**$0**	**$0**	**$61,933**	**$9,763**	**$1,404,772**
Sustaining costs																			
Mining	$'000							$424,662	$466,267	$358,201	$351,581	$298,552	$145,373	$110,954	$63,980	$123,221	$61,783	$0	**$2,404,574**
Processing and infrastructure	$'000							$62,575	$65,000	$65,013	$64,990	$64,990	$65,000	$65,013	$64,948	$64,948	$64,958	$1,568	**$649,002**
Total sustaining costs	**$'000**							**$487,236**	**$531,267**	**$423,214**	**$416,571**	**$363,542**	**$210,373**	**$175,966**	**$128,928**	**$188,169**	**$126,741**	**$1,568**	**$3,053,576**
Total capital and sustaining costs	**$'000**	**$611,560**	**$937,511**	**$1,202,439**	**$1,113,117**	**$367,093**	**$4,231,720**	**$1,155,126**	**$1,037,612**	**$563,765**	**$434,861**	**$363,542**	**$210,373**	**$175,966**	**$128,928**	**$188,169**	**$188,674**	**$11,331**	**$8,690,067**

Table 22-3 Project capital and sustaining cost estimates for Stage 2 60 Mtpa cashflow modelling

	UNITS	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Subtotal	Y5 - Y9	Y10 -Y14	Y15 - Y19	Y20 - Y24	Y25 - Y29	Y30 - Y34	Y35 - Y39	TOTAL
Project expansion capital																			
Mining	$'000						$105,219				**$105,219**								**$105,219**
Processing	$'000						$79,309	$237,927	$211,490		**$528,726**								**$528,726**
Infrastructure	$'000						$32,818	$98,455	$87,516		**$218,789**								**$218,789**
Indirects	$'000						$24,894	$74,683	$66,385		**$165,962**								**$165,962**
Subtotal development capital	**$'000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$242,240**	**$411,064**	**$365,391**	**$0**	**$1,018,695**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$1,018,695**
Operational capital																			
Deferred stripping	$'000										**$0**	$824,792	$193,576	$225,002					**$1,243,370**
Closure costs	$'000										**$0**							$77,803	**$77,803**
Subtotal operational capital	**$'000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$824,792**	**$193,576**	**$225,002**	**$0**	**$0**	**$0**	**$77,803**	**$1,321,173**
Sustaining costs																			
Mining	$'000										**$0**	$450,327	$554,113	$473,786	$436,485	$209,149	$87,833	$55,591	**$2,267,284**
Processing and infrastructure	$'000										**$0**	$69,502	$89,927	$89,913	$89,913	$89,913	$89,927	$86,250	**$605,346**
Subtotal sustaining costs	**$'000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$519,829**	**$644,040**	**$563,699**	**$526,398**	**$299,062**	**$177,761**	**$141,841**	**$2,872,630**
Total expansion and sustaining costs	**$'000**	**$0**	**$0**	**$0**	**$0**	**$0**	**$242,240**	**$411,064**	**$365,391**	**$0**	**$1,018,695**	**$1,344,620**	**$837,616**	**$788,702**	**$526,398**	**$299,062**	**$177,761**	**$219,644**	**$5,212,498**

Table 22-4 Unit mining costs for cashflow modelling, before and after adjustments

Mining cost centres (40 Mtpa)	Pre-strip ($M)	Ops ($M)	LOM ($M)	Unit costs ($/t)
LCC	$232.4	$2,808.6	$3,041.1	$0.56
Diesel	$157.7	$2,070.3	$2,228.1	$0.41
Electricity	$22.3	$557.7	$580.0	$0.11
Tyres	$44.4	$583.8	$628.2	$0.12
Drill & Blast	$162.0	$1,981.6	$2,143.6	$0.44
Labour	$132.5	$1,548.9	$1,681.4	$0.31
subtotal	**$751.4**	**$9,550.9**	**$10,302.3**	**$1.94**
Adjustments	-$39.1	$0.0	-$39.1	-$0.01
subtotal	**$712.3**	**$9,550.9**	**$10,263.2**	**$1.94**
Plus indirects	$42.4	$454.7	$497.1	$0.09
subtotal	**$754.7**	**$10,005.6**	**$10,760.3**	**$2.03**
Less deferred stripping		-$1,333.1		
total	**$754.7**	**$8,672.5**	**$9,427.2**	**$2.03**

Table 22-5 Unit processing costs for cashflow modelling, before and after adjustments

Processing cost centres (40 Mtpa)	Fixed cost		Variable cost		Total cost	
	%	($M/year)	($M/year)	$/t ore	($M/year)	$/t ore
Labour	100%	$33.5	$0.0	$0.00	$33.5	$0.84
Operating consumables	15%	$13.8	$79.9	$2.00	$93.6	$2.34
Power	33%	$27.0	$56.1	$1.40	$83.1	$2.08
Maintenance materials and contractors	81%	$33.9	$8.2	$0.21	$42.1	$1.05
subtotal	**43%**	**$108.3**	**$144.2**	**$3.60**	**$252.4**	**$6.31**
Adjustments		$14.6	$5.8	$0.17	$20.4	$0.56
subtotal		**$122.9**	**$149.9**	**$3.78**	**$272.8**	**$6.87**
Plus indirects		$15.0			$15.0	$0.38
total		**$137.9**	**$149.9**	**$3.78**	**$287.8**	**$7.25**

Table 22-6 Unit G&A costs for cashflow modelling, before and after adjustments

G&A cost centres (40 Mtpa)	Fixed cost	Total cost	
	($M/year)	($M/year)	$/t ore
Administration	$18.5	$18.5	$0.46
Labour	$27.2	$27.2	$0.68
Maintenance	$10.8	$10.8	$0.27
Energy	$1.1	$1.1	$0.03
subtotal	**$57.6**	**$57.6**	**$1.44**
Adjustments			
subtotal	**$0.0**	**$0.0**	**$0.00**
Plus indirects	$2.6	$2.6	$0.07
total	**$60.2**	**$60.2**	**$1.52**

The above listed unit costs, before adjustments and indirects, are consistent with those estimated and itemised in 20.7 21. The adjustments and indirects are costs identified by the FQM financing team and included into the Project cashflow model.

60 Mtpa operation

The preliminary, estimated overall average unit operating costs in the cashflow model for the 60 Mtpa Stage 2 are:

- operations mining of ore and waste (excluding stockpile reclaim):
  - $1.85/t average

- processing:
  - $6.76/t average
- G&A:
  - $1.40/t processed
- C1 costs and AISC:
  - $1.15/lb Cu and $1.54/lb Cu for the first 20 years of operation
  - $1.26/lb Cu and $1.60/lb Cu for the life of mine

22.2.5 Metal costs

The overall average metal costs (including treatment charges, refining and freight charges) captured in the 40 Mtpa Stage 1 cashflow model are:

- copper = $0.48/lb (range: $0.45/lb to $0.51/lb)
- molybdenum = $0.20/lb
- gold = $5.10/oz

The preliminary, estimated overall average metal costs in the cashflow model for the 60 Mtpa Stage 2 are the same as listed above.

22.2.6 Capital depreciation

The Company's local Argentine finance team and taxation team provided guidance on the modelling of taxation, including taxation depreciation schedules accounting for Project development and expansion capital, and for sustaining capital on fixed assets. These have been the basis for the calculation of taxable income and hence the annual tax to be paid at the 25% corporate rate.

22.3 Cashflow model outcomes

The Stage 1 40 Mtpa cashflow model for the economic analysis supporting the Mineral Reserve estimate is listed in Table 22-7 to Table 22-10.

On a pre-tax basis:

- the undiscounted cashflow is $38,248.6 M
- the NPV is $7,362.4 M at an 8% discount rate
- the NPV is $5,138.4 M at a 10% discount rate
- the internal rate of return is 23.3%
- the Project is cashflow positive from Year 6 and payback on the initial development capital is in Year 8 (i.e., four years after Project commissioning)

On a post-tax basis:

- the estimated taxable income is estimated as $35,982.2 M
- the total tax payable on this amount is $10,739.6 M, yielding an undiscounted post-tax cashflow of $27,509.0 M
- the NPV is $4,691.4 M at an 8% discount rate
- the NPV is $3,065.5 M at a 10% discount rate
- the internal rate of return is 18.4%

- the Project remains cashflow positive from Year 6 and the payback year on the initial development capital remains as Year 9

The comparative Stage 2 60 Mtpa cashflow model is listed in Table 22-11 and Table 22-12.

On a pre-tax basis:

- the undiscounted cashflow is $39,479.7 M
- the NPV is $9,087.3 M at an 8% discount rate
- the NPV is $6433.2 M at a 10% discount rate
- the internal rate of return is 23.3%
- the Project remains cashflow positive throughput the expansion capital spend and the payback year on the incremental positive cashflows is from the fourth year (Year 12) following the expansion

On a post-tax basis:

- the estimated taxable income is estimated as $37,129.3 M
- the total tax payable on this amount is $11,072.2 M, yielding an undiscounted post-tax cashflow of $28,407.5 M
- the NPV is $5,917.1 M at an 8% discount rate
- the NPV is $3,973.3 M at a 10% discount rate
- the internal rate of return is 19.3%
- the Project remains cashflow positive throughput the expansion capital spend and the payback year on the incremental positive cashflows is from the fourth year (Year 12) following the expansion

Table 22-7 Stage 1 40 Mtpa cashflow model physicals summary (Year 1 to Year 27)

P H Y S I C A L S	UNITS	TOTAL	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27
MINING (AFTER MINING DILUTION & RECOVERY)																													
Ore mined direct to plant	Mt	**1,268.1**	0.0	0.0	0.0	0.0	17.0	36.5	31.8	33.1	37.8	38.9	40.1	40.0	36.2	36.3	38.2	33.6	39.9	34.7	33.4	31.7	31.7	31.7	31.8	32.1	35.4	31.7	31.8
Ore mined to stockpile	Mt	**722.0**	0.0	0.0	0.5	9.9	35.3	14.9	13.7	9.3	14.9	7.8	16.1	29.9	21.6	18.7	19.6	29.8	34.1	39.1	23.9	20.2	16.7	11.9	23.7	24.5	25.5	28.6	30.8
Ore reclaimed from stockpile	Mt	**722.0**	0.0	0.0	0.0	0.0	7.9	3.5	8.3	6.9	2.2	1.1	0.0	0.0	3.8	3.7	1.9	6.4	0.1	5.3	6.7	8.3	8.3	8.3	8.3	7.9	4.6	8.3	8.3
Waste mined to dump	Mt	**2,903.4**	19.3	94.3	146.7	161.7	143.0	143.8	150.3	152.8	142.5	148.5	139.5	125.4	137.4	140.2	137.9	116.8	82.0	58.0	59.8	64.9	68.3	73.0	43.3	41.9	37.6	38.2	36.2
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)																													
Total direct feed	Mt	**1,268.1**	0.0	0.0	0.0	0.0	17.0	36.5	31.8	33.1	37.8	38.9	40.1	40.0	36.2	36.3	38.2	33.6	39.9	34.7	33.4	31.7	31.7	31.7	31.8	32.1	35.4	31.7	31.8
	% Cu	**0.56**	0.00	0.00	0.00	0.00	0.66	0.77	0.95	0.84	0.83	0.77	0.71	0.60	0.53	0.49	0.51	0.58	0.59	0.59	0.57	0.56	0.54	0.46	0.54	0.50	0.48	0.54	0.46
	ppm Mo	**136.89**	0.00	0.00	0.00	0.00	86.29	97.09	133.79	130.99	184.62	203.20	147.23	127.53	102.50	103.47	134.48	132.39	136.42	120.06	124.85	108.35	104.15	140.73	150.76	131.59	109.28	130.55	153.55
	g/t Au	**0.10**	0.00	0.00	0.00	0.00	0.10	0.12	0.17	0.13	0.15	0.17	0.13	0.12	0.10	0.10	0.11	0.12	0.13	0.12	0.14	0.11	0.10	0.11	0.11	0.09	0.08	0.09	0.09
Total reclaim feed	Mt	**722.0**	0.0	0.0	0.0	0.0	7.9	3.5	8.3	6.9	2.2	1.1	0.0	0.0	3.8	3.7	1.9	6.4	0.1	5.3	6.7	8.3	8.3	8.3	8.3	7.9	4.6	8.3	8.3
	% Cu	**0.19**	0.00	0.00	0.00	0.00	0.50	0.31	0.14	0.52	0.21	1.57	0.00	0.00	0.21	0.21	0.21	0.21	0.77	0.62	0.67	0.29	0.39	0.22	0.21	0.29	0.21	0.21	0.29
	ppm Mo	**98.05**	0.00	0.00	0.00	0.00	39.93	41.99	21.59	64.01	77.23	335.17	0.00	0.00	100.69	98.62	101.14	105.98	147.54	192.59	169.32	118.37	131.58	100.85	103.84	139.28	119.23	112.23	127.51
	g/t Au	**0.06**	0.00	0.00	0.00	0.00	0.08	0.07	0.04	0.11	0.08	0.31	0.00	0.00	0.10	0.09	0.09	0.10	0.09	0.17	0.17	0.11	0.11	0.07	0.09	0.09	0.08	0.09	0.08
Total plant feed	**Mt**	**1,990.1**	**0.0**	**0.0**	**0.0**	**0.0**	**24.9**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**
	% Cu	**0.42**	**0.00**	**0.00**	**0.00**	**0.00**	**0.61**	**0.73**	**0.78**	**0.78**	**0.80**	**0.80**	**0.71**	**0.60**	**0.50**	**0.46**	**0.49**	**0.52**	**0.59**	**0.59**	**0.59**	**0.50**	**0.50**	**0.41**	**0.47**	**0.46**	**0.45**	**0.47**	**0.42**
	ppm Mo	**122.80**	**0.00**	**0.00**	**0.00**	**0.00**	**71.60**	**92.31**	**110.56**	**119.50**	**178.74**	**206.76**	**147.23**	**127.53**	**102.33**	**103.02**	**132.91**	**128.16**	**136.44**	**129.65**	**132.26**	**110.42**	**109.82**	**132.48**	**141.05**	**133.11**	**110.43**	**126.75**	**148.16**
	g/t Au	**0.09**	**0.00**	**0.00**	**0.00**	**0.00**	**0.09**	**0.12**	**0.14**	**0.13**	**0.15**	**0.18**	**0.13**	**0.12**	**0.10**	**0.10**	**0.11**	**0.12**	**0.13**	**0.12**	**0.14**	**0.11**	**0.10**	**0.10**	**0.11**	**0.09**	**0.08**	**0.09**	**0.08**
Cu insitu	**kt**	**8,426.4**	**0.0**	**0.0**	**0.0**	**0.0**	**151.7**	**292.0**	**312.9**	**312.0**	**320.0**	**320.0**	**284.8**	**240.0**	**200.0**	**184.0**	**196.5**	**208.0**	**236.0**	**236.0**	**236.6**	**200.0**	**200.0**	**164.0**	**188.5**	**184.0**	**180.0**	**188.0**	**168.5**
Mo insitu	**kt**	**244.4**	**0.0**	**0.0**	**0.0**	**0.0**	**1.8**	**3.7**	**4.4**	**4.8**	**7.1**	**8.3**	**5.9**	**5.1**	**4.1**	**4.1**	**5.3**	**5.1**	**5.5**	**5.2**	**5.3**	**4.4**	**4.4**	**5.3**	**5.7**	**5.3**	**4.4**	**5.1**	**5.9**
Au insitu	**k(t)oz**	**5,480.2**	**0.0**	**0.0**	**0.0**	**0.0**	**75.2**	**147.9**	**181.8**	**163.3**	**187.8**	**226.3**	**166.4**	**158.2**	**128.6**	**131.2**	**144.4**	**150.5**	**168.5**	**158.2**	**183.1**	**141.5**	**128.6**	**127.3**	**138.0**	**110.6**	**102.9**	**111.9**	**108.3**
AVERAGE RECOVERIES																													
Copper recovery	%	**87.1%**	0.0%	0.0%	0.0%	0.0%	87.2%	86.6%	86.8%	87.1%	88.5%	88.8%	87.2%	86.7%	85.8%	86.0%	87.0%	87.3%	87.8%	87.9%	87.6%	86.8%	86.5%	86.7%	86.7%	85.9%	85.5%	86.3%	87.0%
Molybdenum recovery	%	**44.3%**	0.0%	0.0%	0.0%	0.0%	40.0%	40.0%	40.3%	40.8%	42.6%	44.2%	45.1%	45.8%	47.8%	49.2%	43.9%	46.1%	44.4%	47.1%	46.2%	48.1%	48.7%	47.7%	45.4%	43.5%	41.8%	42.8%	44.0%
Gold recovery	%	**61.5%**	0.0%	0.0%	0.0%	0.0%	60.3%	59.9%	60.3%	60.0%	60.6%	60.9%	61.4%	61.2%	61.6%	62.1%	60.9%	61.3%	61.3%	61.8%	61.4%	61.7%	62.1%	61.9%	61.4%	60.7%	60.5%	60.7%	61.1%
Ramp-up factor	*%*						*85.0%*	*95.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*
Adjusted copper recovery	%	**86.8%**	0.0%	0.0%	0.0%	0.0%	74.1%	82.3%	86.8%	87.1%	88.5%	88.8%	87.2%	86.7%	85.8%	86.0%	87.0%	87.3%	87.8%	87.9%	87.6%	86.8%	86.5%	86.7%	86.7%	85.9%	85.5%	86.3%	87.0%
Adjusted molybdenum recovery	%	**44.3%**	0.0%	0.0%	0.0%	0.0%	0.0%	38.0%	40.3%	40.8%	42.6%	44.2%	45.1%	45.8%	47.8%	49.2%	43.9%	46.1%	44.4%	47.1%	46.2%	48.1%	48.7%	47.7%	45.4%	43.5%	41.8%	42.8%	44.0%
Adjusted gold recovery	%	**61.5%**	0.0%	0.0%	0.0%	0.0%	51.2%	56.9%	60.3%	60.0%	60.6%	60.9%	61.4%	61.2%	61.6%	62.1%	60.9%	61.3%	61.3%	61.8%	61.4%	61.7%	62.1%	61.9%	61.4%	60.7%	60.5%	60.7%	61.1%
METAL RECOVERED																													
unadjusted Cu recovered	**kt**	**7,342.8**	**0.0**	**0.0**	**0.0**	**0.0**	**132.3**	**252.8**	**271.5**	**271.6**	**283.1**	**284.0**	**248.3**	**208.0**	**171.6**	**158.3**	**171.0**	**181.7**	**207.1**	**207.4**	**207.3**	**173.6**	**173.1**	**142.2**	**163.4**	**158.1**	**153.9**	**162.2**	**146.6**
unadjusted Mo recovered	**kt**	**109.1**	**0.0**	**0.0**	**0.0**	**0.0**	**0.7**	**1.5**	**1.8**	**2.0**	**3.0**	**3.7**	**2.7**	**2.3**	**2.0**	**2.0**	**2.3**	**2.4**	**2.4**	**2.4**	**2.5**	**2.1**	**2.1**	**2.5**	**2.6**	**2.3**	**1.8**	**2.2**	**2.6**
unadjusted Au recovered	**k(t)oz**	**3,380.8**	**0.0**	**0.0**	**0.0**	**0.0**	**45.3**	**88.5**	**109.6**	**97.9**	**113.8**	**137.9**	**102.1**	**96.8**	**79.3**	**81.4**	**88.0**	**92.2**	**103.2**	**97.7**	**112.4**	**87.3**	**79.9**	**78.8**	**84.7**	**67.2**	**62.2**	**67.9**	**66.2**
adjusted Cu recovered	**kt**	**7,310.3**	**0.0**	**0.0**	**0.0**	**0.0**	**112.5**	**240.2**	**271.5**	**271.6**	**283.1**	**284.0**	**248.3**	**208.0**	**171.6**	**158.3**	**171.0**	**181.7**	**207.1**	**207.4**	**207.3**	**173.6**	**173.1**	**142.2**	**163.4**	**158.1**	**153.9**	**162.2**	**146.6**
adjusted Mo recovered	**kt**	**108.3**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**1.4**	**1.8**	**2.0**	**3.0**	**3.7**	**2.7**	**2.3**	**2.0**	**2.0**	**2.3**	**2.4**	**2.4**	**2.4**	**2.5**	**2.1**	**2.1**	**2.5**	**2.6**	**2.3**	**1.8**	**2.2**	**2.6**
adjusted Au recovered	**k(t)oz**	**3,369.6**	**0.0**	**0.0**	**0.0**	**0.0**	**38.5**	**84.1**	**109.6**	**97.9**	**113.8**	**137.9**	**102.1**	**96.8**	**79.3**	**81.4**	**88.0**	**92.2**	**103.2**	**97.7**	**112.4**	**87.3**	**79.9**	**78.8**	**84.7**	**67.2**	**62.2**	**67.9**	**66.2**
CONCENTRATE PRODUCED																													
Cu concentrate	kt(wet)	**31,605.6**	0.0	0.0	0.0	0.0	485.3	1,061.5	1,198.8	1,186.4	1,162.3	1,172.8	1,038.6	884.3	757.2	701.6	734.7	773.4	865.5	874.3	878.3	768.7	768.2	633.1	714.9	683.1	668.2	699.9	630.9
Cu concentrate grade	%	**25.7%**	0.0%	0.0%	0.0%	0.0%	25.8%	25.1%	25.2%	25.4%	27.1%	26.9%	26.6%	26.1%	25.2%	25.1%	25.9%	26.1%	26.6%	26.4%	26.2%	25.1%	25.0%	25.0%	25.4%	25.7%	25.6%	25.7%	25.8%
Mo concentrate	kt(wet)	**256.1**	0.0	0.0	0.0	0.0	0.0	3.3	4.2	4.6	7.2	8.6	6.3	5.5	4.6	4.8	5.5	5.6	5.7	5.8	5.8	5.0	5.1	6.0	6.1	5.5	4.4	5.1	6.2
Mo concentrate grade	%	**47.0%**	0.0%	0.0%	0.0%	0.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%

Table 22-8 Stage 1 40 Mtpa cashflow model physicals summary (Year 28 to Year 55)

