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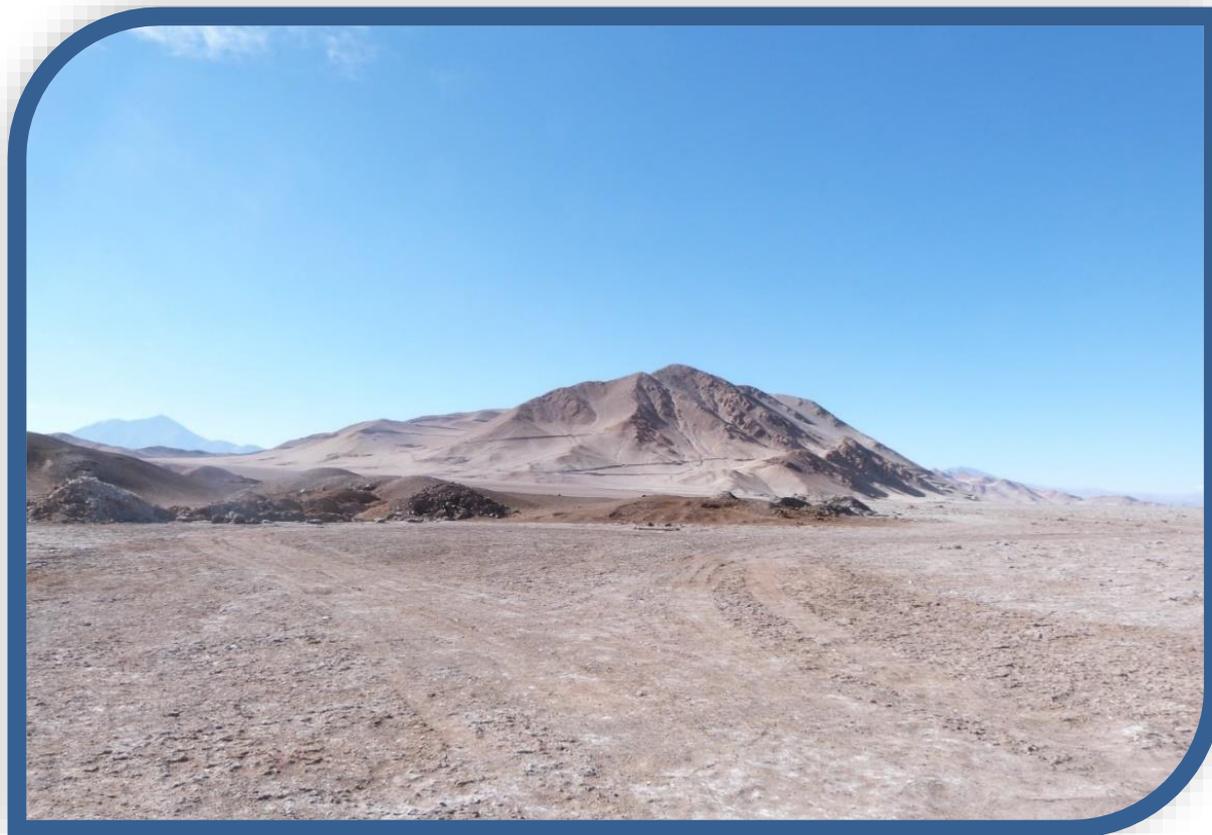


Taca Taca Project

Salta Province, Argentina

Amended and Restated NI 43-101
Technical Report

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ITEM 2 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, subsequent to its acquisition from previous owners, and to provide a commentary on the project development status for the property.

The Initial Technical Report has now been amended and restated 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

The effective date for the Mineral Resource and Mineral Reserve estimates remains as the 30th October 2020.

2.1 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 of 32 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 2-1 shows the location of the Project relative to major roads, railway lines and other existing infrastructure.

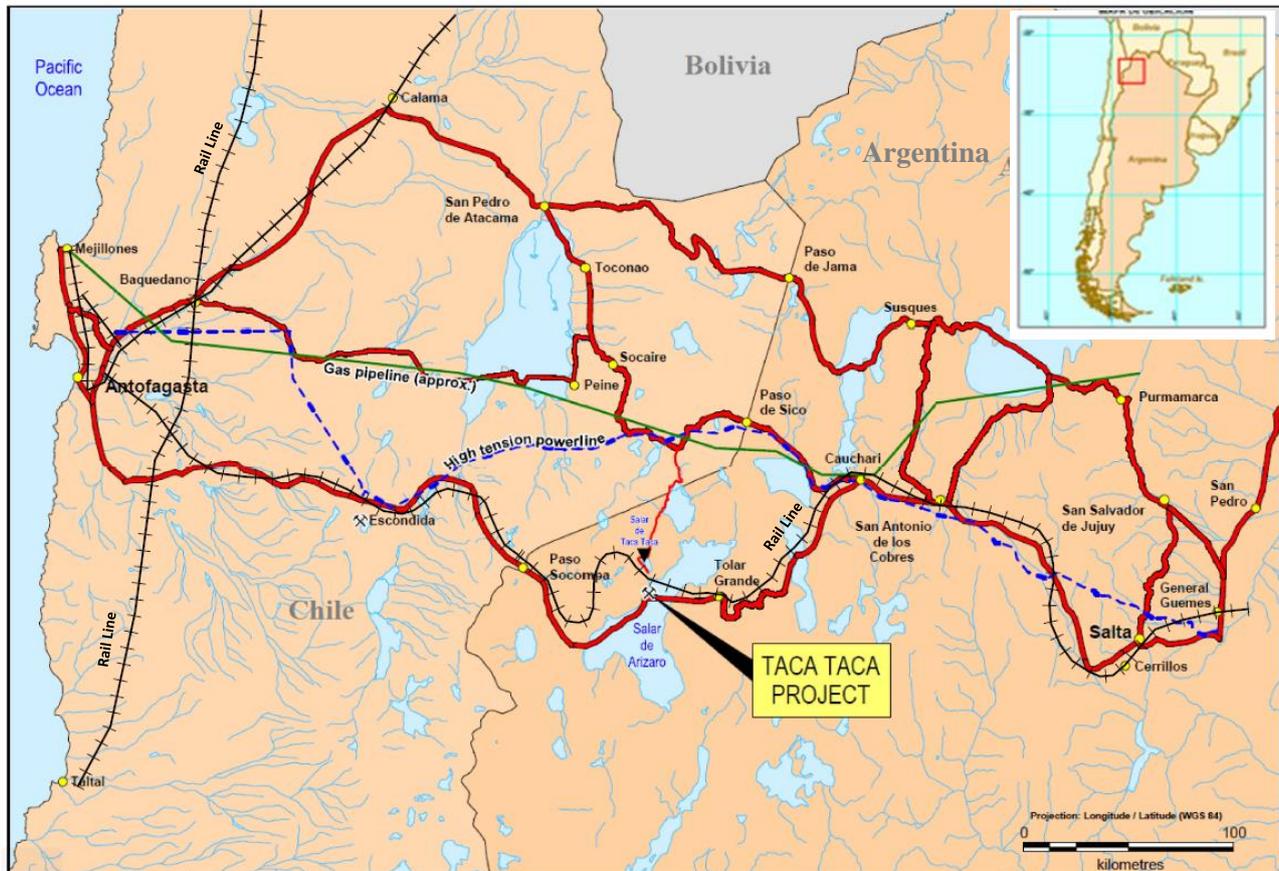
The Project site is situated at a median elevation of 3,625 mRL, in an environment with sparse flora and fauna, and on the edge of an expansive salt lake (salar). The climate at Taca Taca is arid, with an annual precipitation of approximately 40 mm/year and an evaporation rate of 2,500 mm per year. Temperatures range from minus 11°C to 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 75 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.

2.2 Project background

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.

Figure 2-1 Taca Taca Project location



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.

2.3 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*, 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, and associated facilities.

The Project ESIA was submitted to the authorities in February 2019. A response to the submission was received from the Secretariat of Mining at the end of Q3 2019, and it included 62 observations (including requests for clarification or more information). Some of the required additional information will only be available once the Project definition and engineering is more advanced. A compiled document with responses that were able to be provided at this stage of the Project was submitted to the authorities in Q1 2020. Final approval of the ESIA is expected in 2021.

After final approval, the ESIA must be updated and resubmitted to the authorities every two years.

A second ESIA is required to be submitted separately to the Energy Secretariat of Salta Province, for the 345 kV transmission line to connect the Project to the national electrical grid. A third ESIA is required to be submitted to the Salta Road Administration for the proposed bypass road construction for the Project.

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The Project will also require approval from the Water Resources Secretariat of Salta Province of a concession for water supply development and use. The current water exploration programme (Phase III) is intended to develop all of the water supply definition and information that will be required to include in a water exploitation permit. The Phase III water investigation programme was suspended in March 2020 due to the COVID-19 pandemic, and will now be completed at a later date.

These three additional material permit applications are in preparation, and with the exception of the water permit application, are anticipated to be filed with the relevant authorities in 2021.

Other administrative authorisations, detailed construction and operating permits will be required, particularly from the Municipality of Tolar Grande and various provincial authorities in the course of development and operation of the Project.

2.4 Project development status

A primary reference document for the Project is a Preliminary Economic Assessment (PEA) report dated May 2013 and produced for Lumina (Ausenco, 2013). This document describes the conceptual development of the Project as envisaged by its previous owners, and includes technical information sufficient for the Company to review and assess its own options. More specifically, the detailed capital and operating costs set out in the PEA report have formed the basis for review, benchmarking and adaptation by the Company for its own economic analysis, to a level that is considered appropriate for the Project at this current stage.

Further to the Project review and assessments described above, the Company has identified its preferences for the scale and extents of open pit mining and ore processing, and for the location of required infrastructure items. Furthermore, technical work has proceeded from 2017 through 2020, on power and water supply logistics, freight and product transport options, and on designing improved road access into the Project area.

The Company's Project engineering phase has advanced such that an updated Mineral Resource and Mineral Reserve estimate can now be published, along with a proposed mining and processing plan.

Those aspects of the proposed Project which have changed since the Lumina PEA are as follows:

- an updated Mineral Resource model and estimate have been produced
- a maiden Mineral Reserve estimate has been produced
- a revised deep open pit mine design has been developed, and has been constrained from transgressing into the adjacent brine saturated sediments of the Salar de Arizaro
- metallurgical variability criteria for copper mineralisation has been coded into the updated Mineral Resource model for improved production planning
- a flow sheet has been developed which accommodates the processing of a significant proportion of supergene ore with mixed mineralogy
- a Project alternatives analysis and a detailed Project description document have been produced to complement the submission of an Environmental and Social Impact Assessment (ESIA)
- separate ESIA documents, relating to road access and power transmission have been prepared for forthcoming submission
- from the Project alternatives analysis:
 - a new location for the Project tailings storage facility has been selected
 - preferred locations have been identified for a new access road into the Project, and for the route of a power transmission line
- water supply investigations have continued, fresh water supply sources identified, and preliminary engineering completed on borefield and pipeline requirements

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- capital cost estimates have been updated inclusive of new information on road and rail upgrades, and water supply borefields and pipelines
- mine operating cost estimates, mining equipment fleet requirements and process operating costs have been derived from a first principles basis

The Company has progressed through to 2020 with a Project engineering phase, focussing on a scope of work involving further drilling and confirmatory Mineral Resource drilling/sampling/assaying, further Mineral Resource modelling and updates, metallurgical sampling and testing, geotechnical studies, infrastructure engineering and related technical work, and on Mineral Reserve estimation.