P H Y S I C A L S	UNITS	Y28	Y29	Y30	Y31	Y32	Y33	Y34	Y35	Y36	Y37	Y38	Y39	Y40	Y41	Y42	Y43	Y44	Y45	Y46	Y47	Y48	Y49	Y50	Y51	Y52	Y53	Y54	Y55
MINING (AFTER MINING DILUTION & RECOVERY)																													
Ore mined direct to plant	Mt	31.7	37.4	40.0	36.0	35.9	33.3	34.3	35.3	40.0	40.0	40.0	24.0	25.4	29.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Ore mined to stockpile	Mt	27.5	31.0	27.0	20.0	13.7	14.1	11.5	11.7	10.9	13.1	12.7	3.8	2.3	1.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Ore reclaimed from stockpile	Mt	8.3	2.6	0.0	4.1	4.1	6.7	5.7	4.8	0.0	0.0	0.0	16.1	14.6	10.8	40.0	40.1	40.0	40.0	40.0	40.1	40.0	40.0	40.0	40.1	40.0	40.0	40.0	3.9
Waste mined to dump	Mt	39.3	30.1	31.5	19.4	10.6	12.7	14.4	13.3	9.2	7.1	5.5	2.8	2.5	1.5	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)																													
Total direct feed	Mt	31.7	37.4	40.0	36.0	35.9	33.3	34.3	35.3	40.0	40.0	40.0	24.0	25.4	29.2	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	% Cu	0.38	0.44	0.45	0.41	0.40	0.44	0.45	0.45	0.51	0.50	0.47	0.49	0.58	0.58	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	ppm Mo	119.71	105.45	108.85	136.06	149.58	131.10	136.57	149.41	172.74	175.16	161.40	172.23	164.26	163.60	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	g/t Au	0.07	0.07	0.06	0.08	0.08	0.08	0.07	0.08	0.08	0.08	0.09	0.07	0.08	0.07	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Total reclaim feed	Mt	8.3	2.6	0.0	4.1	4.1	6.7	5.7	4.8	0.0	0.0	0.0	16.1	14.6	10.8	40.0	40.1	40.0	40.0	40.0	40.1	40.0	40.0	40.0	40.1	40.0	40.0	40.0	3.9
	% Cu	0.21	0.21	0.00	0.21	0.21	0.21	0.21	0.21	0.00	0.00	0.00	0.27	0.21	0.21	0.21	0.21	0.21	0.21	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13
	ppm Mo	113.48	115.10	0.00	117.32	117.42	117.52	121.05	122.55	0.00	0.00	0.00	133.31	130.66	130.66	130.66	130.41	130.11	130.11	77.70	74.91	74.91	74.91	74.91	74.91	74.91	74.91	74.91	74.91
	g/t Au	0.09	0.09	0.00	0.09	0.09	0.09	0.09	0.08	0.00	0.00	0.00	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.03
Total plant feed	**Mt**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**40.1**	**40.0**	**40.0**	**40.0**	**3.9**
	% Cu	**0.35**	**0.43**	**0.45**	**0.39**	**0.38**	**0.40**	**0.41**	**0.42**	**0.51**	**0.50**	**0.47**	**0.40**	**0.44**	**0.48**	**0.21**	**0.21**	**0.21**	**0.21**	**0.13**	**0.13**	**0.13**	**0.13**	**0.13**	**0.13**	**0.13**	**0.13**	**0.13**	**0.13**
	ppm Mo	**118.42**	**106.09**	**108.85**	**134.14**	**146.26**	**128.82**	**134.36**	**146.18**	**172.74**	**175.16**	**161.40**	**156.62**	**151.99**	**154.74**	**130.66**	**130.41**	**130.11**	**130.11**	**77.70**	**74.91**	**74.91**	**74.91**	**74.91**	**74.91**	**74.91**	**74.91**	**74.91**	**74.91**
	g/t Au	**0.08**	**0.07**	**0.06**	**0.08**	**0.08**	**0.08**	**0.07**	**0.08**	**0.08**	**0.08**	**0.09**	**0.08**	**0.08**	**0.08**	**0.08**	**0.08**	**0.08**	**0.08**	**0.03**	**0.03**	**0.03**	**0.03**	**0.03**	**0.03**	**0.03**	**0.03**	**0.03**	**0.03**
Cu insitu	**kt**	**140.0**	**172.0**	**180.0**	**156.4**	**152.0**	**160.0**	**164.0**	**168.5**	**204.0**	**200.0**	**188.0**	**160.4**	**176.0**	**192.0**	**84.0**	**84.2**	**84.0**	**84.0**	**52.0**	**52.1**	**52.0**	**52.0**	**52.0**	**52.1**	**52.0**	**52.0**	**52.0**	**5.1**
Mo insitu	**kt**	**4.7**	**4.2**	**4.4**	**5.4**	**5.9**	**5.2**	**5.4**	**5.9**	**6.9**	**7.0**	**6.5**	**6.3**	**6.1**	**6.2**	**5.2**	**5.2**	**5.2**	**5.2**	**3.1**	**3.0**	**3.0**	**3.0**	**3.0**	**3.0**	**3.0**	**3.0**	**3.0**	**0.3**
Au insitu	**k(t)oz**	**99.0**	**88.7**	**78.4**	**104.5**	**102.9**	**101.6**	**95.2**	**99.3**	**100.3**	**108.0**	**113.2**	**96.7**	**102.9**	**96.5**	**102.9**	**100.6**	**97.7**	**97.7**	**41.2**	**38.7**	**38.6**	**38.6**	**38.6**	**38.7**	**38.6**	**38.6**	**38.6**	**3.8**
AVERAGE RECOVERIES																													
Copper recovery	%	86.6%	87.6%	88.1%	88.1%	88.2%	88.0%	88.1%	88.3%	88.8%	88.6%	88.0%	87.7%	87.7%	88.1%	85.5%	85.7%	85.8%	85.8%	84.9%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%
Molybdenum recovery	%	45.3%	49.0%	50.7%	48.4%	47.5%	47.3%	45.6%	45.5%	43.6%	42.0%	42.0%	43.4%	43.1%	43.2%	43.3%	44.0%	44.8%	44.8%	43.6%	43.5%	43.5%	43.5%	43.5%	43.5%	43.5%	43.5%	43.5%	43.5%
Gold recovery	%	61.5%	62.9%	63.2%	62.5%	62.8%	62.3%	62.0%	61.7%	61.0%	60.7%	60.5%	61.2%	61.0%	61.3%	60.5%	60.9%	61.0%	61.0%	68.1%	68.2%	68.2%	68.2%	68.2%	68.2%	68.2%	68.2%	68.2%	68.2%
Ramp-up factor	*%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*
Adjusted copper recovery	%	86.6%	87.6%	88.1%	88.1%	88.2%	88.0%	88.1%	88.3%	88.8%	88.6%	88.0%	87.7%	87.7%	88.1%	85.5%	85.7%	85.8%	85.8%	84.9%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%
Adjusted molybdenum recovery	%	45.3%	49.0%	50.7%	48.4%	47.5%	47.3%	45.6%	45.5%	43.6%	42.0%	42.0%	43.4%	43.1%	43.2%	43.3%	44.0%	44.8%	44.8%	43.6%	43.5%	43.5%	43.5%	43.5%	43.5%	43.5%	43.5%	43.5%	43.5%
Adjusted gold recovery	%	61.5%	62.9%	63.2%	62.5%	62.8%	62.3%	62.0%	61.7%	61.0%	60.7%	60.5%	61.2%	61.0%	61.3%	60.5%	60.9%	61.0%	61.0%	68.1%	68.2%	68.2%	68.2%	68.2%	68.2%	68.2%	68.2%	68.2%	68.2%
METAL RECOVERED																													
unadjusted Cu recovered	**kt**	**121.2**	**150.6**	**158.6**	**137.7**	**134.0**	**140.8**	**144.5**	**148.7**	**181.2**	**177.1**	**165.4**	**140.6**	**154.3**	**169.2**	**71.8**	**72.2**	**72.1**	**72.1**	**44.1**	**44.2**	**44.1**	**44.1**	**44.1**	**44.2**	**44.1**	**44.1**	**44.1**	**4.3**
unadjusted Mo recovered	**kt**	**2.1**	**2.1**	**2.2**	**2.6**	**2.8**	**2.4**	**2.4**	**2.7**	**3.0**	**2.9**	**2.7**	**2.7**	**2.6**	**2.7**	**2.3**	**2.3**	**2.3**	**2.3**	**1.4**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**0.1**
unadjusted Au recovered	**k(t)oz**	**60.9**	**55.8**	**49.6**	**65.3**	**64.6**	**63.3**	**59.0**	**61.2**	**61.1**	**65.6**	**68.5**	**59.2**	**62.7**	**59.1**	**62.3**	**61.3**	**59.7**	**59.7**	**28.0**	**26.4**	**26.3**	**26.3**	**26.3**	**26.4**	**26.3**	**26.3**	**26.3**	**2.6**
adjusted Cu recovered	**kt**	**121.2**	**150.6**	**158.6**	**137.7**	**134.0**	**140.8**	**144.5**	**148.7**	**181.2**	**177.1**	**165.4**	**140.6**	**154.3**	**169.2**	**71.8**	**72.2**	**72.1**	**72.1**	**44.1**	**44.2**	**44.1**	**44.1**	**44.1**	**44.2**	**44.1**	**44.1**	**44.1**	**4.3**
adjusted Mo recovered	**kt**	**2.1**	**2.1**	**2.2**	**2.6**	**2.8**	**2.4**	**2.4**	**2.7**	**3.0**	**2.9**	**2.7**	**2.7**	**2.6**	**2.7**	**2.3**	**2.3**	**2.3**	**2.3**	**1.4**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**0.1**
adjusted Au recovered	**k(t)oz**	**60.9**	**55.8**	**49.6**	**65.3**	**64.6**	**63.3**	**59.0**	**61.2**	**61.1**	**65.6**	**68.5**	**59.2**	**62.7**	**59.1**	**62.3**	**61.3**	**59.7**	**59.7**	**28.0**	**26.4**	**26.3**	**26.3**	**26.3**	**26.4**	**26.3**	**26.3**	**26.3**	**2.6**
CONCENTRATE PRODUCED																													
Cu concentrate	kt(wet)	536.6	664.5	701.7	603.0	585.9	613.4	615.7	627.7	734.3	712.9	687.2	595.2	652.0	698.9	335.3	336.4	335.8	335.8	207.8	208.3	207.8	207.8	207.8	208.3	207.8	207.8	207.8	20.4
Cu concentrate grade	%	25.1%	25.2%	25.1%	25.4%	25.4%	25.5%	26.1%	26.3%	27.4%	27.6%	26.8%	26.3%	26.3%	26.9%	23.8%	23.8%	23.9%	23.9%	23.6%	23.6%	23.6%	23.6%	23.6%	23.6%	23.6%	23.6%	23.6%	23.6%
Mo concentrate	kt(wet)	5.1	4.9	5.2	6.2	6.6	5.8	5.8	6.3	7.1	7.0	6.4	6.5	6.2	6.3	5.4	5.4	5.5	5.5	3.2	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	0.3
Mo concentrate grade	%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%

Table 22-9 Stage 1 40 Mtpa cashflow model summary (Year 1 to 27)

CASHFLOWS	UNITS	TOTAL	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27
PAYABILITY																													
Cu	%	**96.1%**	0.0%	0.0%	0.0%	0.0%	96.1%	96.0%	96.0%	96.1%	96.3%	96.3%	96.2%	96.2%	96.0%	96.0%	96.1%	96.2%	96.2%	96.2%	96.2%	96.0%	96.0%	96.0%	96.1%	96.1%	96.1%	96.1%	96.1%
Mo	%	**86.0%**	0.0%	0.0%	0.0%	0.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%
Au	%	**90.0%**	0.0%	0.0%	0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
Payable metal recovered																													
Cu	kt	**7,025.9**	**0.0**	**0.0**	**0.0**	**0.0**	**108.1**	**230.6**	**260.7**	**261.0**	**272.6**	**273.5**	**238.9**	**200.1**	**164.8**	**152.0**	**164.4**	**174.7**	**199.3**	**199.6**	**199.4**	**166.7**	**166.1**	**136.5**	**157.0**	**151.9**	**147.9**	**155.9**	**140.9**
Mo	kt	**93.2**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**1.2**	**1.5**	**1.7**	**2.6**	**3.1**	**2.3**	**2.0**	**1.7**	**1.7**	**2.0**	**2.0**	**2.1**	**2.1**	**2.1**	**1.8**	**1.8**	**2.2**	**2.2**	**2.0**	**1.6**	**1.9**	**2.2**
Au	k(t)oz	**3,032.6**	**0.0**	**0.0**	**0.0**	**0.0**	**34.7**	**75.7**	**98.7**	**88.2**	**102.5**	**124.1**	**91.9**	**87.1**	**71.3**	**73.3**	**79.2**	**83.0**	**92.9**	**88.0**	**101.2**	**78.5**	**71.9**	**71.0**	**76.2**	**60.5**	**56.0**	**61.1**	**59.6**
GROSS REVENUE																													
Metal prices																													
Cu	$/lb	**$4.50**	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50
Mo	$/lb	**$18.00**	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00
Au	$/oz	**$3,000**	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000
Revenue after payability																													
Copper revenue	**$M**	**$69,702.2**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$1,072.4**	**$2,287.9**	**$2,586.1**	**$2,588.8**	**$2,704.5**	**$2,713.1**	**$2,370.3**	**$1,984.9**	**$1,634.8**	**$1,507.8**	**$1,630.7**	**$1,733.2**	**$1,977.5**	**$1,979.7**	**$1,978.7**	**$1,654.0**	**$1,648.3**	**$1,354.4**	**$1,557.5**	**$1,507.1**	**$1,467.1**	**$1,546.2**	**$1,398.2**
Molybdenum revenue	**$M**	**$3,697.5**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$47.9**	**$61.0**	**$66.6**	**$103.9**	**$124.6**	**$90.8**	**$79.8**	**$66.8**	**$69.2**	**$79.8**	**$80.6**	**$82.8**	**$83.3**	**$83.6**	**$72.6**	**$73.0**	**$86.2**	**$87.6**	**$79.0**	**$63.0**	**$74.1**	**$89.1**
Gold revenue	**$M**	**$9,097.8**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$104.0**	**$227.1**	**$296.0**	**$264.5**	**$307.4**	**$372.3**	**$275.7**	**$261.4**	**$214.0**	**$219.9**	**$237.5**	**$249.0**	**$278.7**	**$263.9**	**$303.6**	**$235.6**	**$215.7**	**$212.9**	**$228.7**	**$181.4**	**$168.0**	**$183.4**	**$178.8**
Total revenue	**$M**	**$82,497.6**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$1,176.4**	**$2,562.9**	**$2,943.0**	**$2,919.9**	**$3,115.8**	**$3,210.1**	**$2,736.8**	**$2,326.0**	**$1,915.6**	**$1,796.9**	**$1,948.1**	**$2,062.9**	**$2,338.9**	**$2,326.9**	**$2,365.8**	**$1,962.2**	**$1,937.0**	**$1,653.5**	**$1,873.8**	**$1,767.4**	**$1,698.1**	**$1,803.6**	**$1,666.1**
METAL COSTS																													
TCRCs & freight	$M	**$8,020.5**	$0.0	$0.0	$0.0	$0.0	$127.9	$249.1	$278.2	$276.2	$274.9	$277.2	$247.4	$213.9	$185.5	$173.8	$181.9	$190.4	$210.6	$212.2	$212.9	$188.0	$187.8	$159.5	$177.2	$170.7	$167.2	$174.2	$160.0
Royalties	$M	**$1,117.2**	$0.0	$0.0	$0.0	$0.0	$15.7	$34.7	$40.0	$39.7	$42.6	$44.0	$37.3	$31.7	$26.0	$24.3	$26.5	$28.1	$31.9	$31.7	$32.3	$26.6	$26.2	$22.4	$25.4	$24.0	$23.0	$24.4	$22.6
Total metal costs	**$M**	**$9,137.6**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$143.6**	**$283.8**	**$318.2**	**$315.8**	**$317.5**	**$321.2**	**$284.8**	**$245.6**	**$211.5**	**$198.1**	**$208.4**	**$218.5**	**$242.6**	**$243.9**	**$245.2**	**$214.6**	**$214.0**	**$181.9**	**$202.7**	**$194.7**	**$190.2**	**$198.7**	**$182.6**
CAPITAL COSTS																													
Mining	$M	**$1,756.0**	$277.8	$498.0	$453.8	$428.7	$97.6	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Processing	$M	**$1,077.5**	$145.2	$191.3	$325.8	$297.9	$117.3	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Rail	$M	**$12.8**	$1.7	$2.3	$3.9	$3.5	$1.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Infrastructure	$M	**$727.0**	$98.0	$129.1	$219.8	$201.0	$79.1	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Indirects	$M	**$658.5**	$88.8	$116.9	$199.1	$182.0	$71.7	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal development capital	$M	**$4,231.7**	**$611.6**	**$937.5**	**$1,202.4**	**$1,113.1**	**$367.1**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Deferred stripping	$M	**$1,333.1**	$0.0	$0.0	$0.0	$0.0	$113.9	$123.6	$148.3	$159.3	$122.9	$150.9	$107.4	$45.5	$96.3	$106.3	$95.2	$45.4	$0.0	$0.0	$0.0	$0.0	$0.0	$18.3	$0.0	$0.0	$0.0	$0.0	$0.0
Closure costs	$M	**$71.7**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Total capital costs	**$M**	**$5,636.5**	**$611.6**	**$937.5**	**$1,202.4**	**$1,113.1**	**$481.0**	**$123.6**	**$148.3**	**$159.3**	**$122.9**	**$150.9**	**$107.4**	**$45.5**	**$96.3**	**$106.3**	**$95.2**	**$45.4**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$18.3**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
SUSTAINING COSTS																													
Mining	$M	**$2,404.6**	$0.0	$0.0	$0.0	$0.0	$38.0	$66.8	$95.3	$83.0	$141.6	$108.8	$63.9	$102.5	$106.9	$84.1	$115.3	$103.4	$52.8	$46.7	$40.0	$40.7	$114.4	$32.9	$77.6	$86.0	$38.6	$60.2	$67.3
Processing and infrastructure	$M	**$649.1**	$0.0	$0.0	$0.0	$0.0	$10.6	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0
Total sustaining costs	**$M**	**$3,053.7**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$48.6**	**$79.8**	**$108.3**	**$96.0**	**$154.5**	**$121.8**	**$77.0**	**$115.5**	**$119.9**	**$97.1**	**$128.3**	**$116.4**	**$65.8**	**$59.7**	**$53.0**	**$53.7**	**$127.3**	**$45.9**	**$90.6**	**$99.0**	**$51.6**	**$73.2**	**$80.4**
OPERATING COSTS																													
Mining	$M	**$8,716.7**	$0.0	$0.0	$0.9	$43.2	$230.3	$232.6	$209.8	$192.7	$245.9	$217.6	$258.2	$333.1	$264.1	$245.2	$255.3	$299.4	$302.0	$270.4	$240.7	$246.5	$245.1	$225.2	$225.5	$230.9	$225.3	$233.2	$243.2
Processing	$M	**$14,425.2**	$0.0	$0.0	$0.0	$0.0	$235.0	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2
General and administration	$M	**$3,015.6**	$0.0	$0.0	$0.0	$0.0	$47.6	$67.0	$68.9	$68.7	$69.6	$70.1	$68.0	$66.0	$64.1	$63.6	$64.4	$64.8	$66.1	$66.0	$66.3	$64.3	$64.2	$62.9	$64.0	$63.4	$63.1	$63.6	$63.1
Other	$M	**$101.3**	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8
Total operating costs	**$M**	**$26,258.7**	**$1.8**	**$1.8**	**$2.8**	**$45.1**	**$514.8**	**$590.4**	**$569.7**	**$551.8**	**$606.1**	**$578.5**	**$617.3**	**$689.6**	**$618.7**	**$599.5**	**$610.7**	**$654.7**	**$658.6**	**$627.1**	**$598.1**	**$601.4**	**$599.8**	**$578.8**	**$580.6**	**$584.8**	**$578.9**	**$587.5**	**$597.3**
OTHER COSTS																													
Taxes on bank transactions	$M	**$149.3**	$3.7	$5.6	$7.2	$7.0	$3.9	$3.2	$3.3	$3.3	$3.5	$3.4	$3.3	$3.4	$3.4	$3.2	$3.3	$3.3	$3.0	$2.8	$2.7	$2.7	$2.9	$2.6	$2.7	$2.7	$2.6	$2.7	$2.7
MINERAL RESERVE CASHFLOW (PRE-TAX)																													
Net revenue	$M	**$74,477.1**	$0.0	$0.0	$0.0	$0.0	$1,048.5	$2,313.9	$2,664.8	$2,643.7	$2,840.9	$2,932.9	$2,489.4	$2,112.1	$1,730.0	$1,623.1	$1,766.2	$1,872.5	$2,128.3	$2,114.8	$2,152.9	$1,774.2	$1,749.3	$1,494.0	$1,696.6	$1,596.7	$1,530.9	$1,629.4	$1,506.2
Opex	$M	**$26,258.7**	$1.8	$1.8	$2.8	$45.1	$514.8	$590.4	$569.7	$551.8	$606.1	$578.5	$617.3	$689.6	$618.7	$599.5	$610.7	$654.7	$658.6	$627.1	$598.1	$601.4	$599.8	$578.8	$580.6	$584.8	$578.9	$587.5	$597.3
Royalties and other	$M	**$1,266.4**	$3.7	$5.6	$7.2	$7.0	$19.6	$37.9	$43.3	$42.9	$46.1	$47.4	$40.6	$35.1	$29.3	$27.6	$29.8	$31.4	$34.9	$34.5	$35.0	$29.3	$29.1	$25.0	$28.2	$26.7	$25.5	$27.1	$25.3
Capex	$M	**$5,636.5**	$611.6	$937.5	$1,202.4	$1,113.1	$481.0	$123.6	$148.3	$159.3	$122.9	$150.9	$107.4	$45.5	$96.3	$106.3	$95.2	$45.4	$0.0	$0.0	$0.0	$0.0	$0.0	$18.3	$0.0	$0.0	$0.0	$0.0	$0.0
Sustaining	$M	**$3,053.7**	$0.0	$0.0	$0.0	$0.0	$48.6	$79.8	$108.3	$96.0	$154.5	$121.8	$77.0	$115.5	$119.9	$97.1	$128.3	$116.4	$65.8	$59.7	$53.0	$53.7	$127.3	$45.9	$90.6	$99.0	$51.6	$73.2	$80.4
Working capex	$M	**$13.1**	$0.5	$0.0	$0.2	$11.0	$203.7	$122.5	$22.0	-$6.3	$29.7	$0.5	-$23.9	-$11.5	-$51.4	-$13.4	$14.1	$21.9	$24.7	-$9.1	-$4.2	-$30.3	-$4.5	-$24.7	$16.4	-$7.3	-$5.6	$9.7	-$7.8
Undiscounted cashflow	**$M**	**38,248.6**	**-617.6**	**-945.0**	**-1,212.7**	**-1,176.2**	**-219.1**	**1,359.6**	**1,773.3**	**1,800.0**	**1,881.6**	**2,033.8**	**1,671.1**	**1,238.0**	**917.2**	**806.0**	**888.1**	**1,002.8**	**1,344.3**	**1,402.5**	**1,471.1**	**1,120.1**	**997.5**	**850.5**	**980.9**	**893.6**	**880.5**	**931.8**	**811.0**
NPV10 (indicative)	**$M**	**$5,138.4**																											
NPV8 (indicative)	**$M**	**$7,362.4**																											
IRR	**%**	**23.3%**																											
MINERAL RESERVE CASHFLOW (POST-TAX)																													
Taxable income	**$M**	**$35,982.2**	$0.0	$0.0	$0.0	$0.0	$0.0	-$109.8	$1,623.3	$1,688.7	$1,791.4	$1,887.1	$1,482.7	$1,082.2	$788.3	$721.8	$832.8	$872.9	$1,130.2	$1,182.6	$1,259.0	$922.9	$866.6	$668.5	$868.6	$747.5	$714.8	$812.5	$682.7
Tax paid	**$M**	**-$10,739.6**	$0.0	$0.0	$0.0	$0.0	-$23.7	-$32.2	-$475.9	-$491.6	-$523.3	-$549.9	-$435.6	-$324.3	-$239.3	-$219.4	-$251.5	-$264.8	-$336.8	-$349.4	-$369.6	-$274.3	-$259.4	-$202.2	-$258.3	-$225.0	-$214.9	-$242.3	-$206.2
Undiscounted cashflow	**$M**	**$27,509.0**	**-617.6**	**-945.0**	**-1,212.7**	**-1,176.2**	**-242.8**	**1,327.5**	**1,297.4**	**1,308.4**	**1,358.3**	**1,483.9**	**1,235.5**	**913.6**	**677.9**	**586.5**	**636.6**	**738.0**	**1,007.5**	**1,053.1**	**1,101.6**	**845.8**	**738.1**	**648.3**	**722.6**	**668.5**	**665.6**	**689.5**	**604.8**
NPV10 (indicative)	**$M**	**$3,065.5**																											
NPV8 (indicative)	**$M**	**$4,691.4**																											
IRR	**%**	**18.4%**																											

Table 22-10 Stage 1 40 Mtpa cashflow model summary (Year 28 to 55)

CASHFLOWS	UNITS	Y28	Y29	Y30	Y31	Y32	Y33	Y34	Y35	Y36	Y37	Y38	Y39	Y40	Y41	Y42	Y43	Y44	Y45	Y46	Y47	Y48	Y49	Y50	Y51	Y52	Y53	Y54	Y55
PAYABILITY																													
Cu	%	96.0%	96.0%	96.0%	96.1%	96.1%	96.1%	96.2%	96.2%	96.4%	96.4%	96.3%	96.2%	96.2%	96.3%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%
Mo	%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%
Au	%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
Payable metal recovered																													
Cu	kt	116.4	144.6	152.3	132.3	128.8	135.2	138.9	143.1	174.5	170.7	159.3	135.3	148.4	162.9	68.8	69.1	69.1	69.1	42.3	42.4	42.2	42.2	42.2	42.4	42.2	42.2	42.2	4.1
Mo	kt	1.8	1.8	1.9	2.2	2.4	2.1	2.1	2.3	2.6	2.5	2.3	2.3	2.3	2.3	1.9	2.0	2.0	2.0	1.2	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.1
Au	k(t)oz	54.8	50.2	44.6	58.7	58.1	57.0	53.1	55.1	55.0	59.0	61.6	53.3	56.5	53.2	56.1	55.1	53.7	53.7	25.2	23.8	23.7	23.7	23.7	23.8	23.7	23.7	23.7	2.3
GROSS REVENUE																													
Metal prices																													
Cu	$/lb	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50
Mo	$/lb	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00
Au	$/oz	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000
Revenue after payability																													
Copper revenue	$M	$1,154.7	$1,434.6	$1,511.1	$1,312.6	$1,277.4	$1,341.8	$1,378.1	$1,419.2	$1,731.6	$1,693.7	$1,579.9	$1,342.0	$1,472.2	$1,615.7	$682.6	$685.8	$685.4	$685.4	$419.3	$420.2	$419.1	$419.1	$419.1	$420.2	$419.1	$419.1	$419.1	$41.1
Molybdenum revenue	$M	$73.3	$70.9	$75.3	$88.8	$94.8	$83.2	$83.6	$91.1	$102.8	$100.4	$92.5	$93.1	$89.5	$91.2	$77.3	$78.5	$79.6	$79.6	$46.2	$44.6	$44.5	$44.5	$44.5	$44.6	$44.5	$44.5	$44.5	$4.4
Gold revenue	$M	$164.4	$150.7	$133.9	$176.2	$174.3	$171.0	$159.4	$165.3	$165.1	$177.1	$184.9	$159.8	$169.4	$159.7	$168.2	$165.4	$161.1	$161.1	$75.6	$71.3	$71.1	$71.1	$71.1	$71.3	$71.1	$71.1	$71.1	$7.0
Total revenue	$M	$1,392.5	$1,656.2	$1,720.3	$1,577.6	$1,546.5	$1,595.9	$1,621.0	$1,675.6	$1,999.5	$1,971.2	$1,857.4	$1,594.9	$1,731.1	$1,866.6	$928.0	$929.7	$926.0	$926.0	$541.2	$536.1	$534.6	$534.6	$534.6	$536.1	$534.6	$534.6	$534.6	$52.4
METAL COSTS																													
TCRCs & freight	$M	$139.2	$166.0	$173.8	$153.6	$150.1	$155.8	$156.9	$159.8	$183.9	$179.6	$173.0	$152.9	$164.9	$175.6	$96.3	$96.6	$96.5	$96.5	$69.2	$69.3	$69.2	$69.2	$69.2	$69.3	$69.2	$69.2	$69.2	$29.7
Royalties	$M	$18.8	$22.4	$23.2	$21.4	$20.9	$21.6	$22.0	$22.7	$27.2	$26.9	$25.3	$21.6	$23.5	$25.4	$12.5	$12.5	$12.4	$12.4	$7.1	$7.0	$7.0	$7.0	$7.0	$7.0	$7.0	$7.0	$7.0	$0.3
Total metal costs	$M	$158.0	$188.4	$197.0	$175.0	$171.1	$177.5	$178.9	$182.6	$211.1	$206.4	$198.2	$174.5	$188.4	$200.9	$108.8	$109.1	$108.9	$108.9	$76.3	$76.3	$76.2	$76.2	$76.2	$76.3	$76.2	$76.2	$76.2	$30.1
CAPITAL COSTS																													
Mining	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Processing	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Rail	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Infrastructure	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Indirects	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal development capital	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Deferred stripping	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Closure costs	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$71.7
Total capital costs	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$71.7
SUSTAINING COSTS																													
Mining	$M	$40.8	$91.6	$31.8	$27.9	$29.1	$19.8	$36.8	$25.0	$19.6	$38.6	$16.2	$11.5	$10.9	$23.8	$9.5	$9.4	$10.4	$36.7	$58.0	$6.9	$13.1	$8.4	$6.5	$22.1	$8.4	$6.1	$18.7	$0.0
Processing and infrastructure	$M	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$13.0	$1.6
Total sustaining costs	$M	$53.8	$104.6	$44.8	$41.0	$42.1	$32.8	$49.8	$38.1	$32.6	$51.5	$29.2	$24.5	$23.9	$36.8	$22.5	$22.4	$23.4	$49.7	$71.1	$20.0	$26.1	$21.4	$19.5	$35.1	$21.4	$19.1	$31.7	$1.6
OPERATING COSTS																													
Mining	$M	$236.2	$240.8	$244.8	$187.7	$166.0	$165.9	$169.1	$164.5	$157.7	$165.6	$162.8	$94.3	$96.8	$97.4	$52.9	$47.7	$50.3	$49.2	$50.8	$50.5	$54.6	$51.9	$52.9	$52.7	$51.0	$48.1	$38.7	$0.0
Processing	$M	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$289.2	$288.7	$288.7	$288.9	$35.0
General and administration	$M	$61.7	$62.9	$63.2	$62.7	$62.4	$62.6	$62.8	$63.1	$64.5	$64.4	$63.9	$62.7	$63.3	$63.9	$50.1	$50.2	$50.1	$50.1	$48.3	$48.4	$48.3	$48.3	$48.3	$48.4	$48.3	$48.3	$48.3	$6.4
Other	$M	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8
Total operating costs	$M	$588.4	$594.2	$598.7	$541.4	$518.9	$519.1	$522.6	$518.7	$512.7	$520.5	$517.4	$448.1	$450.6	$451.8	$393.8	$388.9	$390.9	$389.8	$389.9	$389.9	$393.5	$390.7	$392.0	$392.1	$389.8	$386.9	$377.7	$43.3
OTHER COSTS																													
Taxes on bank transactions	$M	$2.6	$2.8	$2.6	$2.4	$2.3	$2.3	$2.3	$2.3	$2.2	$2.3	$2.3	$1.9	$2.0	$2.0	$1.7	$1.7	$1.7	$1.8	$1.8	$1.7	$1.7	$1.7	$1.7	$1.7	$1.7	$1.7	$1.7	$0.4
MINERAL RESERVE CASHFLOW (PRE-TAX)																													
Net revenue	$M	$1,253.3	$1,490.1	$1,546.5	$1,424.0	$1,396.4	$1,440.1	$1,464.1	$1,515.7	$1,815.6	$1,791.7	$1,684.5	$1,442.0	$1,566.2	$1,691.0	$831.7	$833.1	$829.6	$829.6	$472.0	$466.8	$465.4	$465.4	$465.4	$466.8	$465.4	$465.4	$465.4	$22.6
Opex	$M	$588.4	$594.2	$598.7	$541.4	$518.9	$519.1	$522.6	$518.7	$512.7	$520.5	$517.4	$448.1	$450.6	$451.8	$393.8	$388.9	$390.9	$389.8	$389.9	$389.9	$393.5	$390.7	$392.0	$392.1	$389.8	$386.9	$377.7	$43.3
Royalties and other	$M	$21.4	$25.2	$25.8	$23.7	$23.2	$23.9	$24.3	$25.0	$29.5	$29.2	$27.5	$23.6	$25.4	$27.4	$14.2	$14.2	$14.1	$14.2	$8.9	$8.7	$8.7	$8.7	$8.7	$8.7	$8.7	$8.7	$8.7	$0.7
Capex	$M	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$71.7
Sustaining	$M	$53.8	$104.6	$44.8	$41.0	$42.1	$32.8	$49.8	$38.1	$32.6	$51.5	$29.2	$24.5	$23.9	$36.8	$22.5	$22.4	$23.4	$49.7	$71.1	$20.0	$26.1	$21.4	$19.5	$35.1	$21.4	$19.1	$31.7	$1.6
Working capex	$M	-$22.3	$19.6	$7.5	-$24.9	-$8.1	$3.9	$2.4	$3.5	$23.2	-$0.5	-$9.0	-$37.8	$10.9	$10.2	-$85.3	-$1.1	$0.2	-$1.0	-$30.0	$1.0	$0.6	-$0.6	$0.4	-$0.3	-$0.3	-$0.7	-$2.7	-$122.5
Undiscounted cashflow	$M	612.0	746.6	869.7	842.8	820.3	860.5	865.0	930.5	1,217.5	1,190.9	1,119.3	983.6	1,055.4	1,164.9	486.4	408.7	400.9	376.8	32.1	47.2	36.5	45.2	44.9	31.1	45.9	51.4	50.1	27.9
NPV10 (indicative)	$M																												
NPV8 (indicative)	$M																												
IRR	%																												
MINERAL RESERVE CASHFLOW (POST-TAX)																													
Taxable income	$M	$475.2	$671.3	$723.7	$709.6	$712.2	$757.5	$774.3	$826.0	$1,118.3	$1,087.2	$997.5	$861.1	$980.8	$1,093.7	$357.2	$372.9	$368.2	$356.9	-$3.5	$6.7	$25.4	$27.9	$30.6	$25.6	$26.2	$38.2	$43.9	$0.0
Tax paid	$M	-$146.7	-$202.9	-$217.6	-$210.3	-$210.1	-$222.7	-$227.6	-$242.1	-$324.1	-$315.7	-$290.0	-$248.4	-$282.1	-$314.1	-$104.3	-$108.2	-$107.0	-$104.2	-$3.4	-$5.8	-$10.4	-$11.1	-$11.7	-$10.5	-$10.6	-$13.6	-$15.0	$0.6
Undiscounted cashflow	$M	465.3	543.8	652.0	632.5	610.2	637.8	637.3	688.4	893.4	875.2	829.3	735.2	773.3	850.8	382.1	300.5	293.9	272.7	28.7	41.4	26.1	34.2	33.1	20.6	35.2	37.8	35.1	28.5
NPV10 (indicative)	$M																												
NPV8 (indicative)	$M																												
IRR	%																												

Table 22-11 Stage 2 60 Mtpa cashflow model physicals summary

PHYSICALS	UNITS	TOTAL	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
MINING (AFTER MINING DILUTION & RECOVERY)																					
Ore mined total	Mt	**1,990.1**	0.0	0.0	0.5	9.9	52.3	51.5	45.5	45.0	55.1	67.5	81.2	84.6	101.2	95.9	82.6	76.2	74.1	83.1	102.3
Waste mined to dump	Mt	**2,904.9**	19.3	94.3	146.7	161.7	143.0	171.3	179.8	180.9	170.2	157.8	144.8	140.7	124.1	129.5	143.3	149.1	151.2	142.2	75.2
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)																					
Total plant feed	**Mt**	**1,990.1**	**0.0**	**0.0**	**0.0**	**0.0**	**24.9**	**40.0**	**40.1**	**40.0**	**60.0**	**60.0**	**60.2**	**60.0**	**60.0**	**60.0**	**60.2**	**60.0**	**60.0**	**60.0**	**60.2**
	% Cu	**0.42**	**0.00**	**0.00**	**0.00**	**0.00**	**0.61**	**0.73**	**0.78**	**0.79**	**0.58**	**0.62**	**0.62**	**0.61**	**0.55**	**0.56**	**0.51**	**0.47**	**0.47**	**0.45**	**0.47**
	ppm Mo	**123.41**	**0.00**	**0.00**	**0.00**	**0.00**	**71.60**	**92.31**	**110.56**	**124.00**	**165.10**	**159.12**	**129.09**	**151.36**	**144.82**	**137.38**	**106.45**	**104.11**	**91.06**	**93.44**	**113.86**
	g/t Au	**0.09**	**0.00**	**0.00**	**0.00**	**0.00**	**0.09**	**0.12**	**0.14**	**0.13**	**0.14**	**0.15**	**0.11**	**0.13**	**0.13**	**0.12**	**0.10**	**0.11**	**0.10**	**0.09**	**0.09**
Cu insitu	**kt**	**8,450.0**	**0.0**	**0.0**	**0.0**	**0.0**	**151.7**	**292.0**	**312.9**	**316.0**	**348.0**	**372.0**	**373.0**	**366.0**	**330.0**	**336.0**	**306.8**	**282.0**	**282.0**	**270.0**	**282.8**
Mo insitu	**kt**	**245.59**	**0.0**	**0.0**	**0.0**	**0.0**	**1.8**	**3.7**	**4.4**	**5.0**	**9.9**	**9.5**	**7.8**	**9.1**	**8.7**	**8.2**	**6.4**	**6.2**	**5.5**	**5.6**	**6.9**
Au insitu	**k(t)oz**	**5,521.8**	**0.0**	**0.0**	**0.0**	**0.0**	**75.2**	**147.9**	**181.8**	**160.8**	**270.1**	**281.6**	**218.6**	**258.5**	**241.1**	**235.3**	**195.4**	**212.2**	**191.0**	**175.5**	**176.0**
AVERAGE RECOVERIES																					
Copper recovery	%	**87.1%**	0.0%	0.0%	0.0%	0.0%	87.2%	86.6%	86.8%	87.4%	87.5%	86.9%	85.7%	88.0%	87.7%	87.8%	86.5%	86.1%	85.4%	85.8%	85.9%
Molybdenum recovery	%	**44.3%**	0.0%	0.0%	0.0%	0.0%	40.0%	40.0%	40.3%	41.6%	43.5%	43.0%	42.8%	49.1%	47.0%	45.8%	43.3%	45.5%	47.1%	47.9%	43.9%
Gold recovery	%	**61.5%**	0.0%	0.0%	0.0%	0.0%	60.3%	59.9%	60.3%	60.4%	60.8%	60.6%	60.9%	62.2%	61.7%	61.5%	60.7%	61.0%	61.2%	61.5%	61.1%
Ramp-up factor	*%*						*85.0%*	*95.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*
Adjusted copper recovery	%	**86.6%**	0.0%	0.0%	0.0%	0.0%	74.1%	82.3%	86.8%	87.4%	87.5%	86.9%	85.7%	88.0%	87.7%	87.8%	86.5%	86.1%	85.4%	85.8%	85.9%
Adjusted molybdenum recovery	%	**44.3%**	0.0%	0.0%	0.0%	0.0%	0.0%	38.0%	40.3%	41.6%	43.5%	43.0%	42.8%	49.1%	47.0%	45.8%	43.3%	45.5%	47.1%	47.9%	43.9%
Adjusted gold recovery	%	**61.4%**	0.0%	0.0%	0.0%	0.0%	51.2%	56.9%	60.3%	60.4%	60.8%	60.6%	60.9%	62.2%	61.7%	61.5%	60.7%	61.0%	61.2%	61.5%	61.1%
METAL RECOVERED																					
unadjusted Cu recovered	**kt**	**7,358.6**	**0.0**	**0.0**	**0.0**	**0.0**	**132.3**	**252.8**	**271.5**	**276.2**	**304.6**	**323.3**	**319.8**	**321.9**	**289.3**	**294.9**	**265.5**	**242.7**	**240.8**	**231.6**	**242.8**
unadjusted Mo recovered	**kt**	**109.2**	**0.0**	**0.0**	**0.0**	**0.0**	**0.7**	**1.5**	**1.8**	**2.1**	**4.3**	**4.1**	**3.3**	**4.5**	**4.1**	**3.8**	**2.8**	**2.8**	**2.6**	**2.7**	**3.0**
unadjusted Au recovered	**k(t)oz**	**3,386.7**	**0.0**	**0.0**	**0.0**	**0.0**	**45.3**	**88.5**	**109.6**	**97.1**	**164.2**	**170.6**	**133.1**	**160.7**	**148.8**	**144.6**	**118.5**	**129.4**	**116.9**	**108.0**	**107.6**
adjusted Cu recovered	**kt**	**7,310.3**	**0.0**	**0.0**	**0.0**	**0.0**	**112.5**	**240.2**	**271.5**	**276.2**	**304.6**	**323.3**	**319.8**	**321.9**	**289.3**	**294.9**	**265.5**	**242.7**	**240.8**	**231.6**	**242.8**
adjusted Mo recovered	**kt**	**108.3**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**1.4**	**1.8**	**2.1**	**4.3**	**4.1**	**3.3**	**4.5**	**4.1**	**3.8**	**2.8**	**2.8**	**2.6**	**2.7**	**3.0**
adjusted Au recovered	**k(t)oz**	**3,369.6**	**0.0**	**0.0**	**0.0**	**0.0**	**38.5**	**84.1**	**109.6**	**97.1**	**164.2**	**170.6**	**133.1**	**160.7**	**148.8**	**144.6**	**118.5**	**129.4**	**116.9**	**108.0**	**107.6**
CONCENTRATE PRODUCED							1,384.3														
Cu concentrate	kt(wet)	**31,605.6**	0.0	0.0	0.0	0.0	485.3	1,061.5	1,198.8	1,197.6	1,316.6	1,384.3	1,376.7	1,343.6	1,213.5	1,239.0	1,162.5	1,081.8	1,084.8	1,029.9	1,058.9
Cu concentrate grade	%	**25.7%**	0.0%	0.0%	0.0%	0.0%	25.8%	25.1%	25.2%	25.6%	25.7%	26.0%	25.7%	26.6%	26.5%	26.5%	25.3%	24.9%	24.7%	25.0%	25.4%
Mo concentrate	kt(wet)	**256.1**	0.0	0.0	0.0	0.0	0.0	3.3	4.2	4.9	10.2	9.7	7.8	10.5	9.7	8.9	6.5	6.7	6.1	6.4	7.1
Mo concentrate grade	%	**47.0%**	0.0%	0.0%	0.0%	0.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%

PHYSICALS	UNITS	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27	Y28	Y29	Y30	Y31	Y32	Y33	Y34	Y35	Y36	Y37	Y38	Y39
MINING (AFTER MINING DILUTION & RECOVERY)																					
Ore mined total	Mt	100.4	105.8	91.7	86.8	86.4	71.3	69.1	77.0	78.9	82.5	31.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Waste mined to dump	Mt	72.0	59.2	31.9	22.0	22.1	18.9	21.2	13.5	11.3	4.5	1.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)																					
Total plant feed	**Mt**	**60.0**	**60.0**	**60.0**	**60.2**	**60.0**	**60.0**	**60.0**	**60.2**	**60.0**	**60.0**	**60.0**	**60.2**	**60.0**	**60.0**	**60.0**	**60.2**	**60.0**	**60.0**	**60.0**	**44.0**
	% Cu	**0.45**	**0.47**	**0.47**	**0.43**	**0.42**	**0.39**	**0.40**	**0.49**	**0.49**	**0.49**	**0.49**	**0.23**	**0.21**	**0.17**	**0.13**	**0.13**	**0.13**	**0.13**	**0.13**	**0.17**
	ppm Mo	**112.70**	**136.58**	**135.16**	**138.40**	**157.32**	**141.92**	**147.54**	**166.23**	**171.31**	**166.88**	**152.36**	**152.89**	**132.87**	**86.71**	**73.30**	**73.30**	**73.30**	**73.30**	**73.30**	**111.69**
	g/t Au	**0.08**	**0.10**	**0.08**	**0.08**	**0.08**	**0.08**	**0.07**	**0.08**	**0.08**	**0.07**	**0.09**	**0.06**	**0.09**	**0.04**	**0.03**	**0.03**	**0.03**	**0.03**	**0.03**	**0.08**
Cu insitu	**kt**	**270.0**	**282.0**	**282.0**	**258.7**	**252.0**	**234.0**	**240.0**	**294.8**	**294.0**	**294.0**	**294.0**	**138.4**	**126.0**	**102.5**	**78.0**	**78.2**	**78.0**	**78.0**	**78.0**	**76.2**
Mo insitu	**kt**	**6.8**	**8.2**	**8.1**	**8.3**	**9.4**	**8.5**	**8.9**	**10.0**	**10.3**	**10.0**	**9.1**	**9.2**	**8.0**	**5.2**	**4.4**	**4.4**	**4.4**	**4.4**	**4.4**	**4.9**
Au insitu	**k(t)oz**	**160.1**	**183.3**	**160.1**	**150.9**	**158.2**	**148.5**	**140.8**	**152.8**	**162.0**	**140.8**	**167.8**	**118.0**	**167.8**	**86.7**	**57.9**	**58.0**	**57.9**	**57.9**	**57.9**	**113.3**
AVERAGE RECOVERIES																					
Copper recovery	%	85.8%	87.2%	88.0%	88.2%	88.1%	88.3%	88.4%	88.6%	88.3%	88.4%	88.1%	87.5%	86.0%	71.4%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%
Molybdenum recovery	%	43.9%	46.2%	48.0%	47.6%	46.3%	47.1%	45.9%	44.2%	41.7%	42.4%	44.0%	44.1%	45.3%	48.6%	44.3%	44.3%	44.3%	44.3%	44.3%	24.0%
Gold recovery	%	60.6%	61.3%	62.2%	62.8%	61.9%	62.2%	61.7%	61.4%	60.4%	60.9%	61.4%	61.2%	61.0%	71.1%	68.7%	68.7%	68.7%	68.7%	68.7%	36.6%
Ramp-up factor	*%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*	*100.0%*
Adjusted copper recovery	%	85.8%	87.2%	88.0%	88.2%	88.1%	88.3%	88.4%	88.6%	88.3%	88.4%	88.1%	87.5%	86.0%	71.4%	84.8%	84.8%	84.8%	84.8%	84.8%	84.8%
Adjusted molybdenum recovery	%	43.9%	46.2%	48.0%	47.6%	46.3%	47.1%	45.9%	44.2%	41.7%	42.4%	44.0%	44.1%	45.3%	48.6%	44.3%	44.3%	44.3%	44.3%	44.3%	24.0%
Adjusted gold recovery	%	60.6%	61.3%	62.2%	62.8%	61.9%	62.2%	61.7%	61.4%	60.4%	60.9%	61.4%	61.2%	61.0%	71.1%	68.7%	68.7%	68.7%	68.7%	68.7%	36.6%
METAL RECOVERED																					
unadjusted Cu recovered	**kt**	**231.7**	**245.9**	**248.1**	**228.3**	**222.1**	**206.6**	**212.1**	**261.2**	**259.6**	**259.9**	**259.1**	**121.0**	**108.3**	**73.2**	**66.1**	**66.3**	**66.1**	**66.1**	**66.1**	**80.4**
unadjusted Mo recovered	**kt**	**3.0**	**3.8**	**3.9**	**4.0**	**4.4**	**4.0**	**4.1**	**4.4**	**4.3**	**4.2**	**4.0**	**4.1**	**3.6**	**2.5**	**1.9**	**2.0**	**1.9**	**1.9**	**1.9**	**1.2**
unadjusted Au recovered	**k(t)oz**	**96.9**	**112.4**	**99.5**	**94.7**	**97.9**	**92.4**	**86.9**	**93.8**	**97.9**	**85.7**	**103.0**	**72.3**	**102.4**	**61.6**	**39.8**	**39.9**	**39.8**	**39.8**	**39.8**	**47.5**
adjusted Cu recovered	**kt**	**231.7**	**245.9**	**248.1**	**228.3**	**222.1**	**206.6**	**212.1**	**261.2**	**259.6**	**259.9**	**259.1**	**121.0**	**108.3**	**73.2**	**66.1**	**66.3**	**66.1**	**66.1**	**66.1**	**64.6**
adjusted Mo recovered	**kt**	**3.0**	**3.8**	**3.9**	**4.0**	**4.4**	**4.0**	**4.1**	**4.4**	**4.3**	**4.2**	**4.0**	**4.1**	**3.6**	**2.5**	**1.9**	**2.0**	**1.9**	**1.9**	**1.9**	**1.2**
adjusted Au recovered	**k(t)oz**	**96.9**	**112.4**	**99.5**	**94.7**	**97.9**	**92.4**	**86.9**	**93.8**	**97.9**	**85.7**	**103.0**	**72.3**	**102.4**	**61.6**	**39.8**	**39.9**	**39.8**	**39.8**	**39.8**	**41.5**
CONCENTRATE PRODUCED																					
Cu concentrate	kt(wet)	1,006.8	1,062.7	1,073.2	986.5	958.7	897.5	901.8	1,074.1	1,055.5	1,061.3	1,078.0	534.9	507.6	344.6	312.2	312.2	312.2	312.2	312.2	266.8
Cu concentrate grade	%	25.6%	25.7%	25.7%	25.6%	25.7%	25.6%	26.1%	27.0%	27.3%	27.2%	26.7%	25.1%	23.7%	23.6%	23.5%	23.5%	23.5%	23.5%	23.5%	23.5%
Mo concentrate	kt(wet)	7.0	9.0	9.2	9.4	10.3	9.5	9.6	10.4	10.1	10.0	9.5	9.6	8.5	6.0	4.6	4.6	4.6	4.6	4.6	2.9
Mo concentrate grade	%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%	47.0%