In addition to resuming the Phase III water supply investigations and confirmation of borefield sustainability, a scope of continuing engineering phase work includes:

- additional Mineral Resource drilling, sampling and analysis, including infill, extensional and sterilisation targets
- mine and civil geotechnical investigations, in conjunction with seismicity investigations
- optimisation of the process plant layout and the concentrate load-out facilities
- further confirmatory metallurgical testwork, not critical for the current processing flow sheet and plant design
- further infrastructure planning for power reticulation
- optimisation of the tailings delivery methodology and the potential for decant water return
- selection of a suitable location for the Project camp and related infrastructure
- review of waste landfill options and locations

2.5 Project scope

The proposed Project has the following material components:

- a multi-phased open pit mine extending to an ultimate depth in excess of 700 m
- surface stockpiles for marginal ore, and separately for auriferous material not in the Mineral Reserve, but which could be economically processed in the future
- a mining waste dump, located to the east of the open pit, on the surface of the adjacent salar (the Salar de Arizaro)
- a processing plant site which is located adjacent to the open pit, in an area of relatively flat topography sheltered from the prevailing wind direction
- a concentrator for the processing of copper mineralisation by flotation, with primary recovery of gold (and silver¹) into the concentrate
- separation of copper and molybdenum concentrates
- ramp-up of the processing rate from 30 Mt in the first year, to 40 Mtpa for the next five years, followed by a 50 Mtpa rate for one year, and then to an eventual rate of 60 Mtpa from Year 8
- a nearby tailings storage facility located within an embayment of the Salar de Arizaro
- process water storage ponds
- borefields for the supply of fresh and saline water
- overland pipelines between the concentrator and the tailings storage facility, and between borefields and the plant site

¹ Silver (Ag) is considered immaterial to the economic value of the Project and hence Ag grades have not been reported in the Project Mineral Resource and Reserve estimates, nor accounted for in the Mineral Reserve economic analysis.

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- surface haulage and internal access roads
- mine services workshops and equipment wash-down facilities
- construction offices, mine administration and accommodation facilities
- storage space and a rail load-out facility for copper and molybdenum concentrate products
- parts and consumables, reagent and explosives handling and storage facilities
- as additional infrastructure, there are roads for transporting supplies into the Project site, a railway for transporting concentrates and supplies, and a high voltage electric transmission line

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.

2.6 Geology and mineralisation

Taca Taca has porphyry copper-gold-molybdenum mineralisation located in the southern half of a 50 km long Ordovician batholith, which forms the Sierra de Taca Taca mountain range. The Taca Taca mineralisation is hosted by plutonic rocks of granitic composition together with lesser dacite, dolerite, and rhyolite intrusions. The porphyry is characterised by kilometre-scale zones of hydrothermally altered rocks that grade from a central potassic core to outer phyllic and argillic zones. Phyllic alteration is most pervasive across the deposit and is closely associated with mineralisation.

Mineralisation is comprised of supergene (chalcocite) and hypogene (chalcopyrite) zones. A sub-surface leached horizon of varying thickness overlies the supergene and hypogene mineralisation. Mineralisation is disseminated and in fractures, veinlets, and quartz vein stockworks.

The leached horizon is largely depleted of copper mineralisation except for a zone of chalcocite-rich ore perched within the leached material to the east of the deposit. In addition, a zone of supergene gold mineralisation, close to surface, is present above the thickest portion of leached material.

Hypogene copper sulphides are mostly chalcopyrite with lesser bornite, chalcocite, covellite, and digenite. The mineralisation is broadly zoned with a chalcopyrite-bornite-molybdenite core yielding to a stronger pyritic halo around the outer edges.

Supergene zones are mostly secondary sulphides formed by enrichment within a discontinuous blanket underneath the leached cap. Supergene mineralisation is often variably mixed with hypogene mineralisation and is often due to deep-seated alteration along structures and host rocks. Fine-grained black chalcocite and lesser covellite are the main secondary copper sulphides.

Mineralisation remains open at depth and around several peripheral areas of the deposit.

2.7 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

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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 the ultimate design throughput of 60 Mtpa in two milling trains.

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.

The data generated from the recent locked cycle testwork was combined with the variability testwork results obtained in the previous testwork campaigns 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 are anticipated:

- copper recovery of 85.0% to a concentrate grade of 25.3% Cu
- molybdenum recoveries of 40% to a concentrate grade of 47% to 50% Mo
- gold recoveries to the copper concentrate of 60%, with a grade of approximately 4.5 g/t

2.8 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.

Open pit mining would proceed in phases from an initial starter pit, supplying pre-strip development waste for site infrastructure and construction, and ore onto stockpile for process plant commissioning. The average and maximum material movements over this three year timeframe are 32.9 Mbcm and 43.3 Mbcm, respectively. There is a pronounced peak in material movements over the next ten years as the first three pit phases are completed and mining proceeds into the fourth phase. The average and maximum material movements over this period are 91.9 Mbcm and 95.7 Mbcm, respectively. Thereafter, the average and maximum material movements reduce to 42.3 Mbcm and 65.2 Mbcm, respectively.

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Figure 2-2 shows the Project site layout and specifically the location of the open pit and associated waste dump and ore stockpile area. Subject to infill drilling and further mine planning assessments, a small satellite pit immediately to the north of the design pit could be mined during the operations phase, although this is not currently part of the Mineral Reserve inventory.

The detailed mine planning for this Technical Report included conventional optimisation processes, phased and ultimate pit designs, surface layout planning and life of mine production scheduling. At the outset, conventional Whittle Four-X software was used to determine the optimal pit shell, operating with a sulphide flotation plant to produce separate copper and molybdenum concentrates. The copper concentrate would contain gold.

The optimisation was completed on a maximum net return basis (undiscounted) and with recoveries of copper into concentrate based on defined mineralogical groupings or clusters. Fixed (conservative) recovery values were used for molybdenum and gold. The optimisation process also considered open pit slope design criteria provided by a geotechnical consultant, in addition to mine operating costs derived from first principles and processing/G&A costs determined on a similar basis.

2.9 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.

Supergene mixed ores would be encountered during the initial processing years, to be followed by increasing quantities of hypogene ore as the open pit deepens. The “leach” cap at surface is auriferous but is mostly barren of copper mineralisation². The auriferous material would be separately stockpiled for future evaluation of the economics of gold recovery.

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 brine, sourced from the adjacent salar. Cleaner flotation would be undertaken in fresh/brackish water, sourced from offsite borefields.

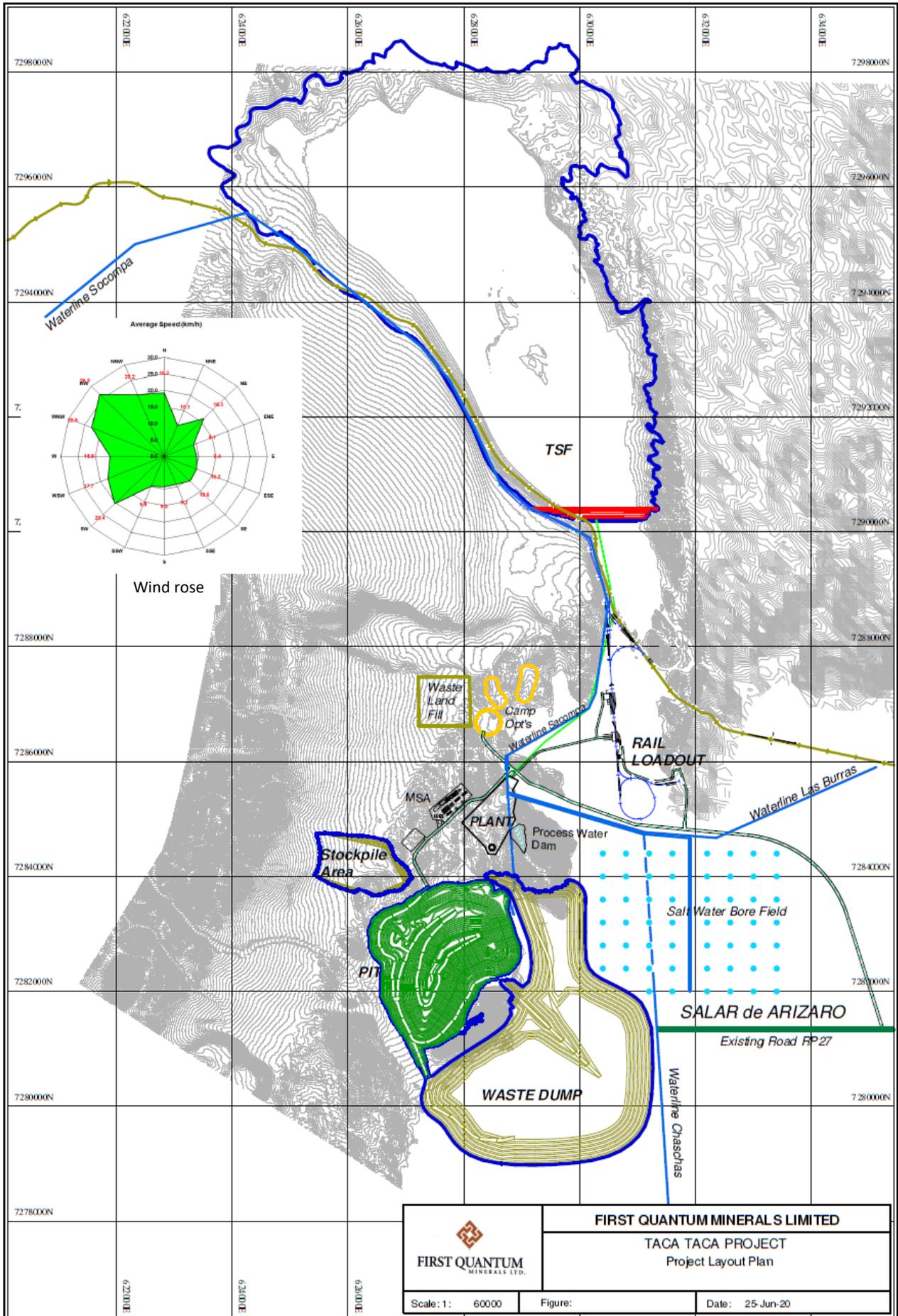
The processing facilities are designed for an initial throughput of 30 Mtpa in Year 1, then 40 Mtpa in Years 2 to 6, then 50 Mt in Year 7, and finally 60 Mtpa from Year 8.