Table 22-12 Stage 2 60 Mtpa cashflow model summary

CASHFLOWS	UNITS	TOTAL	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16	Y17	Y18	Y19
PAYABILITY																					
Cu	%	**96.1%**	0.0%	0.0%	0.0%	0.0%	96.1%	96.0%	96.0%	96.1%	96.1%	96.1%	96.1%	96.2%	96.2%	96.2%	96.0%	96.0%	95.9%	96.0%	96.1%
Mo	%	**86.0%**	0.0%	0.0%	0.0%	0.0%	0.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%
Au	%	**90.0%**	0.0%	0.0%	0.0%	0.0%	0.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
Payable metal recovered																					
Cu	**kt**	**7,009.9**	**0.0**	**0.0**	**0.0**	**0.0**	**108.1**	**230.6**	**260.7**	**265.5**	**292.8**	**310.8**	**307.4**	**309.8**	**278.4**	**283.8**	**255.0**	**233.0**	**231.0**	**222.4**	**233.3**
Mo	**kt**	**92.8**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**1.2**	**1.5**	**1.8**	**3.7**	**3.5**	**2.9**	**3.8**	**3.5**	**3.2**	**2.4**	**2.4**	**2.2**	**2.3**	**2.6**
Au	**k(t)oz**	**2,975.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**75.7**	**98.7**	**87.4**	**147.8**	**153.5**	**119.8**	**144.6**	**134.0**	**130.2**	**106.7**	**116.4**	**105.2**	**97.2**	**96.8**
GROSS REVENUE																					
Metal prices																					
Cu	$/lb	**$4.50**	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50
Mo	$/lb	**$18.00**	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00
Au	$/oz	**$3,000**	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000
Revenue after payability																					
Copper revenue	**$M**	**$69,697.3**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$1,072.4**	**$2,287.9**	**$2,586.1**	**$2,633.7**	**$2,904.7**	**$3,083.8**	**$3,049.3**	**$3,073.5**	**$2,761.8**	**$2,815.4**	**$2,530.3**	**$2,311.4**	**$2,291.8**	**$2,206.0**	**$2,314.1**
Molybdenum revenue	**$M**	**$3,697.5**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$47.9**	**$61.0**	**$70.4**	**$147.1**	**$140.1**	**$113.4**	**$152.0**	**$139.5**	**$128.9**	**$94.6**	**$97.0**	**$87.9**	**$91.7**	**$102.7**
Gold revenue	**$M**	**$9,097.8**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$104.0**	**$227.1**	**$296.0**	**$262.3**	**$443.3**	**$460.5**	**$359.4**	**$433.9**	**$401.9**	**$390.5**	**$320.0**	**$349.3**	**$315.6**	**$291.6**	**$290.5**
Total revenue	**$M**	**$82,492.6**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$1,176.4**	**$2,562.9**	**$2,943.0**	**$2,966.3**	**$3,495.1**	**$3,684.5**	**$3,522.1**	**$3,659.5**	**$3,303.2**	**$3,334.8**	**$2,944.8**	**$2,757.7**	**$2,695.3**	**$2,589.3**	**$2,707.3**
METAL COSTS																					
TCRCs and freight	$M	**$7,622.6**	$0.0	$0.0	$0.0	$0.0	$127.9	$249.1	$278.2	$279.0	$305.5	$320.3	$318.4	$313.3	$285.2	$290.4	$272.0	$253.8	$253.8	$242.9	$250.5
Royalties	$M	**$1,123.1**	$0.0	$0.0	$0.0	$0.0	$15.7	$34.7	$40.0	$40.3	$47.8	$50.5	$48.1	$50.2	$45.3	$45.7	$40.1	$37.6	$36.6	$35.2	$36.9
Total metal costs	**$M**	**$8,745.6**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$143.6**	**$283.8**	**$318.2**	**$319.3**	**$353.4**	**$370.7**	**$366.5**	**$363.5**	**$330.5**	**$336.1**	**$312.1**	**$291.4**	**$290.4**	**$278.1**	**$287.3**
CAPITAL COSTS																					
Mining	$M	**$1,861.2**	$277.8	$498.0	$453.8	$428.7	$97.6	$105.2	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Processing	$M	**$1,606.2**	$145.2	$191.3	$325.8	$297.9	$117.3	$79.3	$237.9	$211.5	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Rail	$M	**$12.8**	$1.7	$2.3	$3.9	$3.5	$1.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Infrastructure	$M	**$945.8**	$98.0	$129.1	$219.8	$201.0	$79.1	$32.8	$98.5	$87.5	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Indirects	$M	**$824.4**	$88.8	$116.9	$199.1	$182.0	$71.7	$24.9	$74.7	$66.4	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal development capital	$M	**$5,250.4**	**$611.6**	**$937.5**	**$1,202.4**	**$1,113.1**	**$367.1**	**$242.2**	**$411.1**	**$365.4**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Deferred stripping	$M	**$1,243.9**	$0.0	$0.0	$0.0	$0.0	$114.0	$166.4	$192.1	$194.5	$158.3	$113.9	$47.3	$32.4	$0.0	$0.0	$40.6	$69.1	$76.6	$38.8	$0.0
Closure costs	$M	**$71.7**	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Total capital costs	**$M**	**$6,566.0**	**$611.6**	**$937.5**	**$1,202.4**	**$1,113.1**	**$481.1**	**$408.6**	**$603.2**	**$559.9**	**$158.3**	**$113.9**	**$47.3**	**$32.4**	**$0.0**	**$0.0**	**$40.6**	**$69.1**	**$76.6**	**$38.8**	**$0.0**
SUSTAINING COSTS																					
Mining	$M	**$2,267.3**	$0.0	$0.0	$0.0	$0.0	$38.0	$55.9	$96.3	$86.8	$173.3	$138.0	$80.8	$122.7	$86.4	$126.2	$126.3	$111.7	$71.2	$111.0	$53.5
Processing and infrastructure	$M	**$605.6**	$0.0	$0.0	$0.0	$0.0	$11.1	$13.5	$13.5	$13.5	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0
Total sustaining costs	**$M**	**$2,872.9**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$49.1**	**$69.4**	**$109.8**	**$100.3**	**$191.2**	**$156.0**	**$98.8**	**$140.7**	**$104.4**	**$144.2**	**$144.3**	**$129.7**	**$89.2**	**$129.0**	**$71.6**
OPERATING COSTS		$0.05																			
Mining	$M	**$8,356.1**	$0.0	$0.0	$0.9	$43.2	$230.5	$223.6	$200.9	$196.2	$250.9	$324.4	$365.7	$402.3	$420.4	$438.5	$366.0	$353.3	$336.7	$391.3	$363.6
Processing	$M	**$13,457.5**	$0.0	$0.0	$0.0	$0.0	$245.8	$299.7	$300.0	$299.5	$399.6	$399.9	$400.3	$399.6	$399.6	$399.9	$400.3	$399.6	$399.6	$399.9	$400.3
General and administration	$M	**$2,777.8**	$0.0	$0.0	$0.0	$0.0	$49.3	$68.8	$70.6	$70.6	$90.5	$91.3	$90.7	$91.3	$89.6	$89.7	$88.0	$87.0	$86.7	$86.3	$87.0
Other	$M	**$92.4**	$1.8	$1.8	$1.8	$1.8	$1.8	$1.8	$2.5	$1.8	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5
Total operating costs	**$M**	**$24,683.7**	**$1.8**	**$1.8**	**$2.8**	**$45.1**	**$527.5**	**$593.9**	**$573.9**	**$568.1**	**$743.4**	**$818.1**	**$859.2**	**$895.6**	**$912.1**	**$930.6**	**$856.9**	**$842.4**	**$825.5**	**$879.9**	**$853.4**
OTHER COSTS																					
Taxes on bank transactions	$M	**$144.7**	$3.7	$5.6	$7.2	$7.0	$3.9	$4.1	$4.8	$4.6	$4.4	$4.4	$4.1	$4.3	$4.1	$4.3	$4.2	$4.2	$4.0	$4.2	$3.8
MINERAL RESERVE CASHFLOW (PRE-TAX)																					
Net return	$M	**$74,870.0**	$0.0	$0.0	$0.0	$0.0	$1,048.5	$2,313.9	$2,664.8	$2,687.4	$3,189.6	$3,364.2	$3,203.6	$3,346.1	$3,018.0	$3,044.4	$2,672.7	$2,503.9	$2,441.5	$2,346.4	$2,456.9
Opex	$M	**$24,683.7**	$1.8	$1.8	$2.8	$45.1	$527.5	$593.9	$573.9	$568.1	$743.4	$818.1	$859.2	$895.6	$912.1	$930.6	$856.9	$842.4	$825.5	$879.9	$853.4
Royalties and other	$M	**$1,267.7**	$3.7	$5.6	$7.2	$7.0	$19.7	$38.8	$44.7	$44.9	$52.2	$54.8	$52.1	$54.5	$49.4	$50.0	$44.3	$41.8	$40.7	$39.4	$40.6
Capex	$M	**$6,566.0**	$611.6	$937.5	$1,202.4	$1,113.1	$481.1	$408.6	$603.2	$559.9	$158.3	$113.9	$47.3	$32.4	$0.0	$0.0	$40.6	$69.1	$76.6	$38.8	$0.0
Sustaining	$M	**$2,872.9**	$0.0	$0.0	$0.0	$0.0	$49.1	$69.4	$109.8	$100.3	$191.2	$156.0	$98.8	$140.7	$104.4	$144.2	$144.3	$129.7	$89.2	$129.0	$71.6
Working capex	$M	**$0.0**	$0.5	$0.0	$0.2	$11.0	$207.0	$119.2	$21.8	$0.5	$85.3	$36.0	$0.9	$20.4	-$20.8	$5.9	-$50.9	-$18.0	-$8.6	$6.3	$4.9
Undiscounted cashflow	**$M**	**$39,479.7**	**-$617.6**	**-$945.0**	**-$1,212.7**	**-$1,176.2**	**-$235.8**	**$1,083.9**	**$1,311.3**	**$1,413.6**	**$1,959.2**	**$2,185.4**	**$2,145.3**	**$2,202.5**	**$1,972.9**	**$1,913.7**	**$1,637.6**	**$1,439.0**	**$1,418.2**	**$1,252.9**	**$1,486.4**
NPV_{10} (indicative)	**$M**	**$6,433.2**																			
NPV_8 (indicative)	**$M**	**$9,087.3**																			
IRR	**%**	**23.3%**																			
MINERAL RESERVE CASHFLOW (POST-TAX)																					
Taxable income	**$M**	**$37,129.3**	$0.0	$0.0	$0.0	$0.0	$0.0	-$110.0	$1,318.6	$1,488.0	$1,821.8	$1,951.4	$1,807.0	$1,926.4	$1,625.9	$1,635.5	$1,355.9	$1,232.6	$1,216.4	$1,082.3	$1,221.5
Tax paid	**$M**	**-$11,072.2**	$0.0	$0.0	$0.0	$0.0	-$23.3	-$31.7	-$399.3	-$442.4	-$537.5	-$575.3	-$534.4	-$568.6	-$483.7	-$486.9	-$405.8	-$369.9	-$364.0	-$327.7	-$365.7
Undiscounted cashflow	**$M**	**$28,407.5**	**-$617.6**	**-$945.0**	**-$1,212.7**	**-$1,176.2**	**-$259.2**	**$1,052.1**	**$912.0**	**$971.2**	**$1,421.8**	**$1,610.1**	**$1,610.9**	**$1,633.8**	**$1,489.2**	**$1,426.8**	**$1,231.8**	**$1,069.1**	**$1,054.2**	**$925.3**	**$1,120.7**
NPV_{10} (indicative)	**$M**	**$3,973.3**																			
NPV_8 (indicative)	**$M**	**$5,917.1**																			
IRR	**%**	**19.3%**																			

CASHFLOWS	Y20	Y21	Y22	Y23	Y24	Y25	Y26	Y27	Y28	Y29	Y30	Y31	Y32	Y33	Y34	Y35	Y36	Y37	Y38	Y39
PAYABILITY																				
Cu	96.1%	96.1%	96.1%	96.1%	96.1%	96.1%	96.2%	96.3%	96.3%	96.3%	96.3%	96.0%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%	95.8%
Mo	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%	86.0%
Au	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
Payable metal recovered																				
Cu	**222.6**	**236.3**	**238.5**	**219.4**	**213.5**	**198.5**	**204.0**	**251.5**	**250.1**	**250.3**	**249.4**	**116.2**	**103.7**	**70.1**	**63.3**	**63.5**	**63.3**	**63.3**	**63.3**	**46.4**
Mo	**2.6**	**3.3**	**3.3**	**3.4**	**3.8**	**3.4**	**3.5**	**3.8**	**3.7**	**3.7**	**3.5**	**3.5**	**3.1**	**2.2**	**1.7**	**1.7**	**1.7**	**1.7**	**1.7**	**0.7**
Au	**87.3**	**101.2**	**89.6**	**85.2**	**88.1**	**83.1**	**78.2**	**84.4**	**88.1**	**77.1**	**92.7**	**65.0**	**92.2**	**55.5**	**35.8**	**35.9**	**35.8**	**35.8**	**35.8**	**14.4**
GROSS REVENUE																				
Metal prices																				
Cu	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50	$4.50
Mo	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00	$18.00
Au	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000	$3,000
Revenue after payability																				
Copper revenue	**$2,208.6**	**$2,344.7**	**$2,365.8**	**$2,176.2**	**$2,117.7**	**$1,969.0**	**$2,023.6**	**$2,495.4**	**$2,481.5**	**$2,483.6**	**$2,474.5**	**$1,152.6**	**$1,029.2**	**$695.4**	**$628.3**	**$630.0**	**$628.3**	**$628.3**	**$628.3**	**$613.6**
Molybdenum revenue	**$101.2**	**$129.3**	**$132.8**	**$135.4**	**$149.1**	**$136.9**	**$138.6**	**$150.8**	**$146.2**	**$144.9**	**$137.3**	**$138.4**	**$123.3**	**$86.3**	**$66.5**	**$66.7**	**$66.5**	**$66.5**	**$66.5**	**$40.2**
Gold revenue	**$261.8**	**$303.5**	**$268.8**	**$255.7**	**$264.3**	**$249.4**	**$234.5**	**$253.3**	**$264.3**	**$231.4**	**$278.0**	**$195.1**	**$276.5**	**$166.4**	**$107.4**	**$107.7**	**$107.4**	**$107.4**	**$107.4**	**$112.1**
Total revenue	**$2,571.6**	**$2,777.5**	**$2,767.4**	**$2,567.2**	**$2,531.0**	**$2,355.3**	**$2,396.7**	**$2,899.6**	**$2,892.0**	**$2,859.9**	**$2,889.9**	**$1,486.1**	**$1,429.0**	**$948.1**	**$802.2**	**$804.4**	**$802.2**	**$802.2**	**$802.2**	**$765.9**
METAL COSTS																				
TCRCs and freight	$239.1	$251.6	$253.7	$236.0	$229.9	$216.6	$218.3	$257.0	$253.1	$254.0	$256.7	$140.0	$132.8	$98.3	$91.2	$91.3	$91.2	$91.2	$91.2	$89.3
Royalties	$35.0	$37.9	$37.7	$35.0	$34.5	$32.1	$32.7	$39.6	$39.6	$39.1	$39.5	$20.2	$19.4	$12.7	$10.7	$10.7	$10.7	$10.7	$10.7	$10.1
Total metal costs	**$274.1**	**$289.5**	**$291.4**	**$270.9**	**$264.4**	**$248.7**	**$251.0**	**$296.7**	**$292.7**	**$293.1**	**$296.2**	**$160.2**	**$152.2**	**$111.0**	**$101.8**	**$102.0**	**$101.8**	**$101.8**	**$101.8**	**$99.4**
CAPITAL COSTS																				
Mining	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Processing	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Rail	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Infrastructure	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Indirects	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Subtotal development capital	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**
Deferred stripping	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Closure costs	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$21.2
Total capital costs	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$0.0**	**$21.2**
SUSTAINING COSTS																				
Mining	$108.4	$74.8	$102.7	$28.5	$122.1	$70.4	$28.5	$57.7	$29.0	$23.5	$18.1	$17.5	$17.1	$9.7	$25.4	$12.0	$10.9	$19.9	$7.8	$5.0
Processing and infrastructure	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$18.0	$14.3
Total sustaining costs	**$126.4**	**$92.8**	**$120.6**	**$46.5**	**$140.1**	**$88.4**	**$46.5**	**$75.7**	**$47.0**	**$41.5**	**$36.1**	**$35.5**	**$35.1**	**$27.7**	**$43.4**	**$30.0**	**$28.9**	**$37.9**	**$25.8**	**$19.3**
OPERATING COSTS																				
Mining	$366.7	$356.9	$292.2	$275.9	$272.7	$236.1	$237.6	$231.7	$235.3	$243.1	$123.6	$69.0	$71.4	$67.0	$70.1	$69.7	$70.1	$69.9	$54.4	$34.1
Processing	$399.6	$399.6	$399.9	$400.3	$399.6	$399.6	$399.9	$400.3	$399.6	$399.6	$399.9	$400.3	$399.6	$399.6	$399.9	$400.3	$399.6	$399.6	$399.9	$318.1
General and administration	$86.2	$87.2	$87.1	$86.3	$86.0	$85.2	$85.4	$87.9	$87.7	$87.6	$87.7	$69.0	$68.6	$66.4	$65.7	$65.8	$65.7	$65.7	$65.7	$53.6
Other	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5	$2.5
Total operating costs	**$854.9**	**$846.1**	**$781.7**	**$765.0**	**$760.8**	**$723.4**	**$725.4**	**$722.4**	**$725.1**	**$732.7**	**$613.7**	**$540.7**	**$542.0**	**$535.4**	**$538.1**	**$538.3**	**$537.8**	**$537.6**	**$522.4**	**$408.3**
OTHER COSTS																				
Taxes on bank transactions	$4.0	$3.8	$3.6	$3.3	$3.6	$3.3	$3.2	$3.3	$3.2	$3.2	$2.7	$2.4	$2.4	$2.3	$2.4	$2.3	$2.3	$2.4	$2.3	$1.8
MINERAL RESERVE CASHFLOW (PRE-TAX)																				
Net return	$2,332.5	$2,525.9	$2,513.7	$2,331.3	$2,301.1	$2,138.7	$2,178.4	$2,642.5	$2,638.9	$2,605.9	$2,633.2	$1,346.1	$1,296.2	$849.8	$711.1	$713.1	$711.1	$711.1	$711.1	$676.6
Opex	$854.9	$846.1	$781.7	$765.0	$760.8	$723.4	$725.4	$722.4	$725.1	$732.7	$613.7	$540.7	$542.0	$535.4	$538.1	$538.3	$537.8	$537.6	$522.4	$408.3
Royalties and other	$38.9	$41.7	$41.3	$38.3	$38.1	$35.4	$35.9	$42.9	$42.8	$42.3	$42.2	$22.6	$21.8	$15.1	$13.0	$13.0	$13.0	$13.0	$12.9	$11.9
Capex	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$21.2
Sustaining	$126.4	$92.8	$120.6	$46.5	$140.1	$88.4	$46.5	$75.7	$47.0	$41.5	$36.1	$35.5	$35.1	$27.7	$43.4	$30.0	$28.9	$37.9	$25.8	$19.3
Working capex	-$11.3	$14.5	-$18.5	-$17.2	-$6.2	-$21.6	$4.9	$36.6	$1.2	-$0.6	-$28.5	-$124.7	-$3.8	-$38.2	-$11.1	$0.6	-$0.2	-$0.3	-$3.6	-$32.3
Undiscounted cashflow	**$1,323.6**	**$1,530.8**	**$1,588.5**	**$1,498.6**	**$1,368.4**	**$1,313.2**	**$1,365.7**	**$1,764.9**	**$1,822.9**	**$1,790.0**	**$1,969.9**	**$872.0**	**$701.1**	**$309.8**	**$127.6**	**$131.2**	**$131.6**	**$122.9**	**$153.6**	**$248.1**
NPV_{10} (indicative)																				
NPV_8 (indicative)																				
IRR																				
MINERAL RESERVE CASHFLOW (POST-TAX)																				
Taxable income	$1,118.9	$1,301.6	$1,365.8	$1,251.6	$1,228.7	$1,103.8	$1,191.5	$1,631.1	$1,636.0	$1,626.4	$1,790.9	$685.1	$643.2	$233.7	$99.7	$101.3	$108.6	$106.4	$125.0	$206.6
Tax paid	-$336.4	-$387.8	-$403.4	-$369.3	-$362.7	-$326.5	-$349.6	-$473.4	-$474.5	-$471.2	-$512.7	-$197.9	-$186.0	-$70.2	-$32.6	-$33.0	-$34.8	-$34.3	-$38.8	-$60.9
Undiscounted cashflow	**$987.2**	**$1,143.0**	**$1,185.2**	**$1,129.3**	**$1,005.7**	**$986.7**	**$1,016.1**	**$1,291.6**	**$1,348.4**	**$1,318.8**	**$1,457.1**	**$674.2**	**$515.1**	**$239.6**	**$95.0**	**$98.1**	**$96.8**	**$88.6**	**$114.7**	**$187.2**
NPV_{10} (indicative)																				
NPV_8 (indicative)																				
IRR																				

22.4 Sensitivity analysis

A pre-tax cashflow sensitivity analysis was completed as part of the pit optimisation work described in Item 15. The most sensitive variable was shown to be copper metal price (and recovery, since the magnitude of impact is similar). Further to the optimisation analyses and referring to the Project post-tax cashflow model summarised in Table 22-7 to Table 22-10, a range of variables for testing the Stage 1 40 Mtpa cashflow model sensitivity are listed in Table 22-13.

Table 22-14 and Table 22-15 show, respectively, the impact to NPV_8 and IRR when testing the cashflow model sensitivity. Comments on this analysis are as follows:

- copper metal price and copper recovery are confirmed to be the most sensitive variables
- the next most sensitive cashflow model item is the magnitude of the development capital costs; a 10% increase in which would reduce the NPV by 8% and the IRR by 8%
- processing and mine operating costs follow next and then gold metal price and gold recovery
- the least sensitive variables are the molybdenum price and recovery, TCRCs and G&A operating costs

Table 22-13 40 Mtpa discounted cashflow model sensitivity analysis parameters

Sensitivity parameter 40 Mtpa	80%	90%	Base	110%	120%
Cu metal price ($/lb)	$3.60	$4.05	**$4.50**	$4.95	$5.40
Cu recovery (%)	69.4%	78.1%	**86.8%**	95.4%	
Au metal price ($/oz)	$2,400	$2,700	**$3,000**	$3,300	$3,600
Au recovery (%)	49.2%	55.3%	**61.5%**	67.6%	73.8%
Mo metal price ($/lb)	$14.40	$16.20	**$18.00**	$19.80	$21.60
Mo recovery (%)	35.5%	39.9%	**44.3%**	48.8%	53.2%
Development capital costs ($M)	$3,385	$3,809	**$4,232**	$4,655	$5,078
Process operating costs ($/t processed)	$5.80	$6.52	**$7.25**	$7.97	$8.70
Mining operating costs ($/t mined)	$1.54	$1.74	**$1.93**	$2.12	$2.32
Sustaining and other capital costs ($M)	$3,509	$3,948	**$4,387**	$4,825	$5,264
Copper freight charges ($/wmt)	$106.85	$120.20	**$133.56**	$146.92	$160.27
Copper treatment charges ($/dmt)	$68.00	$76.50	**$85.00**	$93.50	$102.00
G&A operating costs ($/t processed)	$1.21	$1.36	**$1.52**	$1.67	$1.82

Table 22-14 40 Mtpa discounted cashflow model (post-tax) sensitivity analysis, NPV impact

Sensitivity parameter	NPV_8 ($M) post-tax					Change relative to Base			
	80%	90%	Base	110%	120%	80%	90%	110%	120%
Cu metal price	$2,212	$3,453	**$4,691**	$5,926	$7,161	47%	74%	126%	153%
Cu recovery	$2,448	$3,571	**$4,691**	$5,809	n/a	52%	76%	124%	
Au metal price	$4,387	$4,539	**$4,691**	$4,844	$4,996	94%	97%	103%	106%
Au recovery	$4,387	$4,539	**$4,691**	$4,844	$4,996	94%	97%	103%	106%
Mo metal price	$4,595	$4,643	**$4,691**	$4,739	$4,788	98%	99%	101%	102%
Mo recovery	$4,597	$4,644	**$4,691**	$4,739	$4,786	98%	99%	101%	102%
Development capital costs	$5,400	$5,046	**$4,691**	$4,337	$3,983	115%	108%	92%	85%
Process operating costs	$5,108	$4,900	**$4,691**	$4,483	$4,272	109%	104%	96%	91%
Mining operating costs	$5,030	$4,861	**$4,691**	$4,522	$4,353	107%	104%	96%	93%
Sustaining and other capital costs	$4,864	$4,778	**$4,691**	$4,605	$4,519	104%	102%	98%	96%
Copper freight charges	$4,829	$4,760	**$4,691**	$4,622	$4,553	103%	101%	99%	97%
Copper treatment charges	$4,777	$4,734	**$4,691**	$4,649	$4,606	102%	101%	99%	98%
G&A operating costs	$4,782	$4,737	**$4,691**	$4,646	$4,601	102%	101%	99%	98%

Table 22-15 40 Mtpa discounted cashflow model (post-tax) sensitivity analysis IRR impact

Sensitivity parameter	IRR (%) post-tax					Change relative to Base			
	80%	90%	Base	110%	120%	80%	90%	110%	120%
Cu metal price	13.4%	16.0%	**18.4%**	20.7%	22.9%	72%	87%	112%	124%
Cu recovery	13.9%	16.2%	**18.4%**	20.5%	n/a	75%	88%	111%	
Au metal price	17.9%	18.2%	**18.4%**	18.7%	19.0%	97%	98%	101%	103%
Au recovery	17.9%	18.2%	**18.4%**	18.7%	19.0%	97%	98%	101%	103%
Mo metal price	18.3%	18.4%	**18.4%**	18.5%	18.6%	99%	100%	100%	101%
Mo recovery	18.3%	18.4%	**18.4%**	18.5%	18.6%	99%	100%	100%	101%
Development capital costs	22.1%	20.1%	**18.4%**	17.0%	15.7%	120%	109%	92%	85%
Process operating costs	19.2%	18.8%	**18.4%**	18.1%	17.7%	104%	102%	98%	96%
Mining operating costs	19.1%	18.8%	**18.4%**	18.1%	17.8%	103%	102%	98%	96%
Sustaining and other capital costs	18.8%	18.6%	**18.4%**	18.3%	18.1%	102%	101%	99%	98%
Copper freight charges	18.7%	18.6%	**18.4%**	18.3%	18.2%	101%	101%	99%	99%
Copper treatment charges	18.6%	18.5%	**18.4%**	18.4%	18.3%	101%	100%	100%	99%
G&A operating costs	18.6%	18.5%	**18.4%**	18.4%	18.3%	101%	100%	100%	99%

Stage 2 60 Mtpa cashflow model sensitivity parameters are listed in Table 22-16,whilst Table 22-17 and Table 22-18 show, respectively, the impact to NPV_8 and IRR when testing the 60 Mtpa cashflow model sensitivity.