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 two x 22 MW ball mills (for 60 Mtpa).

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.

² There are discrete zones (i.e., metallurgical domains) of “perched” copper and gold mineralisation in the near surface “leach” cap.

Figure 2-2 Layout of the proposed Project site



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The addition of sodium hydrosulphide (NaHS) is proposed as a means of improving the recovery from the oxidised and mixed ores; it will sulphidise the oxidised and tarnished mineral surfaces and assist in flotation. Facilities will be provided for NaHS addition to the circuit; it is expected to be beneficial for approximately 22% of the ore feed.

Copper and molybdenum concentrates would be separated from the bulk cleaner concentrate, filtered and dispatched to off-site smelters.

Ore would be delivered from the mine by haul trucks and crushed in four primary crushers located on surface. Following primary crushing, the proposed processing plant would comprise:

- secondary crushing, to reduce the SAG mill feed size
- 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
- a gravity recovery circuit on the ball mill cyclone underflow for coarse gold recovery
- pebble crushing on scats generated from the SAG mills
- rougher and scavenger flotation of cyclone overflow slurry with controlled sulphidisation flotation (using NaHS) for oxidised and tarnished supergene ores
- thickening of flotation tails
- pumping of thickened tailings to the tailings storage facility (TSF)
- potential reclaim of decant water from the TSF for usage within the process³
- dewatering of rougher concentrates, to remove brine
- regrind of dewatered rougher concentrates in a high intensity grinding (HIG) mill, followed by dilution in fresh water
- 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

An average of 985,500 wet tonnes of copper concentrate is expected to be generated annually at an average grade of 25.3% Cu, along with 6,200 tonnes of molybdenum concentrate at a grade of 47% to 50% Mo.

Gold would be recovered to the copper concentrate through flotation. Coarse gold recovery would be enhanced by the addition of gravity concentrators.

Flotation tailings would be dewatered in thickeners and the thickened slurry pumped to a tailings storage facility (TSF) located approximately 5 km from the process plant, within an embayment of the Salar de Arizaro.

³ The current process plant water balance assumes that there will be no reclaim from the TSF.

A gold recovery circuit for the treatment of the auriferous leach cap is not proposed at this stage. This material would be stockpiled separately, subject to further testwork during the early phases of operations, and could be reclaimed and treated at a later date if deemed economic to do so.

2.10 Tailings storage and water reclamation

An upstream raised TSF is planned to be located in a natural embayment of the Salar de Arizaro (i.e., the Salar de Taca Taca), located to the north of the processing plant site. The ultimate capacity is approximately 1.37 Bm³ 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 height (25 m plus an additional 3 m of freeboard), short length embankment at the entrance to the salar.

The starter embankment would be constructed as an initial waste rock bund, and then upstream raised in continuous stages using cycloned tailings.

Flotation tailings slurry would typically be at a slurry density of 32% solids and would be discarded to tailings at a thickened density of 55% to 60% solids. 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.

2.11 Power and water supply

The total power demand for the Project is expected to be in the range of 180 MW to 240 MW at a processing rate of 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 a 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.

In the arid environment characterising the Project site, local and regional borefields will be developed to supply a combination of high and low salinity water for the Project. Brine water from the adjacent salar is intended for use in milling and rougher flotation, comprising approximately 72% of the estimated 6,318 m³/hour make-up water required for processing. The balance of the processing water supply is intended to be fresh or brackish water abstracted from regional borefields.

Fresh water supply investigations to date have now focussed the search and drilling investigations to four regional basins located at 30 km to 50 km distance from the Project site. Relative to the water supply status of the Project at the time of the Lumina PEA (Ausenco, 2013), the status as at Q3 2020 is:

- Major water resources have been identified at Valle de Arizaro, Valle de las Burras, Valle de Chaschas, and Socompa, 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.

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- Remote sensing weather data, monitoring, and environmental baselining works are continuing, thereby allowing for increased confidence in water balance predictions for the Project.
- Brine extraction from the Salar de Arizaro is still being investigated, although indications to date are that a significant number of bores will need to be located in the adjacent salar in order to supply the quantity of brine required for the Project.

Contemporary with the ongoing investigations, 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 by the Company, considering the number of bores required, the drilling depth, bore pumps, the pipeline distances and the pumping head.

2.12 Road and rail access

Existing public roads provide access to the Project site. However a bypass of an existing road is planned to avoid a section with narrow switchbacks and another section which is subject to seasonal weather disruptions. A preferred deviation has been selected and would require an approximate 26.5 km length of new road construction. In addition, there is an 18.5 km length of existing road to be diverted around Tolar Grande, and a 31 km length of existing road to be diverted around the Project site.

The Project is located approximately 5 km from an existing railway line that connects Salta with Mejillones, Chile. It is expected that 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 a significant length of the railway will be required. Engineering of the railway line is now being addressed in some detail and high-level discussions have been initiated with the owners of respective sections of the rail corridor.

It is envisaged that the cost of upgrading the existing railway formation would, ordinarily, be borne by the separate railway owners and that this cost would be recovered by them in the concentrate and other freight charges levied on the Company. In the Project capital cost estimates, however, it is assumed that the comprehensively itemised costs estimated for the Company by a specialist consultant, will be included as owner's costs. The cost of the required rolling stock and the concentrate load-out facility are similarly included in the Company's capital cost ledger.

2.13 Port for concentrate export

Potential concentrate export shipping ports in Mejillones Bay have been visited by Company representatives and preliminary discussions held with the port owners.

It is envisaged that the cost of upgrading and expanding one of these ports would be at the expense of the port owner, who would then recoup that cost through a concentrate handling charge levied on the Company. In the Project capital cost estimates, however, it is assumed that the Company would bear the estimated cost for port upgrades and/or expansion works.

2.14 Mineral Resource estimate

The Company produced a revised estimate in December 2019 incorporating the following improvements:

- Geological domains were identified from weathering, rock-type, alteration, and dominant mineralisation characteristics. These formed the basis of a new 3D geological model, including interpretations of an updated base of leach, a partial leach zone, and a perched copper horizon within

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the leached cap. Sequential copper assay data was corrected as per a rate of oxidation study and then used to guide domain classification. Domain geostatistics were also updated accordingly.

- A deposit-scale fault position was updated based on visual and core photography verification.
- The Mineral Resource estimate was reclassified based on confidence in the estimates as per kriging efficiency, regression slope, and degree of geology and slope continuity. It was constrained to within 100 m from the current design pit limits.

Data from a total of 435 diamond and reverse circulation drilled holes, for a total of 75,803 analysed samples, was included in the Mineral Resource estimate. Drill data (logging and sampling) was combined with surface geology mapping and geology modelling to provide defined zones of mineralisation.

The grade estimates were completed in the following stages:

- domains of mineralisation were defined via comprehensive data analysis
- domains were spatially represented as wireframe models and coded into a block model
- sample chemical data was geostatistically assessed for each domain and then estimated using ordinary kriging
- copper, gold, molybdenum, silver, iron and sulphur grades were estimated

Block model grade estimates were validated using summary statistics, visual validations, swath plots and comparison with previous estimates. Estimates were classified as Measured, Indicated and Inferred Mineral Resources. Mineral Resource classification was guided by confidence in the grade estimates and underlying geology model. In addition, drill grid spacing, QAQC and an ultimate pit shell were used to guide the classification limits of mineralisation having reasonable prospects for eventual economic extraction.

The block model estimates were reported at a 0.13 % copper equivalent (Cu_{eq}) cut-off grade, which is consistent with the Mineral Reserve estimate. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The October 2020 Mineral Resource statement is listed in Table 2-1. The Mineral Resource inventory is inclusive of the Mineral Reserve inventory.

Table 2-1 Mineral Resource statement as at October 2020, using a 0.13% Cu_{eq} cut-off grade

Classification	Volume (Mbcm)	Tonnes (Mt)	Density (t/m^3)	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

2.15 Mineral Reserve estimate

A maiden Mineral Reserve estimate was produced for Taca Taca in Q3 2020 (Table 2-2). 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 \$3.00/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 varies according to the copper recovery assigned to the various mineralogical groupings. However, the overall average marginal copper cut-off grade is in the order of 0.13% Cu_{eq} . An elevated cut-off grade of 0.20% Cu_{eq} applies to the plant feed inventory for the production schedule.

Table 2-2 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 is a small area in the north west that crosses over into a joint venture concession. This encroachment amounts to approximately 1.7 Mt of ore at an average grade of 0.38% Cu.

2.16 Production schedule

The production schedule for this Technical Report is listed in Table 1-3, whilst Figure 2-3 shows the annual mining volumes (ore and waste, by mining phase) for the Taca Taca life of mine. Figures 1-4 and 1-5 show the plant feed and recovered metal profiles, respectively.

Features of the mining and production schedule are as follows:

- Mining commences in Year -3 starting with the pre-strip period, whilst processing commences in Year 1. The Project life (processing years) is 32 years.
- 240.1 Mt of waste is mined in the three year pre-strip period, during which time 17.4 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 4,543.0 Mt (1,737.0 Mbcm) of which:
 - 1,758.5 Mt is ore with average grades of 0.44% Cu, 0.012% Mo and 0.09g/t Au, and
 - 2,784.5 Mt is waste
- The overall life of mine strip ratio (waste tonnes: ore tonnes) is 1.6 : 1.
- The direct feed ore to the plant is 1,390.4 Mt at an average grade of 0.50% Cu, whilst 57.1 Mt at an average grade of 0.43% Cu is ore reclaimed from active stockpiles, and 311.0 Mt at an average grade of 0.15% Cu is ore (marginal ore) reclaimed from longer term stockpiles (mostly in Years 27 to 32).
- The total ore mined includes 39.0 Mt of ore grading 0.46% Cu from the near-surface “leached cap”, of which over 15 Mt is mined to stockpile during the pre-strip years. Most of this ore is then processed over the following three years.
- There is a small area in the north west of the ultimate pit design that crosses over into a joint venture concession. The encroachment occurs between Years 5 and 15 and involves 1.7 Mt of ore and 47.5 Mt of waste.
- The Inferred Mineral Resource that is mined as waste amounts to 69 Mt at an average grade of 0.31% Cu (i.e., about 2.5% of the total waste mined). This material is encountered in the mining schedule after Year 6, and following completion of mining phases 1 and 2.
- The crusher feed ramps up from Year 1 at 30 Mt, to 40 Mt in Year 2, at which level it remains until Year 6. The feed rate then rises to 50 Mt in Year 7, and thereafter to 60 Mtpa until Year 32.
- In terms of total plant feed (after mining dilution/recovery):
 - the average copper grade is 0.72% Cu for the first six years when processing at up to 40 Mtpa,
 - then 0.45% Cu to Year 27 when processing at up to 60 Mtpa,
 - and finally 0.15% Cu for the remaining five years of Project life when reclaiming from longer term stockpiles
- Before the final five years of marginal ore reclaim, the total plant feed is 1,476.3 Mt at an average grade of 0.50% Cu.

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- The annual average copper metal production to Year 6 is 227.0 kt, and ranging between 97.5 kt and 275.2 kt. Thereafter, the annual average is 200.4 kt, and ranging between 72.5 kt and 271.1 kt (ignoring the final year of processing). In terms of life of Project totals:
 - 1,362.0 kt of copper is recovered in the first six years,
 - then 4,869.3 kt of copper to Year 27,
 - and finally 341.8 kt of copper for the remaining five years of Project life when reclaiming from longer term stockpiles
- The annual average molybdenum metal production to Year 6 is 2,205 t, and ranging between 1,434 t and 2,912 t. Thereafter, the annual average is 2,776 t, and ranging between 1,745 t and 4,147 t (ignoring the final year of processing).
- The annual average gold recovered into concentrate to Year 6 is 106.3 koz and ranging between 90.9 koz and 134.1 koz. Thereafter, the annual average is 92.9 koz, and ranging between 45.7 koz and 156.6 koz (excluding the final year of processing).
- Of the total 6,573.1 kt of copper recovered over the Project life, only 5,535 tonnes (0.08% of the total) of this would be attributable to ore mined within the adjoining joint venture concession.

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Table 2-3 Life of mine production schedule

Year	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)
-3					43.5	43.5							
-2	3.6	0.22	85.80	0.10	97.1	100.6							
-1	13.9	0.39	107.20	0.12	99.5	113.4							
1	33.6	0.45	115.90	0.16	171.2	204.7	29.9	0.52	119.80	0.16	97.5	1.4	91.8
2	40.4	0.60	93.18	0.11	205.4	245.8	39.9	0.63	106.56	0.14	193.2	1.7	104.1
3	34.8	0.94	135.26	0.13	210.8	245.6	39.9	0.82	128.22	0.13	275.2	2.0	98.5
4	37.4	0.74	145.01	0.12	208.7	246.2	39.9	0.76	143.66	0.12	262.1	2.3	90.9
5	43.7	0.74	167.10	0.16	202.2	245.8	39.9	0.79	178.34	0.17	274.6	2.8	134.1
6	57.8	0.61	166.46	0.15	185.3	243.0	39.9	0.74	182.47	0.15	259.3	2.9	118.4
7	63.1	0.53	134.67	0.12	174.4	237.5	49.9	0.63	163.15	0.15	271.1	3.3	142.8
8	61.2	0.49	128.28	0.12	174.3	235.5	59.9	0.52	147.62	0.14	268.5	3.5	156.6
9	79.9	0.44	102.43	0.10	156.3	236.2	59.8	0.52	126.57	0.12	270.9	3.0	137.0
10	78.2	0.41	96.23	0.09	158.2	236.4	59.9	0.48	116.80	0.11	246.9	2.8	122.3
11	74.5	0.36	79.27	0.08	96.7	171.2	59.9	0.42	90.02	0.09	214.6	2.2	102.9
12	77.1	0.38	80.36	0.08	74.3	151.5	59.9	0.44	92.14	0.09	225.4	2.2	103.8
13	73.9	0.39	77.01	0.07	67.4	141.3	59.9	0.45	84.64	0.08	227.2	2.0	92.3
14	72.5	0.40	83.89	0.07	67.6	140.1	59.9	0.45	90.85	0.08	231.0	2.2	92.8
15	76.6	0.38	93.84	0.08	63.9	140.5	59.9	0.45	101.85	0.09	231.3	2.4	107.2
16	80.0	0.37	110.48	0.08	59.6	139.7	59.9	0.45	117.28	0.09	230.1	2.8	105.4
17	75.8	0.34	107.37	0.08	56.1	132.0	59.8	0.40	119.66	0.10	203.2	2.9	109.7
18	79.8	0.37	103.36	0.07	50.9	130.8	59.9	0.44	108.84	0.08	227.4	2.6	92.7
19	77.6	0.35	114.73	0.07	36.3	113.9	59.9	0.41	119.37	0.08	208.8	2.9	95.7
20	78.4	0.36	128.68	0.07	34.3	112.7	59.9	0.43	135.04	0.08	219.6	3.2	92.6
21	75.8	0.37	135.20	0.08	27.7	103.5	59.8	0.43	138.43	0.09	221.2	3.3	107.5
22	80.4	0.37	151.16	0.08	23.2	103.6	59.9	0.44	152.79	0.09	230.1	3.7	102.8
23	67.8	0.39	156.85	0.07	14.4	82.2	59.9	0.43	164.26	0.08	221.8	3.9	92.6
24	65.3	0.43	167.85	0.07	10.8	76.1	59.9	0.46	173.23	0.08	238.9	4.1	88.2
25	67.7	0.44	158.83	0.07	8.0	75.8	59.9	0.48	160.99	0.08	248.5	3.9	90.7
26	59.2	0.49	161.29	0.08	5.5	64.8	59.9	0.48	160.91	0.08	251.3	3.9	91.9
27	28.4	0.56	187.44	0.08	0.7	29.1	59.9	0.35	133.30	0.06	181.8	3.2	71.2
28							59.9	0.15	72.89	0.04	72.5	1.7	45.7
29							60.0	0.15	72.89	0.04	72.7	1.7	45.8
30							59.9	0.15	72.89	0.04	72.5	1.7	45.7
31							59.9	0.15	72.89	0.04	72.5	1.7	45.7
32							42.7	0.15	72.89	0.04	51.7	1.2	32.6
33													
Total	1,758.5	0.44	121.40	0.09	2,784.5	4,543.0	1,758.5	0.44	121.40	0.09	6,573.1	85.4	3,052.0
Year -3 to -1	17.4	0.36	102.82	0.12	240.1	257.5	0.0	0.00	0.00	0.00	0.0	0.0	0.0
Year 1 to 6	247.6	0.67	140.13	0.14	1,183.6	1,431.2	229.4	0.72	144.19	0.14	1,362.0	13.2	637.8
Year 7 to 27	1,493.4	0.40	118.51	0.08	1,360.8	2,854.3	1,246.9	0.45	128.19	0.09	4,869.3	63.9	2,198.6
Subtotal	1,741.1	0.44	121.59	0.09	2,544.4	4,285.5	1,476.3	0.50	130.67	0.10	6,231.3	77.2	2,836.4
Year 28 to 32	0.0	0.00	0.00	0.00	0.0	0.0	282.2	0.15	72.89	0.04	341.8	8.2	215.6
Total	1,758.5	0.44	121.40	0.09	2,784.5	4,543.0	1,758.5	0.44	121.40	0.09	6,573.1	85.4	3,052.0