Table 22-16 60 Mtpa discounted cashflow model sensitivity analysis parameters

Sensitivity parameter 60 Mtpa	80%	90%	Base	110%	120%
Cu metal price ($/lb)	$3.60	$4.05	**$4.50**	$4.95	$5.40
Cu recovery (%)	69.2%	77.9%	**86.5%**	95.2%	
Au metal price ($/oz)	$2,400	$2,700	**$3,000**	$3,300	$3,600
Au recovery (%)	48.8%	54.9%	**61.0%**	67.1%	73.2%
Mo metal price ($/lb)	$14.40	$16.20	**$18.00**	$19.80	$21.60
Mo recovery (%)	35.3%	39.7%	**44.1%**	48.5%	52.9%
Development capital costs ($M)	$4,200	$4,725	**$5,250**	$5,775	$6,300
Process operating costs ($/t processed)	$5.41	$6.09	**$6.76**	$7.44	$8.11
Mining operating costs ($/t mined)	$1.50	$1.69	**$1.88**	$2.07	$2.26
Sustaining and other capital costs ($M)	$3,293	$3,705	**$4,117**	$4,528	$4,940
Copper freight charges ($/wmt)	$96.50	$108.56	**$120.62**	$132.69	$144.75
Copper treatment charges ($/dmt)	$68.00	$76.50	**$85.00**	$93.50	$102.00
G&A operating costs ($/t processed)	$1.12	$1.26	**$1.40**	$1.54	$1.67

Table 22-17 60 Mtpa discounted cashflow model (post-tax) sensitivity analysis, NPV impact

Sensitivity parameter	NPV_8 ($M) post-tax					Change relative to Base			
	80%	90%	Base	110%	120%	80%	90%	110%	120%
Cu metal price	$2,984	$4,451	**$5,917**	$7,381	$8,843	50%	75%	125%	149%
Cu recovery	$3,265	$4,591	**$5,917**	$7,241	n/a	55%	78%	122%	
Au metal price	$5,545	$5,731	**$5,917**	$6,103	$6,289	94%	97%	103%	106%
Au recovery	$5,546	$5,731	**$5,917**	$6,103	$6,288	94%	97%	103%	106%
Mo metal price	$5,792	$5,855	**$5,917**	$5,980	$6,042	98%	99%	101%	102%
Mo recovery	$5,794	$5,855	**$5,917**	$5,979	$6,040	98%	99%	101%	102%
Development capital costs	$6,748	$6,333	**$5,917**	$5,502	$5,086	114%	107%	93%	86%
Process operating costs	$6,428	$6,173	**$5,917**	$5,661	$5,406	109%	104%	96%	91%
Mining operating costs	$6,310	$6,113	**$5,917**	$5,721	$5,525	107%	103%	97%	93%
Sustaining and other capital costs	$6,107	$6,012	**$5,917**	$5,822	$5,728	103%	102%	98%	97%
Copper freight charges	$6,073	$5,995	**$5,917**	$5,839	$5,761	103%	101%	99%	97%
Copper treatment charges	$6,019	$5,968	**$5,917**	$5,866	$5,815	102%	101%	99%	98%
G&A operating costs	$6,025	$5,971	**$5,917**	$5,863	$5,809	102%	101%	99%	98%

Table 22-18 60 Mtpa discounted cashflow model (post-tax) sensitivity analysis IRR impact

Sensitivity parameter	IRR (%) post-tax					Change relative to Base			
	80%	**90%**	**Base**	**110%**	**120%**	**80%**	**90%**	**110%**	**120%**
Cu metal price	14.3%	16.9%	**19.3%**	21.6%	23.7%	74%	87%	112%	122%
Cu recovery	14.8%	17.2%	**19.3%**	21.4%	n/a	77%	89%	111%	
Au metal price	18.8%	19.1%	**19.3%**	19.6%	19.9%	97%	99%	101%	103%
Au recovery	18.8%	19.1%	**19.3%**	19.6%	19.9%	97%	99%	101%	103%
Mo metal price	19.2%	19.3%	**19.3%**	19.4%	19.5%	99%	100%	100%	101%
Mo recovery	19.2%	19.3%	**19.3%**	19.4%	19.5%	99%	100%	100%	101%
Development capital costs	23.0%	21.0%	**19.3%**	17.9%	16.6%	119%	109%	92%	86%
Process operating costs	20.1%	19.7%	**19.3%**	19.0%	18.6%	104%	102%	98%	96%
Mining operating costs	20.0%	19.7%	**19.3%**	19.0%	18.7%	103%	102%	98%	97%
Sustaining and other capital costs	19.7%	19.5%	**19.3%**	19.2%	19.0%	102%	101%	99%	98%
Copper freight charges	19.6%	19.5%	**19.3%**	19.2%	19.1%	101%	101%	99%	99%
Copper treatment charges	19.5%	19.4%	**19.3%**	19.3%	19.2%	101%	100%	100%	99%
G&A operating costs	19.5%	19.4%	**19.3%**	19.3%	19.2%	101%	100%	100%	99%

Similar sensitivity trends are noted as for the 40 Mtpa scenario. In this instance and in regard to the magnitude of the development capital costs, a 10% increase would reduce the NPV by 7% and the IRR by 8%.

ITEM 23 ADJACENT PROPERTIES

The porphyry copper deposit that is the subject of this Technical Report is the Taca Taca Bajo deposit. The Taca Taca Alto deposit is located 4 km west, outside of the Project area.

The QPs for this Technical Report have not considered the Taca Taca Alto deposit, in respect of any similarities to Taca Taca Bajo geology or mineralisation style.

ITEM 24 OTHER RELEVANT DATA AND INFORMATION

The supply of water in such an arid environment as the Argentine Altiplano is a critical engineering aspect for development of the Project. The following commentary is included here to provide an update on the water supply investigations since the 2021 Technical Report, and which remain in progress.

FQM commissioned several consultants to advise on aspects of Project water supply, namely:

- Piteau Associates (Piteau, 2022, 2025); hydrogeological settings for fresh water supply basins
- Montgomery and Associates (M&A, 2019, 2023a, 2023b, 2023c); field investigations and groundwater modelling
- Flow Hydro Consulting (2023, 2024); brine sources from the salars

Company hydrogeological specialists provided support during water supply investigations and subsequent review of the interpreted supply sources including the expected recharge rates.

24.1 Introduction

Whilst the Project setting is unchanged, the Project water balance has been updated following recent revelations regarding the supply of both fresh and brine water for processing. The proposed fresh water borefields and abstraction estimates have been updated to reflect the investigations since 2021.

The 2021 Technical Report referred to a proposed brine bore field on the Salar de Arizaro located immediately to the east of the mine waste dump. This concept was changed following the drilling of several initial test wells on the salar. An ESIA annexure (FQM, October 2023) consequently described an array of 100 m long, 5 m wide and 5 m deep brine collection trenches in the salar surface.

It had been interpreted that there would be an unlimited volume of brine available for processing (i.e., abstracted from the Salar de Arizaro). Following a further period of field investigations and analyses it became evident that this interpretation required qualification. Whilst the brine water volume may be substantial, the transmissivity of the salar sediments is such that an apparent low rate of recharge into trenches may limit the volume of brine that can be practically abstracted for sustainable operations.

Field teams are currently concluding another brine exploration campaign that includes the construction and testing of new pumping and observation wells. Initial results from this most recent investigation have indicated that brine abstraction from similarly constructed pumping wells could be a viable alternative to the pumping of brine from excavated trenches.

The following commentary outlines the continuing water supply investigations, groundwater modelling and updated supply estimates for both fresh water and brine.

24.2 Water supply requirements

In the 2021 Technical Report water balance summaries, about 75% of the required plant water demand was to be as brine abstracted from the salar. The specification of the process plant flowsheet took this into account when designing the rougher flotation circuit to operate in brine, albeit at the expense of slightly reduced metal recovery.

The Project water demand now reflects that fresh water will be used for both rougher and cleaner flotation. A lesser volume of brine is then required, primarily for the repulp of tailings prior to deposition into the TSF.

With an apparent low rate of recharge into trenches, and subject to the completion of the current well drilling and pumping investigations, there may be a restricted volume of brine that can be practically abstracted for sustainable operations at the scale envisaged. That situation adds a further complication to the amount of compensating fresh water that can be supplied. Improved brine water abstraction is therefore a subject of continuing water supply investigations.

The Stage 1 production profile has now been set at a 40 Mtpa maximum throughput. This revised scenario allows the Project to proceed initially with an achievable fresh and brine water supply from near-Project sources. Subject to the continuing water supply investigations mentioned above, additional water for processing at up to 60 Mtpa may be required from additional sources.

The updated Project water balance details are presented in Table 24-1, Table 24-2 and Figure 24-1.

24.3 Water sources from fresh water basins

Groundwater in the Project area occurs in aquifers located in alluvial environments (i.e., gravel beds) located in the valleys and basins surrounding the Salar de Arizaro and Salar de Taca Taca. The source of the fresh groundwater is from infiltration recharge due to precipitation at higher altitudes.

Figure 24-2 shows the major fresh water catchment areas within the Siete Curvas basin, four of which, Valle de Arizaro, Socompa, Valle de Chaschas and Valle de las Burras appear to be the most prospective for nearby sustainable fresh water supply to the Project.

Piteau (2022) characterised the hydrogeological setting for the fresh water catchments, as follows:

- The hydrogeological system is associated with volcanic and plutonic bedrock, where groundwater flows primarily through fractured rock. There are two main bedrock systems, i.e.
  - fractured/weathered bedrock, composed mainly of granite and intensely fractured rhyolites, with flow through faults and lithological contacts
  - fresh bedrock, composed mainly of granite with smaller amounts of dacite, dolerite and rhyolite, and exhibiting low to very low permeability
- Groundwater flows from the fractured/weathered rock to fresh rock.
- Due to low permeability, water is stored and flows through secondary structures such as faults, fractures and bedding planes.
- Groundwater flow direction is from northwest to southeast towards the Salar de Arizaro.

Table 24-1 Summary fresh water and brine demand (consumption)

Water Demand; Stage 1 Project	Average, 40 Mtpa				Peak, 40 Mtpa			
	ML/annum	kL/day	m^3/h	L/s	ML/annum	kL/day	m^3/h	L/s
Fresh water demand								
for processing	17,214.4	47,162.6	1,965.1	545.9	17,214.4	47,162.6	1,965.1	545.9
for the camp	75.6	207.0	8.6	2.4	189.8	520.0	21.7	6.0
for mining	5.3	14.5	0.6	0.2	15.9	43.5	1.8	0.5
for site services, rail load-out & construction	7.3	20.0	0.8	0.2	21.9	60.0	2.5	0.7
Subtotal	**17,302.5**	**47,404.1**	**1,975.2**	**548.7**	**17,441.9**	**47,786.1**	**1,991.1**	**553.1**
Brine demand								
for processing	4,124.6	11,300.3	470.8	130.8	4,124.6	11,300.3	470.8	130.8
for mining	1,089.6	2,985.1	124.4	34.6	2,179.1	5,970.2	248.8	69.1
for road maintenance & construction	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**5,243.4**	**14,365.4**	**598.6**	**166.3**	**6,376.8**	**17,470.6**	**727.9**	**202.2**
Water in ore processed	1,354.6	3,711.3	154.6	43.0	1,354.6	3,711.3	154.6	43.0
TOTAL	**23,900.5**	**65,480.9**	**2,728.4**	**757.9**	**25,173.3**	**68,968.0**	**2,873.7**	**798.2**
Processing summary								
fresh make-up	17,214.4	47,162.6	1,965.1	545.9	17,214.4	47,162.6	1,965.1	545.9
brine make-up	4,124.6	11,300.3	470.8	130.8	4,124.6	11,300.3	470.8	130.8
water in ore processed	1,354.6	3,711.3	154.6	43.0	1,354.6	3,711.3	154.6	43.0
Subtotal	**22,693.6**	**62,174.3**	**2,590.6**	**719.6**	**22,693.6**	**62,174.3**	**2,590.6**	**719.6**
Mining summary								
fresh	5.3	14.5	0.6	0.2	15.9	43.5	1.8	0.5
brine	1,089.6	2,985.1	124.4	34.6	2,179.1	5,970.2	248.8	69.1
Subtotal	**1,094.9**	**2,999.6**	**125.0**	**34.7**	**2,195.0**	**6,013.7**	**250.6**	**69.6**
Camp and other								
fresh	83.5	228.7	9.5	2.6	211.7	580.0	24.2	6.7
brine	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**112.7**	**308.7**	**12.9**	**3.6**	**284.7**	**780.0**	**32.5**	**9.0**
TOTAL	**23,901.2**	**65,482.6**	**2,728.4**	**757.9**	**25,173.3**	**68,968.0**	**2,873.7**	**798.2**

Water Demand; Stage 2 Project	Average, 60 Mtpa				Peak, 60 Mtpa			
	ML/annum	kL/day	m^3/h	L/s	ML/annum	kL/day	m^3/h	L/s
Fresh water demand								
for processing	25,821.5	70,743.9	2,947.7	818.8	25,821.5	70,743.9	2,947.7	818.8
for the camp	76.2	208.7	8.7	2.4	189.8	520.0	21.7	6.0
for mining	5.3	14.5	0.6	0.2	15.9	43.5	1.8	0.5
for site services, rail load-out & construction	7.3	20.0	0.8	0.2	21.9	60.0	2.5	0.7
Subtotal	**25,910.3**	**70,987.1**	**2,957.8**	**821.6**	**26,049.1**	**71,367.4**	**2,973.6**	**826.0**
Brine demand								
for processing	6,186.9	16,950.5	706.3	196.2	6,186.9	16,950.5	706.3	196.2
for mining	1,196.1	3,277.0	136.5	37.9	2,392.2	6,554.0	273.1	75.9
for road maintenance & construction	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**7,412.2**	**20,307.5**	**846.1**	**235.0**	**8,652.1**	**23,704.5**	**987.7**	**274.4**
Water in ore processed	2,032.0	5,567.0	232.0	64.4	2,032.0	5,567.0	232.0	64.4
TOTAL	**35,354.5**	**96,861.6**	**4,035.9**	**1,121.1**	**36,733.2**	**100,638.8**	**4,193.3**	**1,164.8**
Processing summary								
fresh make-up	25,821.5	70,743.9	2,947.7	818.8	25,821.5	70,743.9	2,947.7	818.8
brine make-up	6,186.9	16,950.5	706.3	196.2	6,186.9	16,950.5	706.3	196.2
water in ore processed	2,032.0	5,567.0	232.0	64.4	2,032.0	5,567.0	232.0	64.4
Subtotal	**34,040.4**	**93,261.3**	**3,885.9**	**1,079.4**	**34,040.4**	**93,261.3**	**3,885.9**	**1,079.4**
Mining summary								
fresh	5.3	14.5	0.6	0.2	21.9	60.0	2.5	0.7
brine	1,196.1	3,277.0	136.5	37.9	2,392.2	6,554.0	273.1	75.9
Subtotal	**1,201.4**	**3,291.5**	**137.1**	**38.1**	**2,414.1**	**6,614.0**	**275.6**	**76.6**
Camp and other								
fresh	83.5	228.7	9.5	2.6	211.7	580.0	24.2	6.7
brine	29.2	80.0	3.3	0.9	73.0	200.0	8.3	2.3
Subtotal	**112.7**	**308.7**	**12.9**	**3.6**	**284.7**	**780.0**	**32.5**	**9.0**
TOTAL	**35,354.5**	**96,861.6**	**4,035.9**	**1,121.1**	**36,739.2**	**100,655.3**	**4,194.0**	**1,165.0**

Table 24-2 Summary overall Project water balance

Overall Water Balance 40 Mtpa processing		Average m3/h	Peak m3/h	Overall Water Balance 60 Mtpa processing		Average m3/h	Peak m3/h
Fresh water In				**Fresh water In**			
	regional borefields	2,776.0	3,190.0		regional borefields	3,712.0	4,126.0
	Pit dewatering/pit slope drainage	81.0	108.0		pit slope drains/bores	81.0	108.0
	Subtotal	**2,857.0**	**3,298.0**		**Subtotal**	**3,793.0**	**4,234.0**
To Operations				**To Operations**			
	process plant	1,965.1	1,965.1		process plant	2,947.7	2,947.7
	water in ore processed	154.6	154.6		water in ore processed	232.0	232.0
	mine operations	0.6	1.8		mine operations	0.60	1.8
	camp. etc	9.5	24.2		camp. etc	9.53	24.2
	Subtotal	**2,129.8**	**2,145.7**		**Subtotal**	**3,189.8**	**3,205.6**
Water consumption				**Water consumption**			
	process plant				process plant		
	in concentrates	8.0	8.0		in concentrates	12.0	12.0
	in tailings	2,111.7	2,111.7		in tailings	3,167.6	3,167.6
	mine operations				mine operations		
	to sewage treatment	8.7	21.7		to sewage treatment	8.70	21.7
	evaporation	1.4	4.3		evaporation	1.44	4.3
	Subtotal	**2,129.9**	**2,145.7**		**Subtotal**	**3,189.8**	**3,205.6**
Fresh water supply surplus/defecit		**881.8**	**1,306.9**	**Fresh water supply surplus/defecit**		**835.2**	**1,260.4**
Brine water In				**Brine water In**			
	salar trenches and bores	828.0	1,008.0		salar trenches and bores	828.0	1,008.0
	tails seepage	180.0	180.0		tails seepage	180.0	180.0
	TSF return	0.0	0.0		TSF return	0.0	0.0
	Subtotal	**1,008.0**	**1,188.0**		**Subtotal**	**1,008.0**	**1,188.0**
To Operations				**To Operations**			
	process plant	470.8	470.8		process plant	706.3	706.3
	mine operations	124.4	248.8		mine operations	136.5	273.1
	road maint. etc	3.3	8.3		road maint. etc	3.3	8.3
	Subtotal	**598.6**	**727.9**		**Subtotal**	**846.1**	**987.7**
Water consumption				**Water consumption**			
	process plant				process plant		
	evaporation at TSF	470.8	470.8		evaporation at TSF	706.3	706.3
	mine operations				mine operations		
	to the ground	14.9	29.9		to the ground	16.0	31.9
	evaporation	112.8	227.2		evaporation	123.9	249.6
	Subtotal	**598.6**	**727.9**		**Subtotal**	**846.1**	**987.7**
Brine water supply surplus/defecit		**409.4**	**460.1**	**Brine water supply surplus/defecit**		**161.9**	**200.3**
Water to TSF				**Water to TSF**			
	Brine				**Brine**		
	Brine consumption in plant	-470.8	-470.8		Brine consumption in plant	-706.3	-706.3
	Brine to TSF (evaporated)	470.8	470.8		Brine to TSF (evaporated)	706.3	706.3
	Fresh				**Fresh**		
	Fresh consumption in plant	-2,119.7	-2,119.7		Fresh consumption in plant	-3,179.6	-3,179.6
	Fresh to concentrates	8.0	8.0		Fresh to concentrates	12.0	12.0
	Fresh to TSF	**2,111.7**	**2,111.7**		**Fresh to TSF**	**3,167.6**	**3,167.6**
	Balance	**0.0**	**0.0**		**Balance**	**0.0**	**0.0**
TSF water balance				**TSF water balance**			
	Total to TSF	2,582.6	2,582.6		Total to TSF	3,873.9	3,873.9
	TSF decant	0.0	0.0		TSF decant	0.0	0.0
	Total evaporation	-470.8	-470.8		Total evaporation	-706.3	-706.3
	Total in settled solids	-2,111.7	-2,111.7		Total in settled solids	-3,167.6	-3,167.6
	Balance	**0.0**	**0.0**		**Balance**	**0.0**	**0.0**

Figure 24-1 Overall Project water balance, 40 Mtpa processing

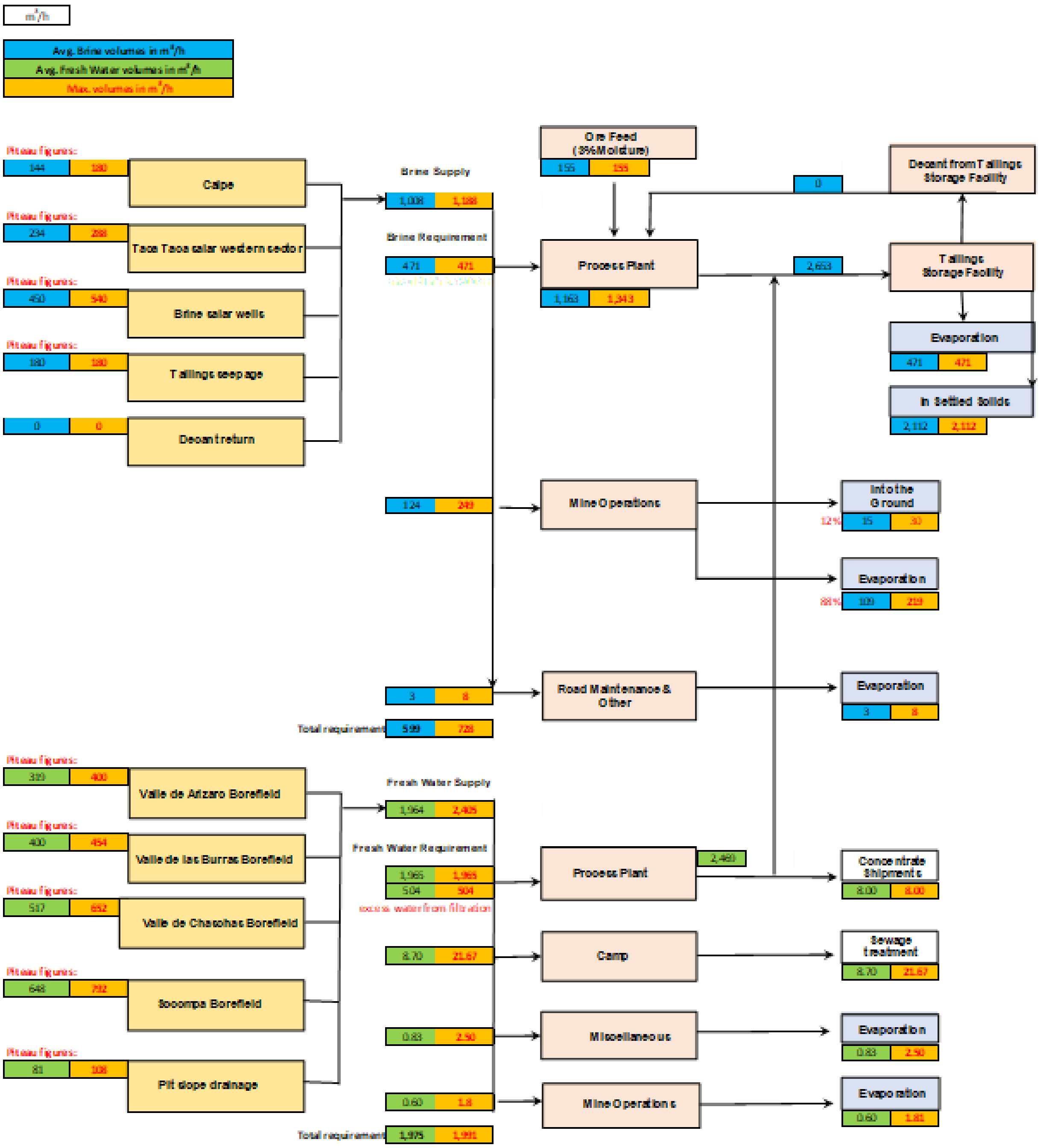
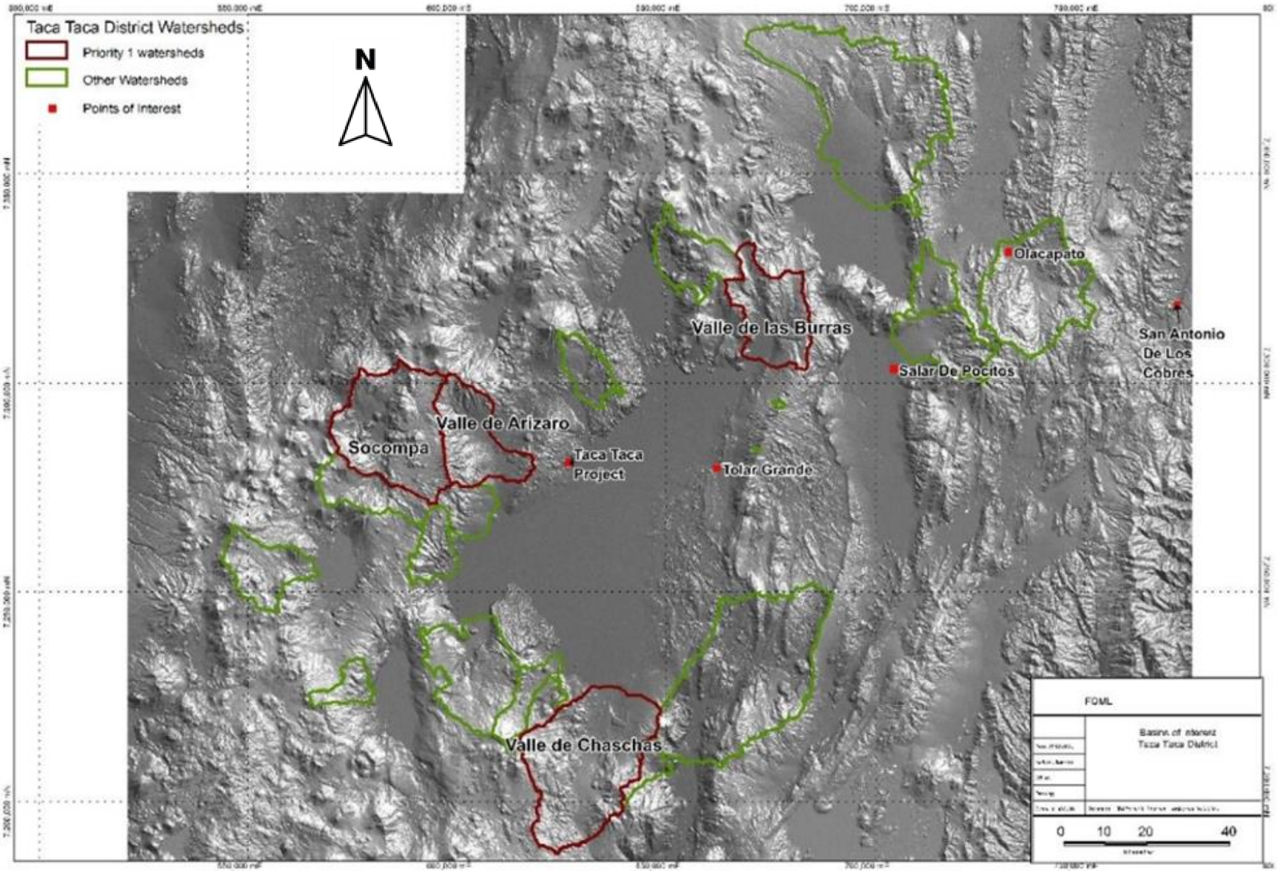