Figure 2-3 Chart of annual mining volumes

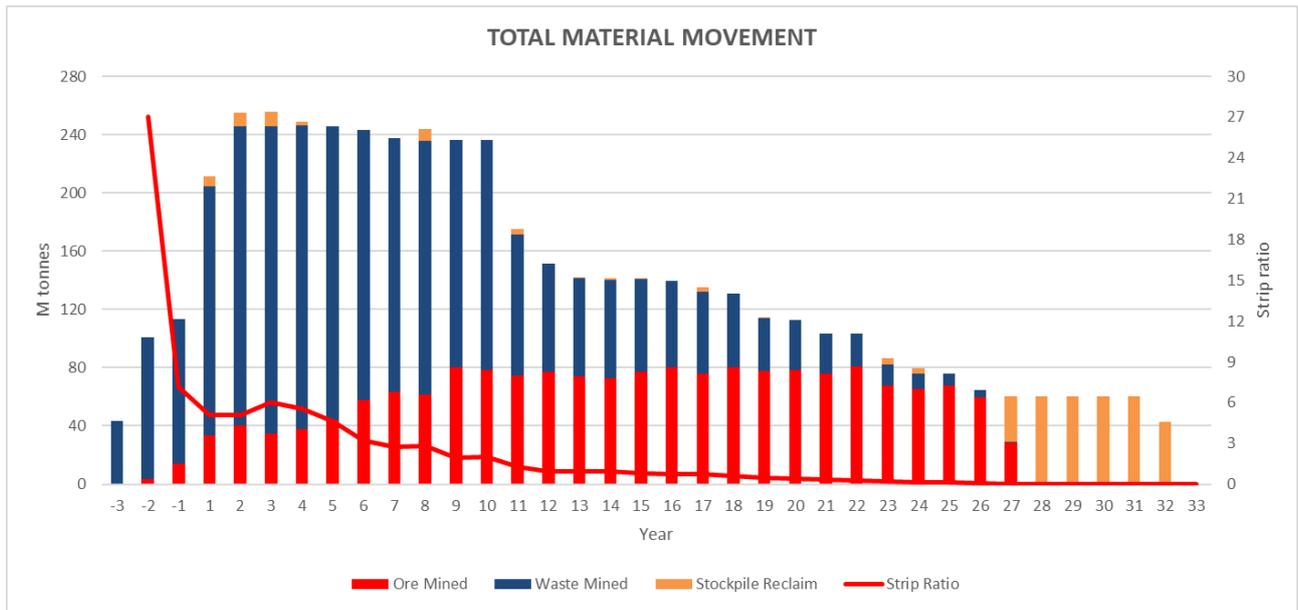


Figure 2-4 Chart of annual plant feed tonnage and grade profile

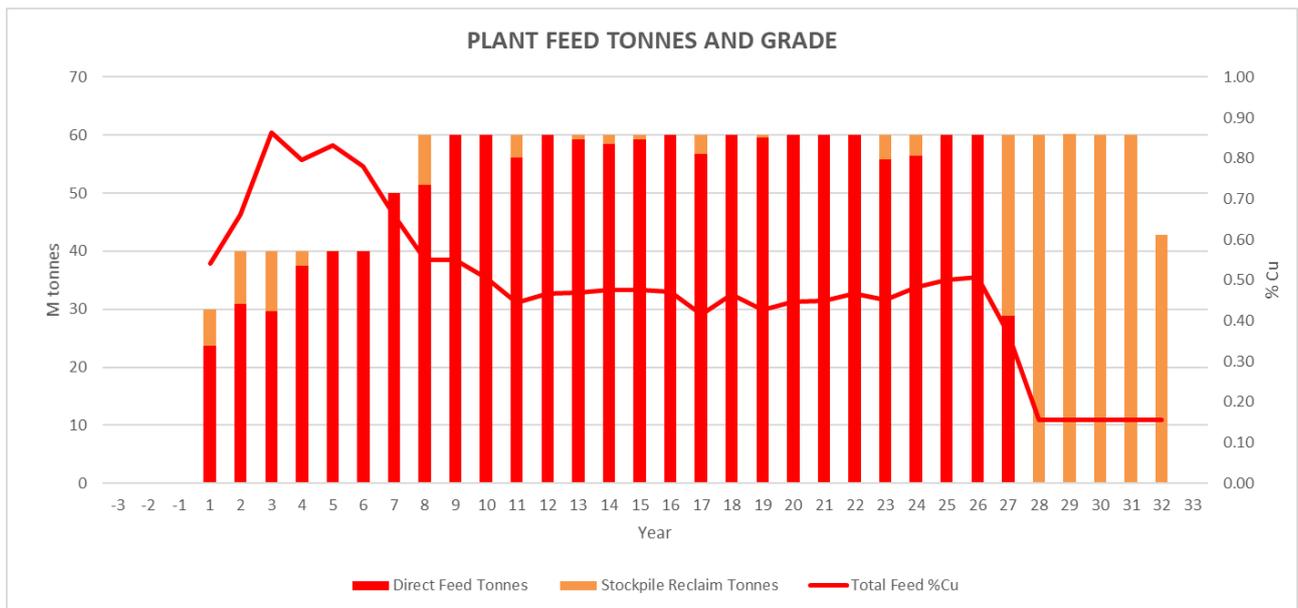
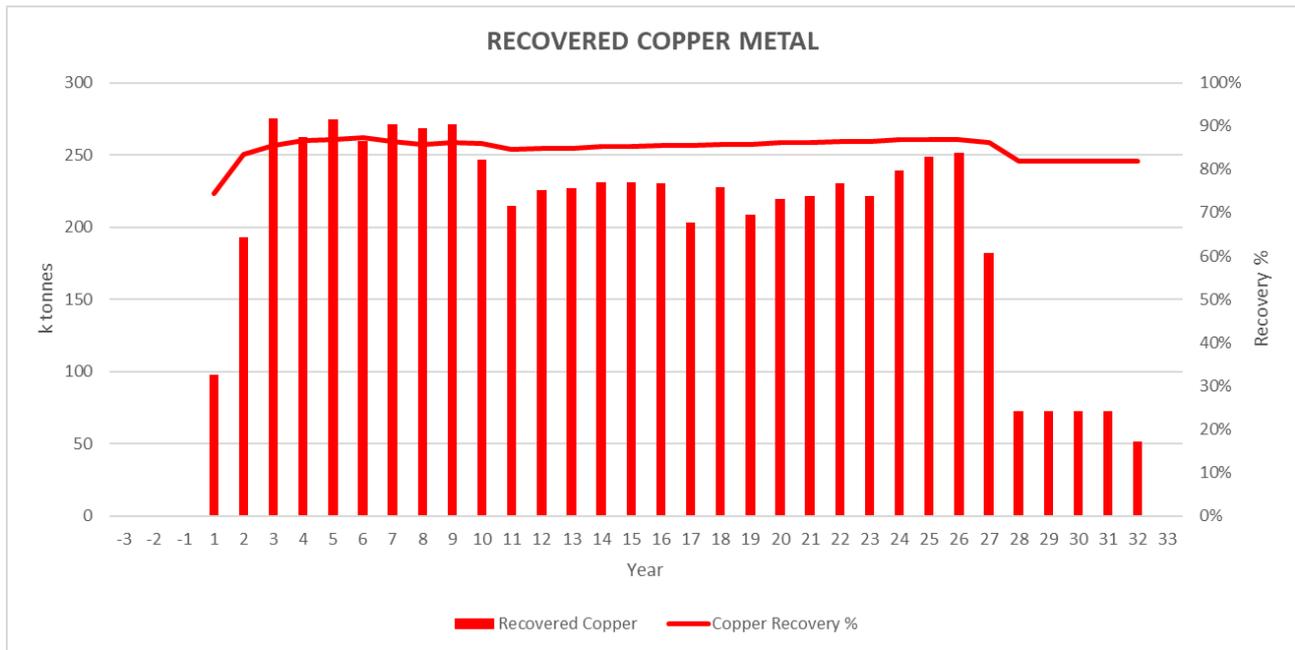


Figure 2-5 Chart of annual recovered copper metal and plant recovery profile



2.17 Capital and operating cost estimates

A capital cost estimate was produced by Ausenco (2013) for the PEA report, to a stated level of accuracy of between minus 25% and plus 35%. This comprehensive itemisation was reviewed by the Company, benchmarked where possible and adjusted accordingly. In particular, new information was included for such as the initial mining pre-strip and mining equipment purchase costs, power supply/transmission costs, water supply costs and rail infrastructure costs.

The updated capital cost estimate, at this stage of the Project engineering phase (Q3 2020) is:

- total initial capital spend over a three year construction phase (including an average 15% contingency) of \$3,274.8 M, split between:
 - \$2,636.7 M of direct costs
 - \$638.1 M of indirect costs
- total expansion capital spend (including an average 15% contingency) of \$308.1 M, split between
 - \$223.9 M of direct costs
 - \$84.2 M of indirect costs

The indicative order of accuracy of the updated capital cost estimates is considered to be now in the range of minus 15% and plus 15%. Substantial items totalling approximately 85% of the itemised capital costs have benefited from either first principles estimates and material take-offs, or are based on actual costs incurred in the construction of the Company’s Cobre Panamá project. Contingency provisions on the itemised costs vary from 0% to 20%, with an overall average of 11%.

General and administration (G&A) operating costs, process operating costs and metal costs (i.e. product transport, refining charges and royalties) as set out in the PEA report (Ausenco, 2013) were reviewed, benchmarked and updated. Mining and process operating costs were estimated from first principles.

The overall average unit operating costs are:

- mining ore and waste = \$1.69/t

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- ore mining, excluding the pre-strip period = \$1.82/t
- waste mining, excluding the pre-strip period = \$1.45/t
- stockpile reclaim = \$0.74/t reclaimed
- processing = \$4.69/t processed
- rail load-out infrastructure and water supply tariff = \$0.08/t processed

The cashflow model metal costs (including treatment charges, refining charges, plus 3% provincial royalties and 1.5% third party royalties) are:

- copper = \$0.52/lb (including a net royalty charge equating to \$0.09/lb)
- molybdenum = \$0.56/lb (including a net royalty charge equating to \$0.38/lb)
- gold = \$56.01/oz (including a net royalty charge equating to \$51.42/oz)

2.18 Economic analysis

An economic analysis in the form of a basic cashflow model was produced to support the Mineral Reserve estimate, and in order to demonstrate an overall positive cashflow for mining and processing over the life of the Project. The initial development capital and expansion capital costs are included in the analysis for completeness, and the model is summarised in Table 2-4. A more detailed cashflow table is provided in Item 22.

The revenues in this cashflow model are calculated from the following metal prices:

- copper = \$3.00/lb (\$6,614/t)
- molybdenum = \$12.00/lb (\$26,455/t)
- gold = \$1,500/oz

On a pre-tax basis, the Project is cashflow positive from Year 2 and payback on the initial development capital is in Year 9. The undiscounted cashflow for the Mineral Reserve production schedule is \$17,306.3 M, with an NPV reflecting an 8% discount rate equal to \$3,428.8 M. The internal rate of return is 17.4%.

After adopting notional depreciation schedules and a flat 25% corporate tax rate, the estimated tax payable for the Project is \$4,331.3 M. Under these circumstances, the NPV reflecting an 8% discount rate is equal to \$2,361.2 M, and the internal rate of return is 15.3%.

A Monte Carlo simulation on the Mineral Reserve cashflow model was carried out to assess the discounted value of the Project; 10,000 iterations were simulated for a number of cashflow variables, over a specified range of values for each. Assuming an 8% discount rate, the simulation indicated a 70% probability that the NPV would be in excess of \$3,000 M and a 72% probability that the IRR would be in excess of 16%, on a pre-tax basis. An additional simulation indicated a 70% probability that the NPV would be in excess of \$2,000 M and a 72% probability that the IRR would be in excess of 14%, on a post-tax basis.

2.19 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. The ESIA was submitted to the authorities in February 2019. The Company has subsequently filed documents responding to an ESIA review and observations made by the provincial Secretariat of Mining.

Social Capital Group has assisted the Company with socioeconomic studies including a socioeconomic baseline for the ESIA, as well as an identification of stakeholders. Related ESIA documents are in preparation for approval of the 345 kV transmission line, and separately for the proposed road access diversion.

Table 2-4 Mineral Reserve cashflow model summary

PHYSICALS	UNITS	TOTAL	Yr -3 to -1	Yr 1 to 10	Yr 11 to 20	Yr 21 to 30	Post Yr 30
MINING (AFTER MINING DILUTION & RECOVERY)							
Ore mined direct to plant	Mt	1,390.4	0.0	422.0	588.1	380.3	0.0
Ore mined to stockpile	Mt	368.1	17.4	106.7	178.7	65.3	0.0
Ore reclaimed from stockpile	Mt	368.1	0.0	36.8	10.4	218.4	102.5
Waste mined to dump	Mt	2,784.5	240.1	1,846.8	607.2	90.4	0.0
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)							
Total direct feed	Mt	1,390.4	0.0	422.0	588.1	380.3	0.0
	% Cu	0.50	0.00	0.63	0.44	0.47	0.00
	ppm Mo	131.59	0.00	141.43	106.04	160.17	0.00
	g/t Au	0.10	0.00	0.13	0.09	0.08	0.00
Total reclaim feed	Mt	368.1	0.0	36.8	10.4	218.4	102.5
	% Cu	0.20	0.00	0.56	0.32	0.15	0.15
	ppm Mo	82.93	0.00	133.93	102.11	78.13	72.89
	g/t Au	0.05	0.00	0.15	0.11	0.04	0.04
Total plant feed	Mt	1,758.5	0.0	480.0	598.5	598.7	81.3
	% Cu	0.44	0.00	0.61	0.44	0.34	0.15
	ppm Mo	121.40	0.00	137.62	109.70	126.63	73.28
	g/t Au	0.09	0.00	0.13	0.09	0.07	0.04
Cu insitu	kt	7,734.7	0.0	2,878.6	2,598.7	2,105.9	151.4
Mo insitu	kt	213.5	0.0	64.6	63.4	78.0	7.5
Au insitu	k(t)oz	5,086.7	0.0	1,994.1	1,658.6	1,303.5	130.5
AVERAGE RECOVERIES							
Copper recovery	%	85.5%	0.0%	85.4%	85.4%	86.0%	82.0%
Copper ramp-up factor	%	99.4%	0.0%	98.4%	100.0%	100.0%	100.0%
Adjusted copper recovery	%	85.0%	0.0%	84.0%	85.4%	86.0%	82.0%
Molybdenum recovery	%	40.0%	0.0%	40.0%	40.0%	40.0%	40.0%
Gold recovery	%	60.0%	0.0%	60.0%	60.0%	60.0%	60.0%
METAL RECOVERED							
Unadjusted Cu recovered	kt	6,612.7	0.0	2,459.0	2,218.4	1,811.2	124.2
Adjusted Cu recovered	kt	6,573.1	0.0	2,419.4	2,218.4	1,811.2	124.2
Mo recovered	kt	85.4	0.0	25.8	25.4	31.2	3.0
Au recovered	k(t)oz	3,052.0	0.0	1,196.5	995.1	782.1	78.3
CONCENTRATE PRODUCED							
Cu concentrate	Mwmt	28.5	0.0	10.5	9.6	7.8	0.6
Cu concentrate grade	%	25.3%	0.0%	25.3%	25.7%	25.1%	9.1%
Mo concentrate	Mwmt	0.20	0.00	0.06	0.06	0.07	0.01
Mo concentrate grade	%	47.0%	0.0%	47.0%	47.0%	47.0%	18.8%
CASH FLOWS							
PAYABILITY							
Cu	%	96.2%	0.0%	96.2%	96.2%	96.2%	96.2%
Mo	%	86.0%	0.0%	86.0%	86.0%	86.0%	86.0%
Au	%	90.0%	0.0%	90.0%	90.0%	90.0%	90.0%
Payable metal recovered							
Cu	kt	6,320.3	0.0	2,326.3	2,133.1	1,741.5	119.4
Mo	kt	73.4	0.0	22.2	21.8	26.8	2.6
Au	koz	2,746.8	0.0	1,076.8	895.6	703.9	70.5
GROSS REVENUE							
Metal prices							
Cu	\$/lb	\$3.00	\$3.00	\$3.00	\$3.00	\$3.00	\$3.00
Mo	\$/lb	\$12.00	\$12.00	\$12.00	\$12.00	\$12.00	\$12.00
Au	\$/oz	\$1,500	\$1,500	\$1,500	\$1,500	\$1,500	\$1,500
Revenue after payability							
copper revenue	\$M	\$41,801.6	\$0.0	\$15,386.0	\$14,107.8	\$11,518.2	\$789.6
molybdenum revenue	\$M	\$1,942.9	\$0.0	\$588.1	\$577.2	\$709.6	\$68.0
gold revenue	\$M	\$4,120.2	\$0.0	\$1,615.2	\$1,343.4	\$1,055.8	\$105.7
Total revenue	\$M	\$47,864.7	\$0.0	\$17,589.4	\$16,028.4	\$13,283.7	\$963.3
CAPITAL COSTS							
Mining	\$M	\$730.7	\$730.7	\$0.0	\$0.0	\$0.0	\$0.0
Processing	\$M	\$1,108.8	\$924.5	\$184.3	\$0.0	\$0.0	\$0.0
Infrastructure	\$M	\$966.7	\$927.1	\$39.6	\$0.0	\$0.0	\$0.0
Ancillaries and other	\$M	\$776.8	\$692.6	\$84.2	\$0.0	\$0.0	\$0.0
Closure costs	\$M	\$34.4	\$0.0	\$0.0	\$0.0	\$12.3	\$22.1
Total capital costs	\$M	\$3,617.3	\$3,274.8	\$308.1	\$0.0	\$12.3	\$22.1
SUSTAINING COSTS							
Mining	\$M	\$875.1	\$0.0	\$526.7	\$313.7	\$34.7	\$0.0
TSF	\$M	\$56.5	\$0.0	\$0.0	\$18.8	\$31.4	\$6.3
Processing and infrastructure	\$M	\$372.5	\$0.0	\$121.2	\$157.0	\$94.3	\$0.0
Total sustaining costs	\$M	\$1,304.1	\$0.0	\$648.0	\$489.5	\$160.4	\$6.3
OPERATING COSTS							
Ore mining	\$M	\$3,215.0	\$0.0	\$883.8	\$1,343.5	\$987.6	\$0.0
Waste mining	\$M	\$4,033.8	\$0.0	\$2,846.2	\$991.4	\$196.1	\$0.0
Stockpile reclaim	\$M	\$273.6	\$0.0	\$25.6	\$7.8	\$169.5	\$70.8
Processing and other	\$M	\$8,380.5	\$0.0	\$2,186.7	\$2,852.2	\$2,853.0	\$488.6
General and administration	\$M	\$1,846.5	\$0.0	\$481.8	\$628.4	\$628.6	\$107.6
Total operating costs	\$M	\$17,749.3	\$0.0	\$6,424.1	\$5,823.3	\$4,834.9	\$667.0
METAL COSTS							
TCRCs	\$M	\$6,325.3	\$0.0	\$2,326.3	\$2,127.4	\$1,739.4	\$132.1
Royalties	\$M	\$1,562.5	\$0.0	\$606.8	\$521.1	\$415.0	\$19.5
Total metal costs	\$M	\$7,887.7	\$0.0	\$2,933.1	\$2,648.6	\$2,154.4	\$151.6
MINERAL RESERVE CASHFLOW (PRE-TAX)							
Undiscounted	\$M	\$17,306.3	-\$3,274.8	\$7,276.1	\$7,067.0	\$6,121.7	\$116.3
NPV ₁₀ (indicative)	\$M	\$2,212.5					
NPV ₈ (indicative)	\$M	\$3,428.8					
IRR	%	17.4%					
MINERAL RESERVE CASHFLOW (POST-TAX)							
Taxable income	\$M	\$17,325.1	\$0.0	\$4,058.7	\$7,052.4	\$6,082.0	\$132.1
Tax paid	\$M	\$4,331.3	\$0.0	\$1,014.7	\$1,763.1	\$1,520.5	\$33.0
Undiscounted after tax cashflow	\$M	\$12,993.8	-\$3,274.8	\$6,261.4	\$5,303.9	\$4,601.2	\$83.3
NPV ₁₀ (indicative)	\$M	\$1,420.0					
NPV ₈ (indicative)	\$M	\$2,361.2					
IRR	%	15.3%					

2.20 Conclusions and recommendations

2.20.1 Mineral Resource modelling and estimation

The Mineral Resource estimate was completed using quality data and appropriate estimation methods. Estimation domains were based on an updated 3D geological model characterising weathering, alteration, lithology and dominant mineralisation style. Although estimates validate well at the deposit scale, the current drill hole spacing does not always support a locally accurate estimate at a mining scale. Mineral Resource estimates compare well with previous estimates and benefit from improved detail to domains of mineralisation that focus on styles of mineralisation as well as similarities in metallurgical processing.

2.20.2 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 phase and ultimate open pit designs, production scheduling and Mineral Reserve estimation. For the envisaged scale of operations and the size and extents of the proposed open pit, an ultra-class scale of primary mining equipment is considered to be most suited.

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.

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 Taca Taca. A recommended programme of further drilling and the specific location of drillholes has been provided by geotechnical consultants. The consultants also recommend that the geotechnical investigations be integrated with ongoing hydrogeological modelling, for the sake of analysing drained pit slope conditions.

2.20.3 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:

- highlighted the need for secondary crushing
- 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 using brine in rougher flotation and brackish water in cleaner flotation

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- 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:

- 85% Cu recovery to a concentrate grade of 25.3% Cu
- 40% Mo recovery to a concentrate of 47% to 50% Mo
- 60% Au recovery at a grade of 4.5 g/t in the copper concentrate

2.20.4 Processing

The processing flowsheet follows that of typical porphyry copper concentrators common in Chile and Peru, but with three significant differences:

1. Controlled potential sulphidisation (CPS) flotation will be employed, using NaHS to sulphidise any oxidised or tarnished minerals to assist in flotation.
2. The water used in milling and rougher flotation will be brine solutions from the Salar de Arizaro.
3. Rougher concentrates will be dewatered, and re-diluted with brackish or fresh water from local bores, prior to regrind and cleaner flotation.

After the commissioning year, the processing plant will have an initial capacity of 40 Mtpa, rising to 60 Mtpa in Year 8. The 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 will follow standard practice, enhanced through experience gained by the Company in the design, construction, and operation of other similar sized projects.

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

As the engineering phase proceeds, refinements to the processing flowsheet and design will be considered. In particular, 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.

2.20.5 Water supply

Phase III water supply investigations for the Project were continued into early 2020 before being halted due to the COVID-19 pandemic. These investigations have the aim of further testing, modelling and confirmation of the fresh water supply sources for the Project.

The results of investigations that were able to be completed show that there is a high confidence in three of the identified watersheds, i.e. at Valle de Arizaro, at Valle de las Burras, and at Socompa. A fourth supply source has been identified at Valle de Chaschas and remains to be investigated in further detail. A borefield on the nearby Salar de Arizaro appears to be suitable as an infinite source of brine water for processing. Further confirmatory drilling, pump testing, water volume estimation and aquifer modelling will resume during the Project engineering phase to firm-up on the supply estimates.

Average and peak water supply calculations have been produced to support the required fresh and brine water demand for the site. From these calculations it appears that the four identified fresh water supply basins have a combined estimated yield in excess of that required for process water make-up.

2.20.6 Infrastructure

Preferred locations have been identified and designs produced for the Project waste dump and TSF. The concepts for these are considered to be suitable for the Project at this stage of the engineering phase, although requiring confirmatory geotechnical, engineering and seismicity studies.

After identification of preferred alternatives, the Project power transmission line and road access engineering did progress beyond a preliminary scoping level of assessment. Separate ESIA studies and documentation are underway and specialist consultants engaged to provide the required engineering detail.

In terms of logistics infrastructure, and also following identification of preferred alternatives, the engineering of the railway line is now being addressed in some detail. Commercial negotiations will need to proceed with the several owners of respective sections of the preferred rail corridor and also with potential owners/operators of coastal ports at Mejillones Bay, Chile.

2.20.7 Cost estimation and economic outcomes

Mine and process operating costs, plus general and administration costs (G&A) have been estimated from first principles. Metal costs (i.e. including transport and refining charges (TCRCs)) have been advised by the Company's own metals marketing group and by a specialist rail transport consultant. The process plant and related infrastructure capital costs have been estimated by means of benchmarking and factoring, supplemented with consultant/vendor estimates. The order of accuracy of the capital cost estimates reflects the adoption of contingency factors of up to 20%.

The Mineral Reserve pre-tax cashflow model has been tested with a Monte Carlo simulation involving 10,000 iterations through the material cashflow variables.

2.20.8 Further work recommendations

Phase III water supply investigations

At this stage of engineering, it appears that there should be more than adequate supply volumes for the quantity of fresh water required for processing. Nevertheless, the programme of work that was commenced in 2019 and halted in 2020 should be resumed in order to confirm the projected sustainability of fresh water supply from the four identified, distant gravel basins.

Whilst water supply permitting work is in progress for the proposed fresh water borefields, it is recommended that any applicable brine water abstraction permit requirements also be followed-up.