Figure 24-2 Regional catchment areas near to the Project



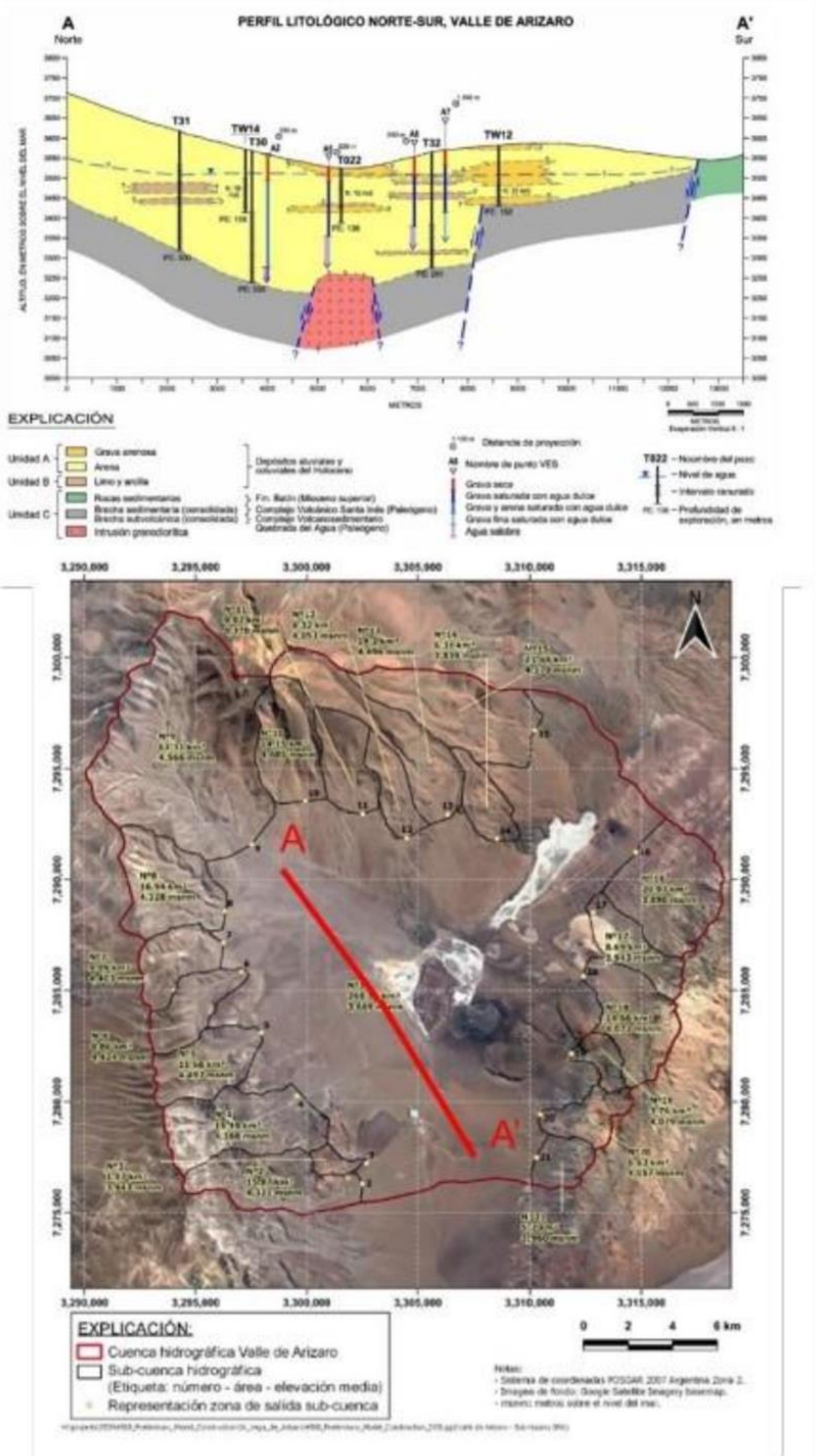
24.3.1 Valle de Arizaro

The Valle de Arizaro gravel basin is 18 km from the Project site and is the nearest potential source of large volumes of fresh water (Figure 24-3).

The revised characteristics of the basin were described by Piteau (2025) drawing on field investigations (M&A, 2019), along with the analysis of basin specific pumping tests and numerical groundwater flow modelling (M&A, 2023a) and can be summarised as follows:

1. The Valle de Arizaro basin covers an area of 488 km^2 and receives an average annual precipitation of 66 mm, which includes 59 mm of water and 7 mm of snow. The basin's recharge rate ranges from 60 L/s to 230 L/s, with an average of 125 L/s, or approximately 0.19 L/s/km^2. The primary hydrogeological unit of interest consists of colluvial and alluvial deposits, covering an area of 87 km^2 with a saturated depth of between 100 m and 275 m.
2. Assuming a conservative specific yield for the basin of 10%, the total volume of water in the basin was estimated by M&A to be 686 Mm^3.
3. The potential water supply from the basin was estimated to be between 108 L/s and 153 L/s,. There is a risk associated with groundwater reduction in environmentally sensitive sectors. In order to mitigate these risks an artificial irrigation requirement of 17 L/s was assumed for current planning.
4. This results in a Piteau nominated extraction rate of 66 - 111 L/s from the Valle de Arizaro basin.

Figure 24-3 Valle de Arizaro basin (source Montgomery and Associates (M&A), 2023a)



24.3.2 Socompa

The Socompa gravel basin is 50 km from the Project site and is fed by a catchment with an aerial extent of 686 km^2 (Figure 24-4). The basin receives an average annual precipitation of 92 mm, which includes 67 mm of rain and 25 mm of snow. The basin's recharge was evaluated in previous studies with a range of 183 L/s to 215 L/s, with an average of 200 L/s, or approximately 0.29 L/s/km^2.

The revised characteristics of this basin were documented by Piteau (2025) referencing the analysis of pumping tests and numerical groundwater flow modelling by M&A (M&A, 2023b), and can be summarised as follows:

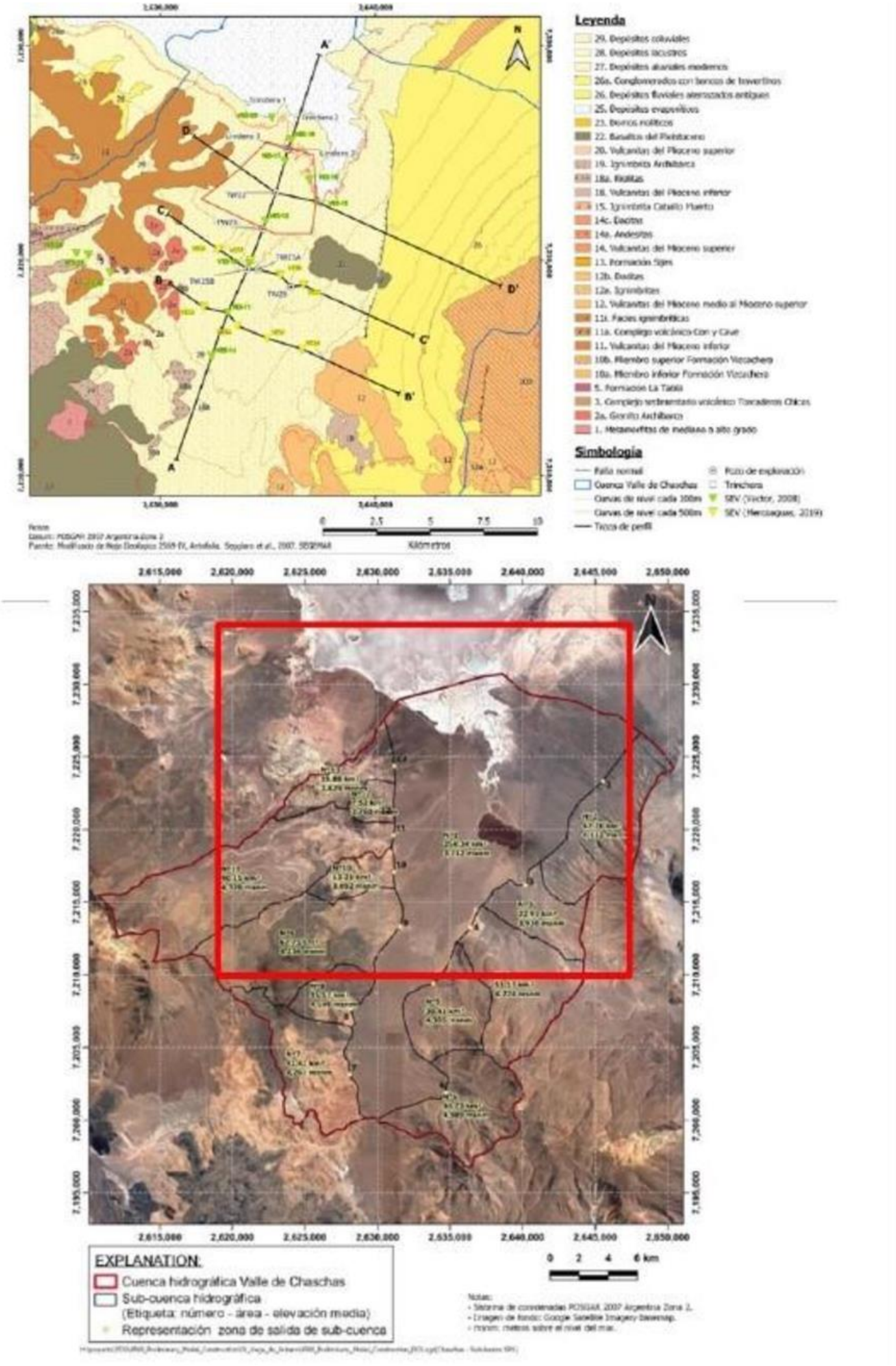
1. The primary hydrogeological unit of interest consists of colluvial and alluvial deposits, covering an area of 190 km^2 with a saturated depth of between 100 m and 200 m.
2. The Piteau review (2025) defined a total area of 190 km² and a recharge range of between 10% and 20%, consistent with previous studies. However, the recharge rate is considered by Piteau as likely to be at the high end of the estimated range due to the higher elevation and evidence of extensive braided channels. These can provide a significant amount of concentrated recharge during occasional high rainfall events. On balance, a reasonable estimate for the range of recharge was estimated by Piteau as between 250 L/s to 330 L/s.
3. The specific yield of the Socompa basin deposits was estimated as at least 10%, and likely higher, depending on the degree of cementation of the alluvial sediments. The total groundwater volume was estimated to be about 700 Mm^3.
4. There are existing water rights for third parties, amounting to 22 L/s and artificial irrigation is required to sustain existing vegetation and fauna at 5 L/s. The potential water supply from the basin was estimated to be between 223 L/s and 303 L/s.

Figure 24-4 Socompa basin (source: M&A, 2023b)

24.3.3 Valle de Chaschas

Valle de Chaschas is located about 60 km southeast of the Project site and is adjacent to the Lindero gold project (Fortuna Silver Mines Inc. (Fortuna)) (Figure 24-5). The Chaschas basin, covering an area of 703 km², experiences an average annual precipitation of 64 mm (rain and snow).

Figure 24-5 Valle de Chaschas basin (source: M&A, 2023c)



The revised characteristics of this basin were documented by Piteau (2025) referencing the numerical groundwater flow modelling by M&A (M&A, 2023c), and can be summarised as follows:

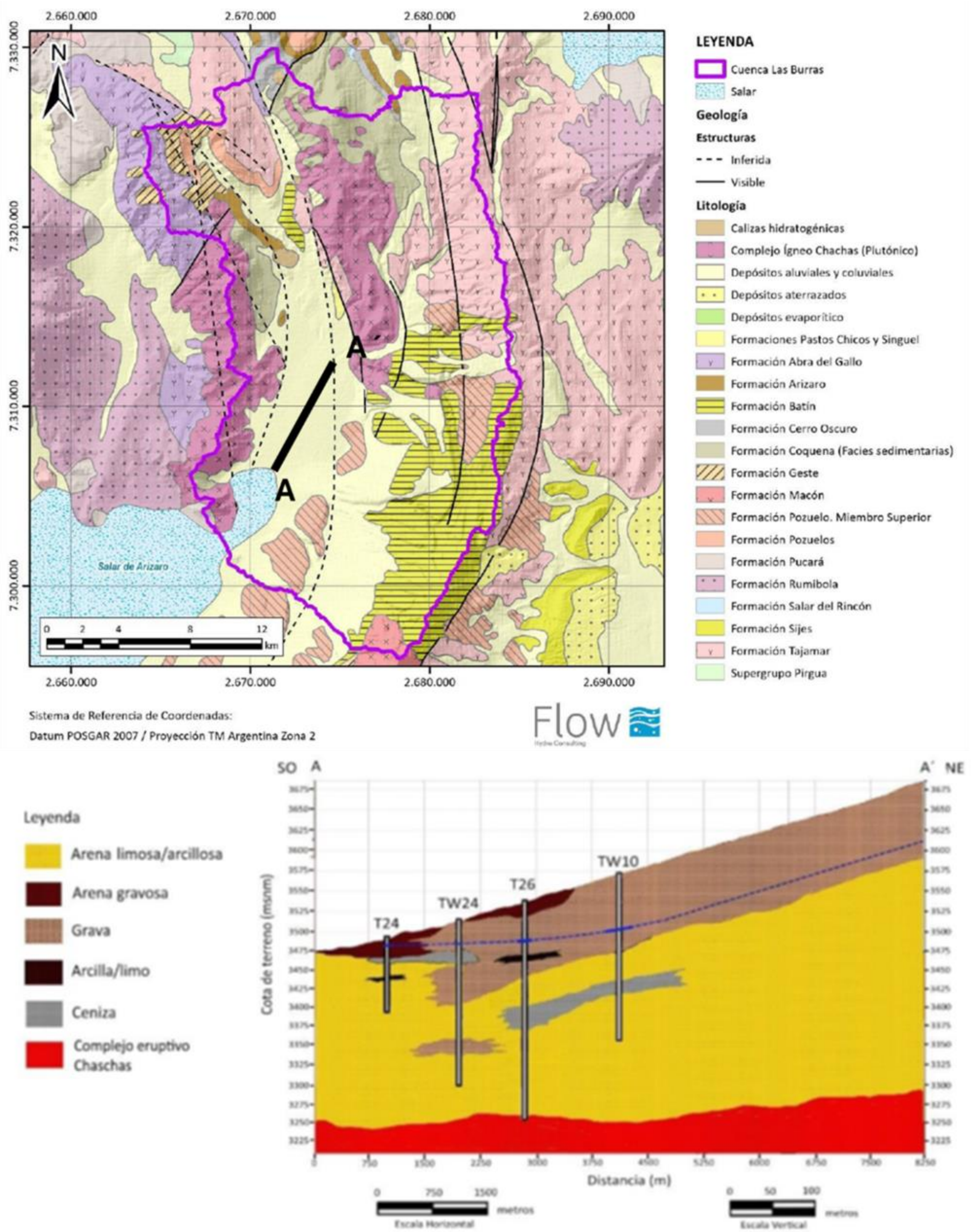
1. The primary hydrogeological units of interest within the basin consist of colluvial and alluvial deposits, spanning an area of 120 km², with a saturated depth of between 75 m and 100 m. The M&A (2023c) review indicates that the basin area covered by these deposits is of the order of 415 km$^{2.}$
2. The existence of some long-braided channels is indicative of occasional but extreme runoff events. It indicates that the effective recharge occurs periodically (potentially every three to five years) after major rainfall episodes in which the rainfall intensity exceeds values in the order of 5 mm/hr to 10 mm/hr. Under these conditions, the recharge rate for the alluvium and colluvium deposits is expected to be in the order of 20%, and potentially 10% to 15% for the weathered volcanic deposits.
3. The basin's recharge rate was estimated to be in the range of 90 L/s to 360 L/s, with an average of 280 L/s, which is approximately 0.36 L/s/km². The effective water supply from the basin was estimated to be between 225 L/s and 300 L/s,. The total volume of the aquifer is estimated to be 810 Mm³.
4. In addition, there are risks associated with groundwater reduction in certain environmentally sensitive sectors identified in the Valle de Chaschas. The predictive simulations provided by M&A (2023c) indicated the need to maintain an artificial irrigation requirement of 15 L/s. However, this measure may be excessive if the configuration of pumping wells for the Taca Taca Project is implemented from wells located south of the Chaschas basin, as the potential for impacts would likely be less.
5. Overall, this results in a nominated extraction rate of 182 to 257 L/s from the Valle de Chachas basin.

24.3.4 Valle de las Burras

The Valle de las Burras gravel basin is 50 km north-east of the Project site (Figure 24-6). The revised characteristics of this basin were documented by Piteau (2025) referencing numerical groundwater flow modelling by others (Flow Hydro Consulting, September and November 2023), and can be summarised as follows:

1. The initial evaluation on the Valle de las Burras indicated that the basin spans an area of 456 km² and receives an average annual precipitation of 85 mm, which includes 66 mm of water and 19 mm of snow. The basin's recharge rate ranges from 123 L/s to 200 L/s, with an average of 162 L/s, or approximately 0.36 L/s/km². The primary hydrogeological unit consists of colluvial and alluvial deposits in the central valley, covering an area of 45 km².
2. From the geology information (maps, sections) and previous drilling results, the total basin area for Las Burras is equivalent to 157 km². Again, the occurrence of long-braided channels in the central area is indicative of extreme runoff events with contributions from the surrounding high ground, suggesting that the recharge rate may be as high as 20%.
3. From the re-interpretation of data, the Piteau (2025) study indicates that the basin may produce a sustained flow rate of between 160 L/s and 190 L/s. The basin has existing water rights for third parties. Additionally, a rate of about 16.5 L/s has been assumed for environmental mitigation.
4. This results in a nominated extraction rate of 143 to 173 L/s from the Valle de las Burras basin.

Figure 24-6 Valle de las Burras basin (source: Flow Hydro Consulting, 2023)



24.3.5 Other basins

Although further afield, there are several other possible fresh water sources, located to the east of Valle de las Burras. Considering new information documented by Piteau (2025), the list of other sources has been reduced from that mentioned in the 2021 Technical Report.

24.4 Brine sources from the salars

With the groundwater flowing into the salars from the surrounding land mass, brine is commonly found in the clastic (lacustrine) sediments and in the overlying evaporitic deposits (M&A, November 2018). The hydraulic gradient in the transition zone from the surrounding fractured rock into the low lying Salar de Arizaro is relatively low (Piteau, 2022). Flow within that salar is mainly south westward and relatively limited due to the low hydraulic conductivity of evaporitic deposits (Piteau, 2022).

FlowHydro (2024) identified two sub-basins in relation to brine deposition. These basins, shown in Figure 24-7 are the Salar de Taca Taca sub-basin (358 km^2) and the Trincheras sub-basin (112 km^2). Both sub-basins are delineated on elevation and terrain geomorphology.

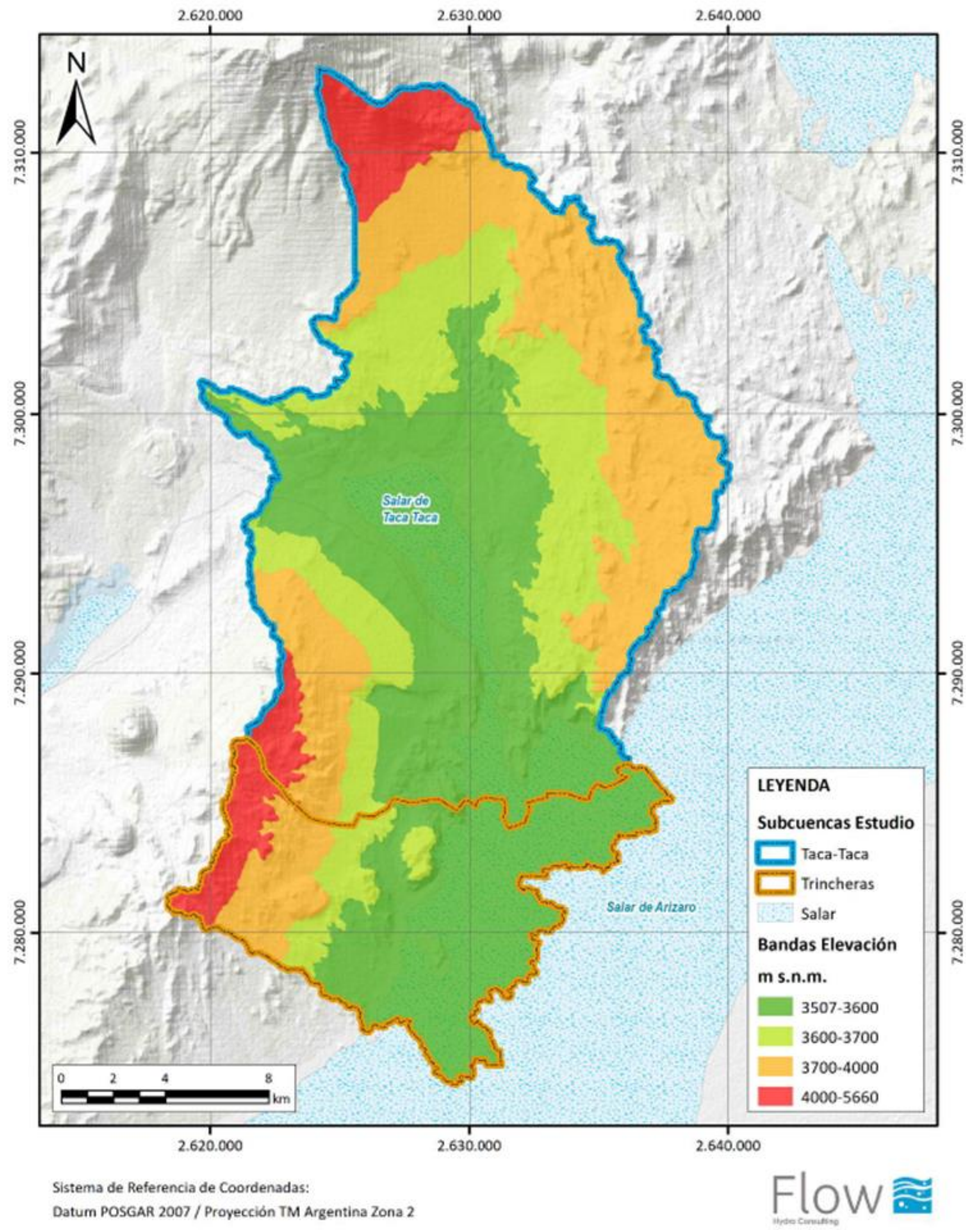
Evaporation significantly exceeds both precipitation and recharge, leading to a net loss in water storage in both sub-basins.

The 2021 Technical Report referred to a proposed brine borefield on the Salar de Arizaro located immediately to the east of the mine waste dump. This concept was subsequently changed following the drilling of several initial test wells on the salar. Instead, the ESIA water supply annexure (FQM, October 2023) described an array of 100 m long, 5 m wide and 5 m deep trenches in the salar surface:

- at Salar de Arizaro, around the perimeter of the waste dump
- at Salar de Taca Taca, within the perimeter of the TSF

Investigations and analyses were carried out in early 2024 to assess the rate of recharge of brine into trenches that could be excavated in the surface of the Salar de Arizaro. This work suggested a modest 36 m^3/hr sustainable extraction rate for a single 100 m long, 2 m wide and 4 m deep trench. Hence, this then posed a question over the transmissivity of the salar sediments and the capability of excavated trenches to supply enough brine water to meet the processing demand as originally envisaged.

Figure 24-7 Sub-basins related to brine deposition (source: FlowHydro 2024)



Water supply investigations are continuing (Item 24.7.2), especially regarding the brine supply. Field teams are currently concluding a successful brine exploration campaign that includes the construction and testing of new pumping and observation wells. Encouraging results have fostered a shift in focus to possible brine supply from pumping wells rather than from excavated trenches at the surface of the salar, as previously considered.

24.5 Other water supply sources

24.5.1 Decant return

According to a tailings storage facility engineering cost study performed by SRK Consulting (2025), water reclamation from the TSF could be between 55 L/s and 139 L/s depending on the season. Reclamation rates would correlate with tailings beach volumes and with evaporation rates. Piteau (2025), however, estimate that decant return could supply brine at between 200 L/s and 500 L/s.