Mineral Resource drilling, sampling and analysis

A programme of further Mineral Resource related investigative work has been devised, focusing on resource infill drilling, extension drilling and sterilisation drilling. Specifically, the programme includes:

- Infill drilling across the initial Phase 1 and Phase 1a starter pit horizons, and extending out across the leach cap to further test the extents and grade of auriferous mineralisation.
- Infill and extensional drilling at the Little Taca deposit, which is currently ill-defined for inclusion in the Mineral Reserve inventory. The location of Little Taca and its possible inclusion in the future Reserve would have a bearing on the design of the existing pit and the adjacent processing plant.
- Extensional drilling in the vicinity of the TK2 fault and in the region of the proposed Project camp.
- Sterilisation drilling of the plant site, camp site and other infrastructure sites.

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It is recommended that future updates to the Mineral Resource modelling include the refinement of estimation domain groupings and the incorporation of higher resolution 3D structural, alteration, and material-type models.

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 Taca Taca. A recommended programme of further drilling and the specific location of drillholes has been provided by geotechnical consultants. The consultants also recommend that the geotechnical investigations be integrated with ongoing hydrogeological modelling, for the sake of ensuring drained pit slope conditions.

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.
- Confirmation of NaHS requirements for sulphidisation in rougher flotation, with estimated improvements in recovery by means of CPS.
- 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).
- 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:

- Primary crusher location, i.e., in pit or on surface.
- Dewatering of rougher concentrates to remove brine and re-dilute with brackish water. Including looking at filtration in place of thickening.
- Re-grind circuit power requirements and mill sizing.
- Configuration of the cleaner flotation circuit, in terms of two parallel trains of first cleaners, or one train of larger cells. Investigate the integration of flotation columns and Jameson cells into the circuit.
- 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 (when that circuit commences operation).
- Copper concentrate filter location options, i.e., at the main concentrator or the rail load-out facility.
- 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), and materials of construction to minimise scaling and corrosion from brine.

These tasks may be an initial scope of engineering and drafting work for third party engineering firm(s).

Infrastructure engineering

In conjunction with an initial scope of process engineering work for a third party engineering firm(s), there are a number of infrastructure aspects for the Project that ought to advance beyond the current stage. These aspects are as follows:

- A preferred connection to the national supply grid and the route of a power transmission line into the site has been identified, and is the subject of a forthcoming detailed ESIA. Much discussion and preliminary negotiations with consultants and government agencies have already taken place. Whilst the power supply requirements have been estimated, a detailed itemisation and specification of power reticulation requirements across the Project site is yet to be commenced.
- Modelling of the tailings deposition over time has been undertaken, with a favourable outcome on the timing for and requirements for the starter embankment. However, some optimisation work is recommended for the tailings delivery and spigotting arrangement, with the objective of prolonging the life of the existing railway line skirting the western edge of the TSF, should that be feasible.
- There are several elements of the site layout plan that require review, optimisation and possible enhancement. The conceptual layout of the process plant is based on the Sentinel configuration and as such, should be designed to suit the prevailing topography at the site selected. Fundamentally, and should it be necessary, the selected location might be discarded in favour of some other convenient site. This review and design should be carried out in conjunction with the civil geotechnical programme. A detailed LIDAR survey of the mine site and the proposed plant site has already been completed.
- Other conceptual infrastructure elements requiring further engineering include the train load-out facility (especially the means by which concentrate will be delivered from the plant site and transferred into railway cars), fuel delivery/storage, mine services area/layout, warehouses, workshops and administration buildings.
- Additionally, there is the matter of the camp site for which there is a conceptual location identified. The site has been reconnoitred and an anemometer has been installed at the site to enable an evaluation of wind conditions.
- A waste landfill study has been completed and a preferred site selected which is possibly too close to the conceptual camp location. It is recommended that the landfill options be reviewed.

ITEM 3 INTRODUCTION

3.1 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 Mineral Resource and Mineral Reserve estimates for the property, completed subsequent to its acquisition from previous owners, and to provide a commentary on the Project development status.

The Initial Technical Report filed in November 2020 is amended and restated herein with the inclusion of some additional clarifications and confirmatory information.

3.2 Terms of reference

This Technical Report has been written to comply with the reporting requirements of the Canadian National Instrument (NI) 43-101 guidelines: 'Standards of Disclosure for Mineral Properties' of April 2011 (the Instrument) and with the 'Australasian Code for Reporting of Mineral Resources and Ore Reserves' of December 2012 (the 2012 JORC Code) as produced by the Joint Ore Reserves Committee of the Australasian Institute of Mining and Metallurgy, Australian Institute of Geoscientists and Minerals Council of Australia (JORC).

The effective date for the Mineral Resource and Mineral Reserve estimates is the 30th October 2020.

3.3 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 3-1 QP details

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

3.4 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).

3.5 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 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.

3.6 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.

Where not explained in the text of this report, specific terms and definitions are as listed in Table 3-2.

Table 3-2 Terms and definitions

Term	Definition	Term	Definition
µm, mm, cm, m, km	microns, millimetres, centimetres, metres, kilometres	Mtpa	million tonnes per annum
bcm	bank cubic metres	MO, LG, MG, HG	marginal ore, low grade, medium grade, high grade
bn	bornite	NPV	net present value
cpy	chalcopyrite	oz	ounces
csv	comma separated value	P ₈₀	80% passing
g, kg	grams, kilograms	pH	potential of hydrogen
g/t, kg/t	grams per tonne, kilograms per tonne	py	pyrite
ha	hectares	Q1, Q2, Q3, Q4	quarter 1 to 4
IRR	internal rate of return	t, kt, Mt	tonnes, thousands of tonnes, millions of tonnes
kWh/t	kilowatt hours per tonne	tpa	tonnes per annum
lb	pounds	tpd	tonnes per day
LOM	life of mine	tph	tonnes per hour
m/s	metres per second	V, kV	volts, kilovolts
Ma	mega annum (million years)	W, MW	watts, megawatts
masl	metres above sea level	WGS	Western Geodetic System
mE, mN	coordinates: metres East, metres North	L/s	Litres per second

ITEM 4 RELIANCE ON OTHER EXPERTS

The authors of this Technical Report do not disclaim any responsibility for the content contained herein, with the exception of certain information included in the Item 22 Economic Analysis. This information, provided on the 16th February 2021 by the Company's internal taxation advisers, relates to the applicable corporate tax rate in Argentina and to the estimated Project taxable income arising from capital depreciation and taxation calculations.

The authors of this Technical Report have relied on this information for the purposes of the Project economic analysis in Item 22.

ITEM 5 PROPERTY DESCRIPTION, LOCATION AND TENURE

5.1 Project description

The proposed Taca Taca Project involves the open pit mining and flotation processing of copper ore for a period of 32 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 salt lake (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.

5.2 Proposed Project components

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

- a multi-phased open pit mine extending to an ultimate depth in excess of 700 m
- surface stockpiles for marginal ore, and separately for auriferous material not in the Mineral Reserve, but which could be economically processed in the future
- a mining waste dump, located to the east of the open pit, on the surface of the adjacent salar (the Salar de Arizaro)
- a processing plant site which is located adjacent to the open pit, in an area of relatively flat topography sheltered from the prevailing wind direction
- a concentrator for the processing of copper mineralisation by flotation, with primary recovery of gold (and silver⁴) into the concentrate
- separation of copper and molybdenum concentrates
- ramp-up of the processing rate from 30 Mt in the first year, to 40 Mtpa for the next five years, followed by a 50 Mtpa rate for one year, and then to an eventual rate of 60 Mtpa from Year 8
- a tailings storage facility located within an embayment of the Salar de Arizaro
- process water storage ponds
- surface haulage and internal access roads
- borefields for the supply of fresh and saline water
- overland pipelines between the concentrator and the tailings storage facility, and between borefields and the plant site
- surface haulage and internal access roads
- mine services workshops and equipment wash-down facilities
- construction offices, mine administration and accommodation facilities
- storage space and a rail load-out facility for copper and molybdenum concentrate products
- parts and consumables, reagent and explosives handling and storage facilities
- as additional infrastructure, there are roads for transporting supplies into the Project site, a railway for transporting concentrates and supplies, and a high voltage electric transmission line

⁴ Silver (Ag) is considered immaterial to the economic value of the Project and hence Ag grades have not been reported in the Project Mineral Resource and Reserve estimates, nor accounted for in the Mineral Reserve economic analysis.

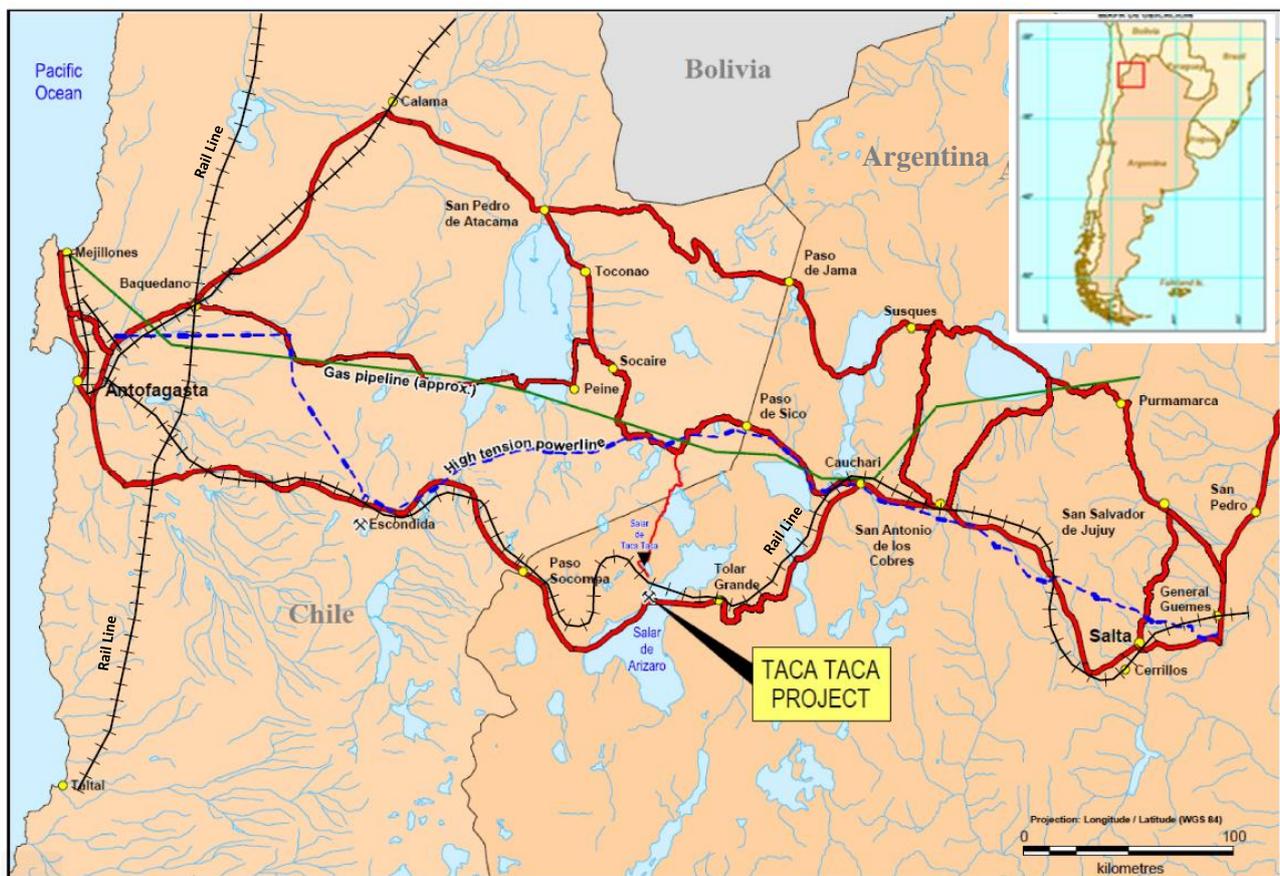
As the Project continues through the engineering phase and into the development and operations phase, production aspects may change and components reviewed and varied to suit current circumstances and new information.