According to Piteau, brine supplied from trenches, wells and TSF seepage would be at an average of 1,008 m^3/h prior to any decant return. Assuming average decant return would increase the brine supply to 2,268 m^3/h.

Regardless, there is no decant return assumed in the current processing plant water balance (refer to Item 17).

24.5.2 Seepage from the TSF

From the Piteau report (2025), the added pumping of TSF seepage could supply about 50 L/s from up to 20 shallow wells drilled to depths up to 50 m within the footprint of the TSF.

24.5.3 Mine inflow

In terms of water inflow to the open pit mine, a full evaluation of the effects of faulting and rock mass alteration on rock mass properties has not yet been completed. However, localised packer testing at the Project site indicates that permeability across fractures often results in higher hydraulic conductivity values, whilst the recorded higher hydraulic heads suggest that groundwater movement within the mining environment will likely be dominated by discrete fracture zones rather than through the intact rock matrix (Ausenco, 2011).

The Ausenco analyses indicated:

- natural inflow arising from excavation of the pit would range from 16 L/s to 33 L/s
- active pit slope depressurisation by means of pit bores and horizontal drains could yield 28 L/s to 54 L/s
- a cone of depression arising from pit slope depressurisation will be steep in and around the pit
- groundwater drawdown as the depth of the mine increases will have limited impact on the adjacent salar

More recent work by Piteau (2022), however, provides an observation that fresh rock has low to very low hydraulic conductivity, with structural features potentially compartmentalising the groundwater flow. This implies the potential for development of excess pore water pressures within the open pit walls.

24.5.4 Desalination

Schlumberger (2013) completed a scoping study level assessment and design of a desalination process to provide fresh water make-up for the Project.

The PEA study (Ausenco, 2013) stated that fresh water of not greater than 1,500 mg/L TDS (total dissolved solids) must be used for cleaner flotation, cooling, reagent mixing and concentrate washing. On the other hand, brine at up to 300,000 mg/L TDS could be used for grinding and rougher flotation.

Information in the PEA study indicates that a desalination plant would require a brine TDS of not greater than 50,000 mg/L. For reference, water samples taken from the Salar de Arizaro (Ausenco, 2012) had a TDS of 255,500 mg/L (Table 24-3).

Table 24-3 Summary of brine water analyses

Parameter	Unit	Salar de Arizaro	Plumas Verde	Pit Water
pH	pH	7.1	8.3	7.05
Conductivity	uS/cm	>200 000	17 420	241 700
TDS	mg/l	255 500	10 700	317 596
Alkalinity	mg/l	49	129	-
Bicarbonate	mg/l	59	158	-
Calcium	mg/l	2 510	169	1558
Magnesium	mg/l	1 350	82	706
Chloride	mg/l	153 000	6 120	184 600
Sulfate	mg/l	3 900	310	
Nitrate	mg/l	600	30	
Sodium	mg/l	84 400	4 120	70 277
Potassium	mg/l	3 030	77	

Bolstering the fresh water volume by means of a desalination plant and treatment of water pumped from the salar may be neither possible nor required. In acknowledging this, Schlumberger devised a “recipe” for blending and mixing of brines to service the water demand as then envisaged. However, the plant water demand and possible borefield locations have since been revised and, subject to further analysis of water quality and sustainability from these locations during the Project engineering phase, fresh and brine abstraction quantities at suitable TDS levels may be possible.

24.6 Recharge to catchment areas

The Piteau (2025) study included an update to the estimated recharge rate into the individual basins. Piteau’s findings were as follows:

- most recharge occurs during intense precipitation and run-off events (> 10 mm/hr)
- the frequency may be once every three to five years
- rapid infiltration occurs along braided, coarse alluvial channels
- secondary recharge occurs as infiltration downslope from areas of accumulated snow
- the recharge co-efficient for the basin alluvial/colluvial deposits is 20% of the average annual precipitation
- the recharge co-efficient for the weathered rock and volcanic deposits is 12.5% of the average annual precipitation

FlowHydro (2024) extended the analysis of recharge rates, refining the estimate by analysing meteorological (i.e., precipitation and evaporation records) and geomorphological information. The annual recharge rate into the Taca Taca sub-basin was estimated as 4.3 mm/year. The corresponding rate for the Trincheras sub-basin was estimated as 9.5 mm/year. These rates were specified as 8% and 18% of annual precipitation, respectively.

The annual brine water storage deficits were estimated to be -1,310 L/s for the Taca Taca sub-basin, and 355 L/s for the Trincheras sub-basin.

24.6.1 Company benchmarking of recharge

Although recharge rates in the 1% to 5% range have been documented in some arid environments globally (Scanlon et al, 2006), most of these study sites are associated with lowland deserts of Australia, the desert southwest of the USA, and similar regions.

Recent work on recharge processes in the Andes highlights the importance of mountain front and alluvial fan recharge processes which increase recharge in high mountain areas relative to other arid environments (Houston, 2002; Houston, 2009). In the Argentine Puna region, and the local catchments considered for this Technical Report, mountain front and alluvial fan recharge processes are expected to dominate – as they do in the Atacama region of Chile – making the low-end estimate of 5% implausible.

As noted in the global compilation by Scanlon et al:

"Inter-annual climate variability related to El Niño Southern Oscillation (ENSO) results in up to three times higher recharge in regions within the SW United States …enhanced recharge related to ENSO is also documented in Argentina."

This highlights the importance of episodic recharge in addition to baseline annual recharge.

Recharge also increases with altitude. Houston (2009) used models and chloride mass balance techniques to assess recharge for the high altitude for the Linzor Basin of the Atacama, which is 50 km west of Jujuy near the Chile-Bolivia border. Houston demonstrated that while recharge rates were ~15% of precipitation at 4,100 m, they were much higher and more variable at elevations of 4,500m (35% to 46% of precipitation). The elevation of recharge areas for the Project's local catchments range from ~3,800 m to more than 5,500 m.

In summary, the long-range average annual recharge for the local catchments surrounding Taca Taca is likely to be substantially higher than the low-end consultant estimates. Recharge may be different in each catchment, dependent on relative contributions from each local recharge process (alluvial fan vs. mountain front, etc.)

Furthermore, the Company is aware of a nearby project to Taca Taca where 14% to 19% of total precipitation is considered to be recharged, based on modelling that considers multiple catchment recharge processes and is supported by isotopic age dating of groundwater.

Based on multiple studies in similar environments, The Company expects that reasonable long-term recharge ranges for local catchments, as a percentage of total precipitation, are in the order of 15% to 25%. Actual recharge will be verified following ongoing investigations.

24.7 Water supply investigations

An initial phase of water supply investigations was completed by Lumina in 2011 and 2012. Following acquisition of the Project by FQM, two more investigation phases were carried out in 2018, and then in 2019 to 2020. The Phase III water supply investigations described in the 2021 Technical Report were suspended before the planned programme could be completed. Table 24-4 lists the Phase III and Phase IV investigation holes as eventually completed.

Table 24-4 Summary of Phase III and IV water supply investigations

Hole_ID	Location	Construction	Type	Hole diameter (inches)	Hole_depth (m)	X	Y	Z	Material (PVC/Fe)	Static WL (m bgl)
TW12	Valle de Arizaro	Phase III	Existing bore	12	147	7,283,240	607,108	3,587	PVC	77
TW13	Valle de Arizaro	Phase III	Existing bore	14	140	7,285,124	604,920	3,623	PVC	104
TW14	Valle de Arizaro	Phase III	Existing bore	12	140	7,287,627	604,877	3,585	PVC	70
TW24	Las Burras	Phase III	Existing bore	12	205	7,307,030	671,165	3,517	Steel	31
TW27A	Las Burras	Phase III	Existing bore	12	210	7,307,429	672,735	3,549	Steel	59
TW26	Las Burras	Phase III	Existing bore	12	131	7,306,265	672,800	3,517	Steel	29
TW22	Chaschas	Phase III	Existing bore	12	180	7,222,305	635,282	3,531	PVC	26
TW23	Chaschas	Phase III	Existing bore	12	210	7,220,607	634,683	3,569	PVC	54
TW25B	Chaschas	Phase III	Existing bore	12	197	7,218,725	633,979	3,603	Steel	81
TW20	Socompa	Phase III	Existing bore	12	200	7,289,557	581,190	3,592	PVC	22
TW19	Socompa	Phase III	Existing bore	12	200	7,285,158	581,331	3,597	PVC	26
TW21	Socompa	Phase III	Existing bore	12	200	7,291,681	578,218	3,606	PVC	37
pAPW-VIB-02	Las Burras	Phase IV	Proposed bores	12	295	7,308,747	673,875	3,660	Steel	150
pAPW-VdA-05	Valle de Arizaro	Phase IV	Proposed bores	12	295	7,289,379	604,200	3,625	Steel	110
pAPW-Soc-04	Socompa	Phase IV	Proposed bores	12	250	7,291,494	578,902	3,600	Steel	35
pAPW-Cha-02	Chaschas	Phase IV	Proposed bores	12	250	7,220,015	633,087	3,600	Steel	75
pAPW-Cha-04	Chaschas	Phase IV	Proposed bores	12	295	7,219,143	635,743	3,600	Steel	90
pAPW-Cha-05	Chaschas	Phase IV	Proposed bores	12	295	7,215,598	632,766	3,650	Steel	130
pT36A	Salar de Taca Taca	Phase IV	Additional	8	150	7,295,742	623,607	3,540	Steel	50
pT37A	Salar de Taca Taca	Phase IV	Additional	8	200	7,293,199	626,645	3,540	Steel	50

24.7.1 Investigations since 2020

Valle de Arizaro

M&A (2023a) carried out 72 hour constant rate pumping test campaigns in the Valle de Arizaro basin, over several years. These tests demonstrated a good extractability from the existing wells located in the sedimentary unit, such as T22 (18 L/s), TW12 (25 L/s), TW13 (26 L/s), and TW14 (30 L/s), with the depths of these pumping wells ranging from 125 m to 230 m. Maximum drawdown in tested wells was 20 m after 72 hours.

The derived hydraulic conductivity (K) of these deposits varied from 1 m/day to 100 m/day, with tests showing values between 10 m/day and 22 m/day.

The water quality in the basin was characterised by an electrical conductivity (EC) ranging from 244 µS/cm to 3,700 µS/cm and total dissolved solids (TDS) of between 159 mg/l and 2,130 mg/l, indicating a range from fresh to brackish water quality.

Socompa

Pumping tests carried out by M&A (2023b) in the Socompa basin included the Socompa well (24 L/s), Quebrada del Agua (22 L/s), TW19 (53 L/s), and TW20 (52 L/s), with the depths of these pumping wells ranging from 208 m to 210 m. In the case of the Socompa and Quebrada del Agua wells, the pumping tests had a duration of 72 hours showing drawdown of 1.6 m and 1.1 m and specific flow rates of 14.9 L/s/m and 20.4 L/s/m, respectively. These results demonstrated a good extractability from the sedimentary unit. The hydraulic conductivity (K) resulting from the recovery tests for these two wells varied between 49 m/day to 79 m/day.

In the case of the TW19 and TW20 wells, the pumping test lasted 72 hours with a drawdown of 4.3 m and 5.0 m and specific flow rates of 12.2 L/s/m and 10.5 L/s/m, respectively. The hydraulic conductivity (K) resulting from the recovery tests for these two wells varies between 32 m/day to 79 m/day and is very similar to the results of the Socompa and Quebrada del Agua pumping wells.

Comparing the specific flow rates of wells TW19 and TW20, with pumping rates that were twice the flow rates tested in the previous wells, Socompa and Quebrada del Agua, it is observed that the reduction in specific flow rate is relatively low, which indicates a good production potential from the sedimentary deposits.

The water quality in the basin is characterised by an electrical conductivity (EC) ranging from 128 µS/cm to 666 µS/cm and total dissolved solids (TDS) of between 86 mg/L and 440 mg/L, indicating fresh water.

Valle de Chaschas

Pumping tests were carried out by M&A (2003c) on the Lindero 3 (20 L/s) well, TW22 (51 L/s), TW23 (47 L/s), and TW25B (3 L/s), with the depths of these pumping wells ranging from 137 m to 220 m. The pumping test carried out in the Lindero 3 well resulted in a drawdown of 16 m in around 10 hours and a specific flow rate that was reduced during the test from an initial 1.75 L/s/m to a final 1.25 L/s/m. However, since the duration of the test was less than 10 hours, these values are only considered for reference. In the case of the wells TW22, TW23 and TW25B, constant flow pumping tests of 26 hours (TW25B) to 72 hours (TW23) duration, and recovery tests, were carried out with each well being used as an observation well for the other during the respective tests. However, due to the distance of around 1.8 km between these wells, no decrease in water level was measured due to the pumping tests.

The hydraulic conductivity (K) of these deposits varied significantly from 0.5 m/day to 70 m/day. There are no hydraulic tests available that allow the storage coefficient (S) to be evaluated. Therefore, a value of S was estimated in the range of 0.1 to 0.2, with a specific yield (Sy) of around 15%.

The water quality in the basin is characterised by an electrical conductivity (EC) of approximately 400 µS/cm and total dissolved solids (TDS) between 267 mg/L and 289 mg/L, indicating fresh water.

Valle de las Burras

Pumping tests have been completed in bores TW-10 (26 L/s), TW-24 (46 L/s), TW-26 (52 L/s), and TW-27A (46 L/s), with the depths of these pumping wells ranging from 130 m to 210 m. Yields in all wells were good. TW-10 was tested in 2018 with a step test followed by a constant rate test undertaken over a duration of 24 hours. TW-27 is located at 15.6 m from the pumping well and has been used as an observation well for test interpretation. A drawdown of approximately 5 m was obtained in the observation well after 24 hours.

The hydraulic conductivity (K) of these deposits varies from 1 m/day to 50 m/day, with tests showing values between 2 m/day and 15 m/day. The available hydraulic test information has not been used to evaluate the storage coefficient (S). The specific yield (Sy) is estimated around 10%. Using those assumptions, the total volume of the aquifer is estimated to be 686 Mm^3.

The water quality in the basin is characterised by electrical conductivity (EC) values ranging from 1,530 to 18,600 µS/cm and total dissolved solids (TDS) between 800 mg/L and 12,600 mg/L, indicating brackish water close to the Vega Las Burras.

Salar de Arizaro and Salar de Taca Taca

The 2021 Technical Report documented the drilling and pump testing of an exploration well at TW11, located on the surface of the Salar de Arizaro, about 2 km offshore from the southern end of the Taca Taca deposit. This well produced 12.7 L/s of brine during a constant rate pumping test with a drawdown of 54.05 m over a 24 hour pumping period.

Since 2021, additional wells (TW15, TW16, TW17 and TW18) have been drilled in the same area to depths reaching 200 m to 250 m. An extraction rate of less than 2.5 L/s of brine was measured in three wells (TW15, TW17 and TW18), whilst 10.3 L/s was measured in one well (TW16).

24.7.2 2025 investigations

Field teams are currently concluding a successful brine exploration campaign that includes the construction of observation wells and the testing of new pumping wells within CASA´s tenements (Table 24-5, Figure 24-8). Considerably better results have been obtained relative to those of the previous 2019 to 2022 drilling campaign, now yielding a total of over 700 m³/hr of abstraction from eight new pumping wells located inside CASA´s tenements, close to the plant, into or near the salar surface.

Based on these pumping tests results, the same wells could theoretically yield over 1,000 m³/hr if pumped at higher rates (i.e., calculated from well specific capacity, drawdown, etc.).

The average lithium concentration in brine samples collected during testing has been ~170 mg/L, indicating that the brine can be considered a hydric (water supply) resource rather than a Mineral Resource.

These encouraging results have fostered a possible shift in focus to brine supply from pumping wells rather than from excavated trenches at the surface of the salar, as previously considered.

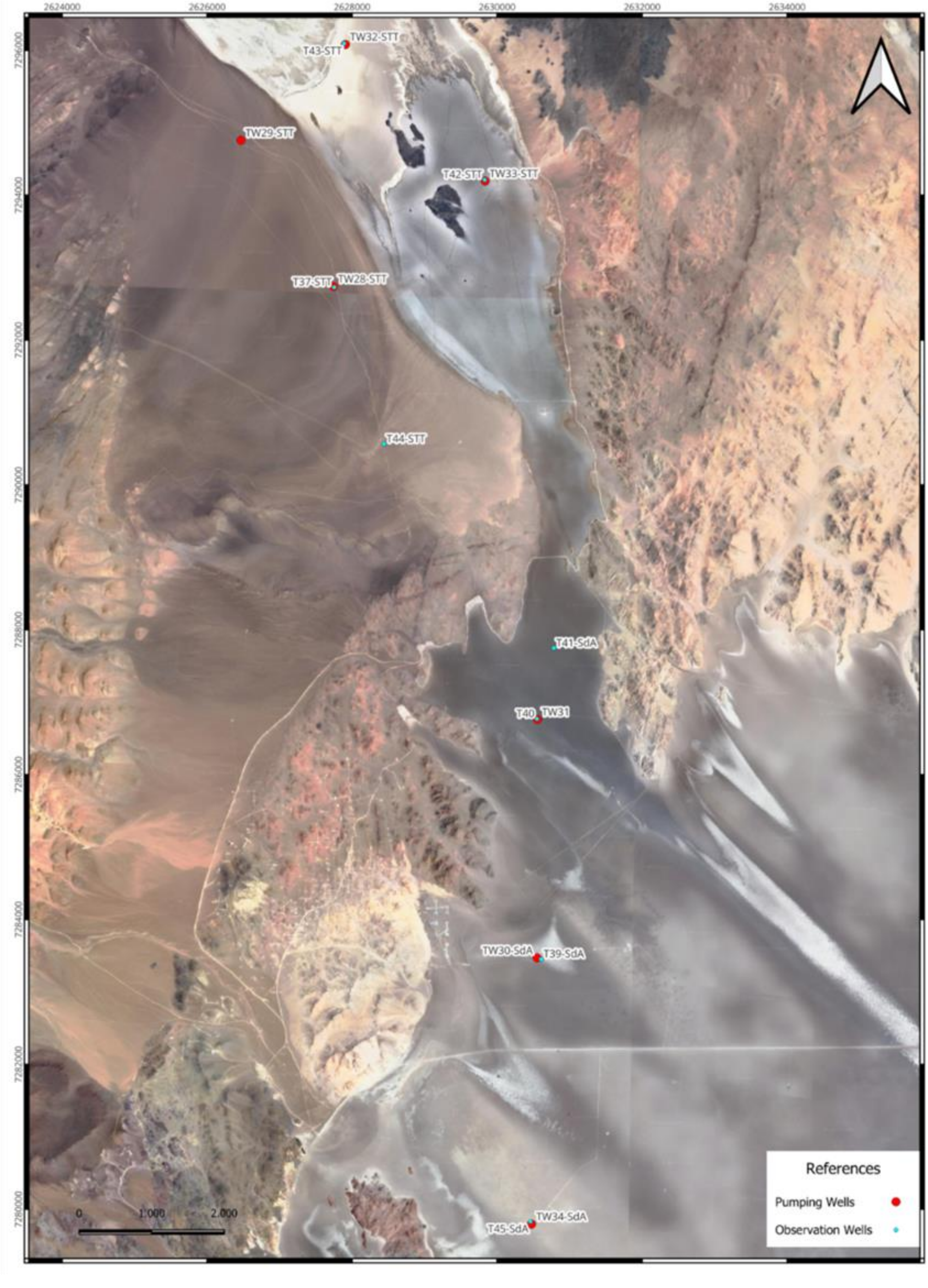
Nevertheless, permits have been received for the excavation and pump testing of exploratory trenches in the Salar de Arizaro. Abstraction and testing information from these trenches will enable an updated evaluation of the two alternative brine supply methodologies (i.e., pumping wells and trenches).

Piteau Associates is updating the conceptual and numerical groundwater flow model for the Project area to improve the understanding of brine transmissivity and storage in the salar, to assess the sustainability of the brine supply throughout the LOM timeframe and update the overall Project water balance.

Table 24-5 Brine exploration wells constructed in the 2025 campaign

Well ID	Well type	Depth	Casing dia.	Pumped rate		Calculated max.	
		(m)	(inches)	(L/s)	(m^3/h)	(L/s)	(m^3/h)
T37-STT	Observation	232	2.0				
T39-SdA	Observation	192	2.0				
T40-SdA	Observation	102	2.0				
T41-SdA	Observation	56	2.0				
T42-STT	Observation	222	2.0				
T43-STT	Observation	295	2.0				
T44-STT	Observation	90	4.0				
T45-Sda	Observation	231	2.0				
TW28-STT	Pumping	231	12.0	28.0	99.4		
TW29-STT	Pumping	223	12.0	24.0	84.7	25.0	90.0
TW30-SdA	Pumping	196	8.0	16.0	55.8	31.0	110.0
TW31-SdA	Pumping	89	8.0	24.0	85.7	50.0	180.0
TW32-STT	Pumping	280	12.0	26.0	94.0	40.0	144.0
TW33-STT	Pumping	207	12.0	57.0	205.8	75.0	270.0
TW34-SdA	Pumping	220	8.0	14.0	52.1	31.0	110.0
TW11	Pumping	293	10.0 - 6.0	12.0	43.1	31.0	110.0
			TOTAL	**201.0**	**720.6**	**283.0**	**1,014.0**

Figure 24-8 Brine exploration wells constructed during 2025



24.8 Water supply summary and outlook

24.8.1 Summary

Table 24-6 and Table 24-7 summarise the nominal fresh water and potential brine supply from the borefields, and from the salar sources, respectively.

In Table 24-6 the sustainable water resource assumes that the recharge allocation is excluded from the supply rates. An allowance has been included to enable irrigation of the local catchments and to minimise impacts to sensitive ecosystems.

Preliminary recharge assessments developed for the local catchments surrounding Taca Taca by M&A (and later referenced by Piteau, 2025) suggest 5% to 20% of annual precipitation is groundwater recharge. These estimates were based on high-level water balances, without developing a conceptual site model of recharge processes for each basin.

Table 24-6 Summary of fresh water supply sources

Water sources	Source: Piteau April 2025			Sustainable water resource	
	Potential gross water supply	Artificial irrigation	Third party allocation		
	L/s	L/s	L/s	L/s	m^3/hr
Main fresh water basins					
Valle de Arizaro	125 - 170	17	NA	108 - 153	389 - 551
Socompa	250 - 330	5	22	223 - 303	803 - 1,091
Chaschas	225 - 300	15	28	182 - 257	655 - 925
Las Burras	160 - 190	17	NA	143 - 173	515 - 623
Current ESIA Application	**760 - 990**	**54**	**50**	**656 - 886**	**2,362 - 3,190**
Possible fresh water basins					
Potential regional sources	260	-	-	260	936
Additional water source					
Pit dewatering/pit slope drainage	15 - 30	NA	NA	15 - 30	54 - 108
Total fresh water sources	**1,035 - 1,280**	**54**	**50**	**931 - 1,161**	**3,352 - 4,180**

Current fresh water sources covered by the ESIA application and pit dewatering should provide a net average supply of 2,856 m^3/hr. Additional regional sources would increase average supply to 3,792 m^3/hr. According to Piteau (2025), dewatering of the open pit excavation could supply an average of 15 to 30 L/s for the life of mine.

Fresh water borefield design considerations per basin are as follows:

- Valle de Arizaro: Five to ten wells with depths of 125 m to 250 m
- Socompa: Seven to twelve wells with depths of 200 m to 250 m
- Valle de Chaschas: Eight to fifteen wells with depths of 150 m to 225 m
- Valle de Las Burras: Eight to twelve wells with depths of 150 m to 225 m
- Caipe (5th priority): Four to six wells with depths of 150 m to 200 m

Table 24-7 summarises the projected supply of brine from several potential sources, amounting to over 1,000 m^3/hr on average. Subject to ongoing evaluations and modelling, the supply investigations in 2025 have indicated that brine could potentially be supplied at this rate solely from deep wells.

Table 24-7 Summary of brine supply sources (Piteau, 2025)

Source/location		Average water supply (sustainable)		Peak water supply (sustainable)	
		L/s	m^3/hr	L/s	m^3/hr
Brine supply					
	Caipe	40.0	144.0	50.0	180.0
	Western sector of Taca Taca salar	65.0	234.0	80.0	288.0
	Brine salar wells	125.0	450.0	150.0	540.0
	Tailings seepage	50.0	180.0	50.0	180.0
	Decant Return	0.0	0.0	0.0	0.0
	Subtotal	**280.0**	**1,008.0**	**330.0**	**1,188.0**

24.8.2 Outlook

The Piteau report (2025) summarised the hydrogeological investigation work completed through to the end of 2024 and updated the water resources outlook. Work undertaken since that report has focused on additional work in the Valle de Chaschas catchment to demonstrate water supply potential to a higher level of confidence.

In summary:

- Water exploration campaigns have been completed in all the local catchments, including surface and downhole geophysics, water supply well installation, single well pumping tests, and predictive groundwater modelling.
- Fresh water demand can be met through a combination of the four local catchments.
- Water supply concession requests for local catchments, totalling 1,021 L/s, have been submitted to provincial authorities. Concessions are approved based on demonstrable extraction rates and not recharge estimates
- Groundwater modelling of local catchments indicates potential for some impacts to sensitive ecosystems; mitigation, reduced pumping rates, or exploration of additional water supply sources may be required.
- Additional pumping and observation well installations were completed at Valle de Chaschas in 2025, in preparation for multi-well, long-term pumping tests planned for early 2026. These tests will help assess groundwater drawdown and impacts under a "near-operational" pumping scenario (combined pumping at ~200 L/sec).
- The 2025 brine exploration programme indicates brine extraction rates significantly higher than achieved from previous wells. The brine model is currently being updated by Piteau.

ITEM 25 INTERPRETATIONS AND CONCLUSIONS

25.1 Geology and Mineral Resource

A Mineral Resource estimate was completed from reasonable quality data using a 3D geology model and appropriate estimation methods. Drill grid spacing is around 150 m and supports adequate definition of the mineralisation domains used for block estimation. Estimation domains were based on an updated 3D geological model characterising weathering, alteration, lithology and dominant styles of copper mineralisation.

While limited sequential copper sample data has not enabled, detailed definition of copper species domains, results clearly identifies leached, mixed secondary and primary copper mineralised domains. The combined geology and copper domains have been used to guide the assignment of metal recoveries and concentrate grades. Although estimates validate well at the deposit scale, the current drill hole spacing does not always support locally accurate estimates at the scale of mining.

Estimate confidence is sufficient to support the reporting of Measured and Indicated Mineral Resources within a life-of-mine pit shell constrained by RPEE (reasonable prospects of economic extraction) assumptions, with 80% of the Mineral Resources classified as Indicated, and to support potential conversion to Proven and Probable Mineral Reserves subject to the application of appropriate modifying factors.

25.2 Mine planning and Mineral Reserve

A conventional approach has been adopted in the process of optimising a mine planning model derived from the Mineral Resource model, to be followed by detailed phase and ultimate open pit designs, production scheduling and Mineral Reserve estimation. The scale of primary mining equipment is considered to be suitable for the envisaged scale of operations and the size and extents of the proposed open pit. Since the 2021 Technical Report, the scale of the primary equipment has been reduced and a pre-stripping scaled fleet introduced for the Project development phase.

The pre-requisite pit optimisation inputs included mining, processing and G&A costs derived from a first principles estimation. This guiding optimisation was completed for an ultimate Stage 2 60 Mtpa production scenario, following which a retrospective sensitivity analysis was completed to assess the compounded impact of higher metal prices (subsequently prevailing) and the higher operating costs for a Stage 1 40 Mtpa operation. The mining costs have been estimated in detail from a comprehensive life of mine production schedule, and account for equipment life cycle costs, consumables and labour. Estimated load and haul costs relate to detailed pit design phases and haul route cycle times to and from a pit-top primary crusher, adjacent waste dump and long term stockpiles.

When considered with the life of mine production schedule and economic analysis (pre-tax cashflow model), the inclusion of a long-term stockpile reclaim inventory in the Mineral Reserve estimate is considered to be reasonable on the basis of positive annual cashflows beyond the initial Project development phase.

In the opinion of Michael Lawlor (QP), therefore, the Mineral Reserve estimate has considered appropriate modifying factors and reflects an achievable mining plan and Stage 1 40 Mtpa production schedule, at this stage of Project evaluation.

25.3 Metallurgy and process engineering

The testwork performed to date has been undertaken by internationally recognised laboratories to a high standard. Sample origins (drill hole number and depth) have been defined and any compositing of samples has been fully described.

Limited comminution data produced in the early testwork programmes provided conflicting information on ore hardness, and the work was repeated with ten samples derived from four metallurgical holes drilled in 2019. These samples represent the first five years of material to be mined during the initial operations. Results from this later work were used for SAG and ball mill sizing and for derivation of operating costs.

Optimum flotation conditions (grind size, reagent additions, pH, slurry density, etc.) were defined by Lumina, but the majority of this work was undertaken in Lima tap water. More recent work replicated this work and locked cycle flotation testwork on the ten samples (using brine from site in the flotation circuit) was used to define recoveries and concentrate grades.

Variability testwork conducted in Lima tap water on 15 supergene and 25 primary ore samples, plus the recent work using brine from site on the ten composite samples provided in 2019, were used to further refine recoveries and concentrate grades for the distinct ore types, and on the ranges of copper head grades, mineralogy and pyrite to be expected over the mine life. Separation of copper and molybdenum concentrates was demonstrated to be achievable in the early testwork, but was not repeated in the most recent testwork programme.

Recent work on water supply to the Project has suggested that there is sufficient fresh water to operate the process at a throughput of 40 Mtpa. Recoveries and concentrate grades for a fresh water operation have thus been used in the production schedules and cash flow modelling. These recoveries and grades have been estimated from the earlier testwork in tap water and from the recent testwork, but this work was not undertaken in water from the actual fresh or brackish water to be provided to site from the local bore holes. Likewise not all of the metallurgical domains were tested in locked cycle tests in fresh or brackish water.

The Company has considerable experience in the design and operation of copper concentrators, with recent designs at the Company's Sentinel and S3 (Zambia) and Cobre Panamá operations forming the basis of equipment sizing together with the testwork data. Process designs and equipment sizings would also incorporate experiences from copper concentrate circuits operating in South America.

Some additional testwork is thus recommended to confirm the recoveries and concentrate grades for all ore types.

25.4 Water supply

In the arid environment characterising the Project site, local and regional borefields will need to be developed to supply a combination of fresh water and brine for the Project. Most of the processing water supply is intended to be fresh water abstracted from regional borefields. Brine from the adjacent salar is intended for use in the re-pulping of thickened tailings prior to pumping to the TSF.

Fresh water supply investigations to date have focussed the search and drilling investigations to four regional basins located at 30 km to 50 km distance from the Project site. Major water resources have been identified at Valle de Arizaro, Socompa, Valle de Chaschas and Valle de las Burras.

The four identified fresh water supply basins have a combined estimated yield in excess of that required for process water make-up at 40 Mtpa. On current indications however, additional and potentially more distant, fresh water supply sources will be required to sustain processing at 60 Mtpa.

Brine extraction from the Salar de Arizaro is still being investigated, although indications to date are that a number of bores will need to be located in the adjacent salar(s) in order to supply the quantity of brine required for the Project.

25.5 Environmental studies and permitting

Detailed environmental baseline data collection began in 2016. A Project Alternatives Analysis and a separate Project Description document were completed in 2018 to complement the Project ESIA. The ESIA was submitted to the authorities in February 2019. The Company has subsequently filed documents responding to the Project ESIA review and observations made by the provincial Secretariat of Mining.

Additional and separate ESIAs were submitted in September 2020 and November 2022 for the proposed access road and powerline, respectively.

These submissions remain under review by the relevant authorities.

The Project ESIA process requires a final report from the Mining and Energy Secretariat on their observations and requests for further information. Once the observations process is satisfactorily concluded, there needs to be a public hearing (*audiencia publica*) prior to the ESIA approval.

In accordance with environmental laws governing the Project, the approved Project ESIA will need to be updated every two years. Updates to the Project Description will be incorporated into subsequent ESIA submissions accordingly.

Abstraction permits will be required for the fresh water borefields located at Valle de Las Burras, Valle de Arizaro, Valle de Chaschas and Socompa. In the case of brine abstraction, permit application for borefields and trenches over the Taca Taca and Arizaro salars will be submitted after the results of the ongoing campaign are assessed.

In addition to water permits, other approvals are required for the construction and operation of the mine and ancillary facilities. Specific building permits, waste and chemical handling authorisations, are granted by the provincial and national authorities.

25.6 Infrastructure

The updated overall site layout, including the location of buildings, roads, rail and power lines, and water supply (borefields, pumps, overland pipelines) has been a collaborative effort between FQM and consultants Lycopodium, Fluor and Process E&I.

The level of detail in these designs is sufficient to support the capital and operating cost estimates presented in this Technical Report but will require further updating as detailed engineering of the Project proceeds.

ITEM 26 RECOMMENDATIONS

26.1 Geology and Mineral Resource

With the current wide (~150 m) drill grid spacing, further staged infill and extension drilling is recommended for the purpose of upgrading the Mineral Resource classifications, delineating key structures, and defining shallow ore extensions. Costs associated with this drilling are estimated to be around $7.3 million over a five-year period.

26.1.1 Infill drilling

The current drill spacing at Taca Taca is wider than normal when compared with other low grade, bulk tonnage deposits (e.g. FQM's Haquira and Cobre Panamá projects are drilled at less than 100 m grid spacing). Although estimates validate well at the deposit scale, drilling does not support accurate estimates at the scale of mining.

A staggered 75 m by 150 m drill grid spacing would provide improved accuracy of prevailing geology and mineralisation with upgrades of some blocks from Indicated to Measured categories. Prioritised drilling is recommended as follows:

Starter Pit

A 75 m by 150 m drill programme across the first two years of inpit ore mining is recommended. Mineralisation in this area is high grade within a supergene chalcocite-rich horizon directly below the leached cap. Ensuring confidence in the relative positions, volumes and grades of this mineralisation will improve the ability to deliver feed according to plan. Results would improve estimate confidence of early ore feed for mine planning and would support future drilling requirements, such as grade control drill grid spacings.

TK2 Fault

Areas proximal and west of the deposit-scale TK2 (West) fault are considered a high priority for further definition. Drilling would investigate the risk of disruptions (losses) to mineralisation continuity across the fault zone, whilst also defining the extents of shallower ore typical to this area. Geotechnical data would also be collected to determine the impact of the fault zone on pit slope stability and hydrogeology.

26.1.2 Extensional drilling

Mineralised areas external to the current pit design have limited to poor drill support. The deposit's mineralisation is open to the south and east of the deposit as well as at depth.

26.1.3 Geological work

It is recommended that the following geological work be undertaken for inclusion in future Mineral Resource estimate updates:

1. Around 9% of samples have sequential copper analysis which limits comprehensive definition of different copper species domains. Future drilled samples with sequential copper analysis will improve definition of these copper species domains. Improved domain resolution based on sequential copper grades, pyrite content, and recent metallurgical test results is possible. All future infill drilling must include analysis of samples for sequential copper to build upon the existing dataset. Domains with less mixing would allow for an improved estimate and more predictable metallurgical performances.

2. 30% of samples were drilled using reverse circulation methods with the remainder been diamond core samples. The potential for bias between these two methods sample assays needs to be assessed and managed.
3. A 3D structural model is required to be compiled from the integration of multiple data sets. A recent high-resolution ground-magnetic geophysical survey should be interpreted by a qualified geophysicist and structural geologist. The results can be compared with topographical surveys, surface mapping, drill core logging, and geo-chemical modelling of multi-element assay data. Abundant evidence for faulting in drill-core suggests a 3D model will be important for predicting local changes in weathering profiles and potential disruptions to mineralisation continuity at a mining scale. It will also contribute to geotechnical and hydrogeological modelling.
4. A 3D alteration model should be created, focussing on relative pyrite abundance and vein type/intensity alongside the delineation of broad gangue mineralogy. Geochemical interpretation of existing multi-element assay and SWIR (short-wave infrared) data combined with visual drill core logging and validation would provide primary inputs. Preliminary groundwork was completed by Scott Halley in 2019 but incorporation into the block model would benefit planning, operations, and processing.
5. Predicted work index values should be compared to the alteration model and historic point load data with a view to delineating zones of variable comminution properties.
6. Geometallurgical recoveries were developed and updated according to a historical and recent test work. Results were linked to rock type and key multi-element and sequential copper data. Additional test work and improved geological definition will increase the accuracy of these geometallurgical variables.
7. LUC was applied at a 7.5m*7.5m*15m SMU block size to provide an indication of the grade and tonnages likely at the scale of mining. It is important to note that these results are not locally accurate.
8. Improved resolution on rock strength and rock quality domains focussed on weathered and supergene zones will benefit pit design. Variations in rock strengths and quality will have implications for slope stability, blasting, mining methods, and comminution.
9. While Mineral Resource classification was based upon drill grid spacing, QAQC results, geological and grade continuity, kriging efficiencies and regression slope values as well as an economically determined ultimate pit shell, it will benefit from infill drilling and more detailed study. Implementation of probabilistic methods, such as conditional simulation, will assist in improving understanding of estimate uncertainty and its impact on classification.
10. A geotechnical risk assessment should be completed to evaluate the implications of gravel basins proximal to the pit, and looser material in the topmost portion of the leached cap, on pit slope stability.
11. Drill and sample metadata (hole depths, assay methods, etc.) should be comprehensively audited to ensure details are accurate, complete, and consistently recorded. Merged logging codes should also be reviewed and incorporated into the database.

26.2 Mine planning and Mineral Reserve

26.2.1 Geotechnical drilling and investigation

Additional geotechnical drilling is recommended for the eastern pit wall. The eastern pit wall is planned to be as close as possible to the salar shoreline and to be as steep as possible to enable maximum ore extraction without having to mine into the salar. As such, further geotechnical analysis of this area is recommended.

Currently, only one previous geotechnical drill hole passes through the planned eastern pit wall position. Three holes are recommended; two angled parallel to the pit wall (300 m apart) and one angled to pass back through the pit wall. This would ensure detection of possible fault and fracture orientations.

Given that most of the known structures in the deposit are sub-vertical, the structural information from previous vertical geotechnical drilling is somewhat biased.

Recommendations from the W&N report (2016) should be addressed in the geotechnical drilling and assessments, as follows:

- Geotechnical drilling should include televiewer soundings (optical and acoustical imaging) and be angled in an attempt to intercept the geologic structure which will be critical for pit wall design.
- Any additional angled boreholes, for Mineral Resource infill or metallurgical sampling purposes, should be televiewer logged to verify the population of steeply-dipping discontinuities and counter-balance the data set developed from the preponderance of vertical holes previously televiewer logged.
- A structural geologic model for healed and unhealed faults should be developed to better constrain ore grades and geotechnical domains for the pit walls as the pit planning proceeds. This would include additional downhole televiewer logging of angled boreholes and re-logging of the existing core to target major structural features.
- Block models for rock quality and alteration should be developed. This will enable future investigators to refine the spatial allocation of rock mass shear strength within the stability models.
- A further hydrogeological study should be undertaken to evaluate the potential for seepage through the east pit wall because of its proximity to the salar and to design an appropriate depressurisation scheme to ensure that the wall remains depressurised during the life of the pit.

A proposal has been received from SRK Consulting (July 2025) in respect of mine geotechnical investigations and the provision of field programme services in line with the above assessments. The selection of drill core samples, the undertaking of mechanical testing, and the subsequent geotechnical analyses and engineering of pit slope designs and production blasting parameters will be the responsibility of Company geotechnical engineers. The cost of the consulting work is in the order of $300,000.

An indicative estimate for the cost of geotechnical drilling and analysis is $2.0 M. The hydrogeological component is part of a $20.0 M estimate for the investigation of local and regional water supply catchments in 2026.

26.3 Metallurgy and process engineering

Some additional testwork has been identified to answer specific questions related to the process design, and to confirm recoveries and concentrate grades for all ore types, with the work to be undertaken in fresh or brackish water supplied from site. This work is not considered to be critical for the current plant design and will be performed in the continuing phase of Project engineering, as follows:

- Locked cycle flotation testwork in fresh or brackish water from site, on all ore types to confirm recoveries and concentrate grades during the first five years of operations.
- Cu-Mo separation testwork to define equipment sizing and reagent requirements.
- Testwork to evaluate the use of NaHS for sulphidisation in rougher flotation, to estimated improvements in recovery by means of CPS.
- Optimisation of flotation reagent requirements – particularly frother (and the type of frother).
- Gold recovery testwork from the leach cap (longer term testwork).

Similarly, some trade-off studies have been suggested for future consideration to optimise the plant design for the initial 40 Mtpa phase, and the expansion to 60 Mtpa.

- Regrind circuit energy requirements and mill sizing.
- Evaluate the economics of producing a molybdenum concentrate. Re-look at the design of the molybdenum flotation circuit in light of experience gained from Cobre Panamá.
- A more detailed investigation of how to discharge slurry to the TSF and at what slurry density, and how to reclaim decant water, if a supernatant pool of water forms on the dam.
- A further review of the economics of leaching the auriferous material from the near-surface leached cap
- A review of process building requirements and equipment design to cater for the climatic conditions (wind), and materials of construction to minimise scaling and corrosion.

A provisional allowance of $300,000 is estimated for further testwork, based on recent quotes for similar work on another Company project. Samples for this work may be provided from half core obtained from diamond drilling for grade control, but a provision of $2.5 million (over five years) should be allowed for drilling for metallurgical testwork.

The trade-off studies will be undertaken in-house, or included in the comprehensive process review and design work that is required, which will be undertaken by third party engineering firm(s). As such, the cost is assumed to be included amongst the Project capital development costs.

26.4 Tailings storage facility (TSF)

Several aspects of the tailings storage facility (TSF) design remain unresolved and will require further analysis during subsequent engineering phases. These outstanding items will be addressed through project team collaboration, targeted studies, or trade-off assessments to refine the design and mitigate uncertainties.

Extensive study work has been undertaken by SRK, including seismic assessments, conceptual TSF Design, and an Engineering Cost Report. A report from SRK on an assessment of the gap analysis between the ANCOLD (Australian National Committee on Large Dams) and GISTM (Global Industry Standard on Tailings Management) approach to TSF design is outstanding. However it is expected that SRK will recommend a more robust dam break analysis and assessment of tailings viscosity to support this analysis be undertaken.

Optimisation work is recommended for the tailings delivery and spigotting arrangement, with the objective of prolonging the life of the existing railway formation skirting the western edge of the TSF, and also possibly delaying the need to build the embankment at the neck of the Salar de Taca Taca where it joins the Salar de Arizaro.

A provisional allowance of $500,000 is recommended for further studies. This work is also not considered to be critical for the current plant design and will be performed in the continuing phase of Project engineering. Costs for detailed design of the TSF are covered under the capital estimate for Project development.

During operations, on-site testing of beach slopes will ensure alignment with design parameters, while a monitoring framework and trigger action response plan will be implemented to maintain facility integrity.

26.5 Water supply

The revised water supply strategy is focused on expanding supply options beyond the existing local catchments proposed in the ESIA, consistent with the recommendations of Piteau (2025). This includes both deep aquifer exploration, and exploration of identified regional catchment targets.

If additional water supply sources are demonstrated, the water supply strategy can be optimised to consider groundwater impact assessments (GWIAs) for individual catchments in addition to other technical and economic factors.

The following work is planned / recommended for 2026:

1. Continued detailed assessments of the Valle de Chaschas and Valle de Las Burras local catchments, including long-term, multi-well pumping tests and groundwater model updates to assess potential impacts.
2. Evaluation of deep aquifer potential in Valle de Chaschas and Valle de Las Burras, including geophysics, drilling, and deep pumping tests
3. Investigation of regional catchments as alternative water sources to some of the local catchments
4. Detailed groundwater studies, including age dating using isotopes and chloride mass balance techniques, to understand groundwater age and improve recharge and water balance estimates.
5. Development and implementation of a comprehensive, quarterly surface water and groundwater monitoring programme, to provide data needed to support baseline studies, engagement with the IFC, and to improve water balances and estimates of water supply potential.
6. Completion of catchment-scale summary reports covering water balances, estimation of probable long-term water supply potential, GWIA, water risk assessments, data gaps assessment, and recommendations for water supply strategy.

An indicative estimate for the ongoing hydrogeological investigations was mentioned above as $20.0 M for 2026.

26.6 Infrastructure

There are several infrastructure aspects for the Project that should advance beyond the current stage of engineering. These aspects are as follows:

- The site layout plan requires further optimisation and possible enhancement. This review and design should be carried out in conjunction with the civil geotechnical programme.
- Whilst the Project power supply requirements have been estimated, a detailed itemisation and specification of power reticulation requirements across the Project site is yet to be commenced.

Much of this optimisation and detailed itemisation work will be completed inhouse. A number of work packages have already been compiled to advance the Project engineering phase. These packages are aimed at reducing any technical risk and progressing the enabling works which support site access, and pioneering mining activities. The cost of these packages is included within the capital estimate for Project development.

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ITEM 28 CERTIFICATES

David Gray
First Quantum Minerals Ltd
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Tel +61 8 9346 0100; david.gray@fqml.com

I, David Gray, do hereby certify that:

1. I am the Group Mine and Resource Geologist employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled "Taca Taca Project, Salta Province Argentina, NI 43-101 Technical Report", dated effective 31st December 2025 (the "Technical Report").
3. I am a professional geologist having graduated with a Bachelor of Science degree with Honours (1988) in Geology from Rhodes University in Grahamstown, South Africa.
4. I am a Fellow Member of the Australian Institute of Geoscientists (FAIG) and a Member of the Australasian Institute of Mining and Metallurgy.
5. I have worked as a geologist for a total of thirty six years since my graduation from university. I have gained over twenty years of experience in production geology and over five years of exploration management of precious, base metal and copper deposits. Over the last fifteen years I have held senior technical mineral resource positions in copper mining companies operating in Central Africa and worldwide.
6. I have read the definition of "qualified person" as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
7. I most recently personally inspected the Taca Taca property described in the Technical Report in March 2019.
8. I am responsible for the preparation of those portions of the Technical Report relating to geology, data collection, data analysis and verification and Mineral Resource estimation (namely Items 7 to 12 and 14).
9. I am not independent (as defined by Section 2.3 of NI 43-101) of First Quantum Minerals Ltd.
10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in the assurance of sampling QAQC, optimisation of estimation methods and the development of geology and mineralisation models.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 19th day of February 2026 at West Perth, Western Australia, Australia.

David Gray

Michael Lawlor
First Quantum Minerals Ltd
Level 2, 18 – 32 Parliament Place, West Perth, Western Australia, 6005
Tel +61 8 9346 0100; mike.lawlor@fqml.com

I, Michael Lawlor, do hereby certify that:

1. I am a Mining Technical Advisor employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled "Taca Taca Project, Salta Province Argentina, NI 43-101 Technical Report", dated effective 31st December 2025 (the "Technical Report").
3. I am a professional mining engineer having graduated with an undergraduate degree of Bachelor of Engineering (Honours) from the Western Australian School of Mines in 1986. In addition, I have obtained a Master of Engineering Science degree from the James Cook University of North Queensland (1993), and subsequent Graduate Certificates in Mineral Economics and Project Management from Curtin University (Western Australia).
4. I am a Fellow of the Australasian Institute of Mining and Metallurgy.
5. I have worked as a mining and geotechnical engineer for a period in excess of thirty years since my graduation from university. Within the last fifteen years I have held senior technical management positions in copper mining companies operating in Central Africa, and before that, as a consulting mining engineer working on mine planning and evaluations for base metals operations and development projects worldwide.
6. I have read the definition of "qualified person" as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a "qualified person" for the purposes of NI 43-101.
7. I most recently personally inspected the Taca Taca property described in the Technical Report in July 2016.
8. I am responsible for the preparation of those portions of the Technical Report relating to Mineral Reserve estimation and Mining, namely Items 15 and 16, respectively, and for Items 1 to 6, 15, 16, and 18 to 20, 21 in respect of mine operating and metal costs, and 22 to 26.
9. I am not independent (as defined by Section 2.3 of NI 43-101) of First Quantum Minerals Ltd.
10. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement has been in mine planning and the preparation of scoping studies, commencing in 2013.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 19th day of February 2026 at West Perth, Western Australia, Australia.

M. Lawlor

Michael Lawlor

Andrew Briggs
First Quantum Minerals Ltd
Level 2, 18 – 32 Parliament Place, West Perth, Western Australia, 6005
Tel +61 8 9346 0100; andrew.briggs@fqml.com

I, Andrew Briggs, do hereby certify that:

1. I am the Group Consulting Project Metallurgist employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled “Taca Taca Project, Salta Province Argentina, NI 43-101 Technical Report”, dated effective 31st December 2025 (the “Technical Report”).
3. I am a professional metallurgist having graduated in 1974 from the Imperial College (Royal School of Mines), London, with a BSc (Eng) First Class in Metallurgy.
4. I am a Fellow of the Southern African Institute of Mining and Metallurgy.
5. I have worked as a process engineer and metallurgist since graduation in 1974 (46 years); the first thirteen years of which were in operating positions up to Metallurgical Manager in the gold mining industry. This was followed by nineteen years in engineering companies in Process Design for projects worldwide, and finally thirteen years with First Quantum Minerals Ltd as a Process Consultant.
6. I have read the definition of “qualified person” as set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I am a “qualified person” for the purposes of NI 43-101.
7. I most recently personally inspected the Taca Taca property described in the Technical Report in September 2018.
8. I am responsible for the preparation of those portions of the Technical Report relating to mineral processing/metallurgical testing and recovery methods, namely Items 13 and 17, respectively. I am also responsible for the estimates in Item 21 pertaining to processing, plus general and administration costs, and for processing commentaries in Items 25 and 26.
9. I am not independent (as defined by Section 2.3 of NI 43-101) of First Quantum Minerals Ltd.
10. I have been involved with the property that is the subject of the Technical Report, since inception. This work has included metallurgical testwork, process design for the plant and associated infrastructure, project planning, and engineering studies. I am also responsible for the estimates in Item 21 pertaining to processing, plus general and administration costs.
11. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required for it to be disclosed and to make the Technical Report not misleading.

Signed and dated this 19th day of February 2026 at West Perth, Western Australia, Australia.

Andrew Briggs