5.3 Project location

The Taca Taca Project is located 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 5-1). The nearest population centre is the village of Tolar Grande (population of approximately 150), located 35 km east of the Project site.

The UTM co-ordinates at the centre of the site are 7,283,500 mN and 2,628,000 mE.

Figure 5-1 Taca Taca Project location



5.4 Project ownership

The Taca Taca Project (the Project) is 100% owned by First Quantum Minerals Ltd (FQM or the Company) 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.

5.5 Mineral tenure, rights, payment agreements and encumbrances

The Company holds a significant package of mining rights in the region, consisting of 75 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 third party groups, though these are not over commercially material areas of the known deposit. The other concessions are held 100% by the Company.

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Under the Mining Code, these mining concessions vest the Company with property title over the mine, including the right to explore and exploit in the concessions area. The mining concessions are granted in perpetuity and are not subject to a validity term as long as 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 \$50k per annum.

Figure 4-2 shows the location of the Project relative to the 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.

Subject to their stated clarifications, limitations and assumptions, the Company's legal counsel (Bruchou, 2019) provided the following opinion on mineral tenure, as at December 2019:

1. CASA owns and holds good, valid, marketable and beneficial title (for title as defined by the Argentine Mining Code) to all of the mining properties listed in Table 4-1, except for two which are each held with a 50% share.
2. The mining properties listed in Table 4-1 are valid and in good standing.
3. Applicable mining exploitation fees have been paid until the first semester 2019.
4. To the best of their knowledge, there are no oppositions filed or registered against the mining properties that may remain pending of resolution and no restrictions have been recorded with respect to any of the mining properties.
5. To the best of their knowledge, there are no records of attachments, mortgages, encumbrances or any other security interest with respect to any of the mining properties.
6. CASA has the right to receive and deal with 100% of the copper, gold any other mineral production from the mining properties except for those listed in Table 4-2 for which CASA's title is restricted to a 50% share.

In Figure 4-2, the thirteen Taca Taca Mining Group concessions are shown in the inset panel. The 50% joint venture concessions are shown with a light purple coloured perimeter.

Table 5-1 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 5-2 Details of Project mineral concessions, additional concessions

Number	Concession	Number	Area (ha)	Owner	Royalty
	Additional concessions				
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 -	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	20821 - 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 Escondida	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%	
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%	
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	Grupo Minero Taca Taca	18690 - 2007	2,557.6	CASA 100%	
	Total area		97,295.6		

5.6 Royalties

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 also process operating costs.

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 two individuals.

5.7 Export levy

According to the regulations in force as at October 2020, the application of a minerals concentrate export levy will expire at the end of 2021. On this basis, no such levy is included with royalties in the estimation of Project metal costs.

5.8 Environmental liabilities

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

5.9 Permits that must be acquired

The primary approval required for the development of the Taca Taca Project is the approval of the 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 to the authorities in February 2019. A response to the submission was received from the Secretariat of Mining at the end of Q3 2019, and this included 62 observations (including requests for clarification or more information). Some of the required additional information will only be available once the Project engineering is more advanced. A compiled document with responses that were able to be provided at this stage of the Project was submitted to the authorities in February 2020, and is currently under review by the Secretariat. The Project ESIA approval is expected in 2021.

According to Argentinian law, after final submission and approval, the ESIA must be updated and resubmitted to the authorities at least every two years.

Another ESIA is required to be submitted separately to the Energy Secretariat of Salta Province for the 345 kV transmission line to connect the Project to the national electrical grid. A third ESIA is required to be submitted to the Salta Road Administration for the proposed bypass road construction for the Project. The Project will also require approval from the Water Resources Secretariat of Salta Province of a concession for water supply development and use.

The two additional ESIA applications are in preparation and are expected to be submitted to the authorities in 2021. The water supply permit application will be submitted following completion of the Phase III water supply definition programme (refer to Item 24 for details).

Other administrative authorisations, detailed construction and operating permits will be required, particularly from the Municipality of Tolar Grande and various provincial authorities, during the course of development and operation of the Project.

5.10 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.

ITEM 6 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

6.1 Topography, elevation and physiography

The 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.*

6.2 Seismic conditions

The Taca Taca 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 convergence 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 Taca Taca 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 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 based on a detailed engineering risk analysis.

6.3 Vegetation

Vegetation is sparse to non-existent in the Project area. The dry puna vegetation is characterised by grasses, alpine herbs and dwarf shrubs.

6.4 Climate

The climate in the Project area is arid with summer temperatures ranging from 2°C to 22°C, and winter temperatures ranging from -3°C to 5°C (from Taca Taca weather station records for October 2011 to November 2018).

The average relative humidity is approximately 34%, with a low average annual precipitation of approximately 40 mm/year (from Taca Taca weather station records for October 2011 to November 2018).

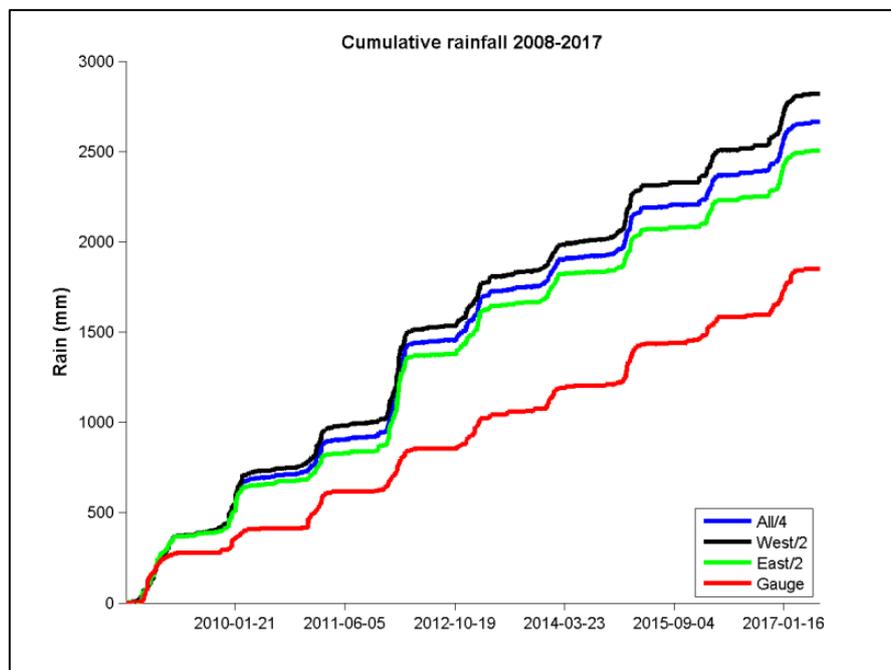
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From the available records, 70 mm of rain was recorded in 2012 and 78 mm in 2017; 3 mm was recorded in 2016. Peak months for rainfall events are January to March.

There are no weather stations capable of measuring snowfall in the region and limited local data for historical precipitation (i.e. from the Taca Taca weather station), hence a reliance on satellite data has been used by the Company to estimate long term average precipitation rates when working on Project water supply investigations. Several different satellite based estimates have been reviewed by the Company, with the CMORPH estimation technique considered to be the most reliable⁵.

This technique uses precipitation estimates that have been derived from low orbiter satellite microwave observations, and whose features are transported via spatial propagation information that is obtained entirely from geostationary satellite IR data. CMORPH is deemed to be reliable as it can be effectively correlated and calibrated with weather stations in the region (Figure 6-1; the Cafayate weather station is located near Salta). It is also most useful in providing a volumetric estimate of total precipitation (snow and rain)⁶.

Figure 6-1 Cafayate rain gauge data (red line) compared with CMORPH satellite data



6.4.1 Wind

Typical wind speeds range from 3.8 m/s to 23.2 m/s, blowing predominantly from the northwest. Figure 6-2 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

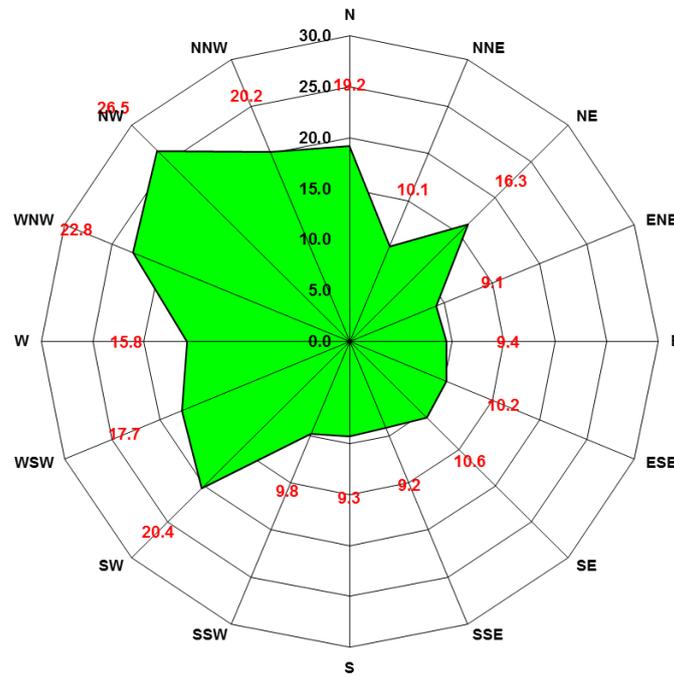
⁵ CMORPH is an acronym for the NOAA (National Oceanic and Atmospheric Administration) - CPC (National Weather Service Climate Prediction Centre) **Morphing** Technique (http://www.cpc.ncep.noaa.gov/products/janowiak/cmorph_description.html).

⁶ The CMORPH data appears to consistently over report precipitation so a calibration factor of -25% has been applied by the Company.

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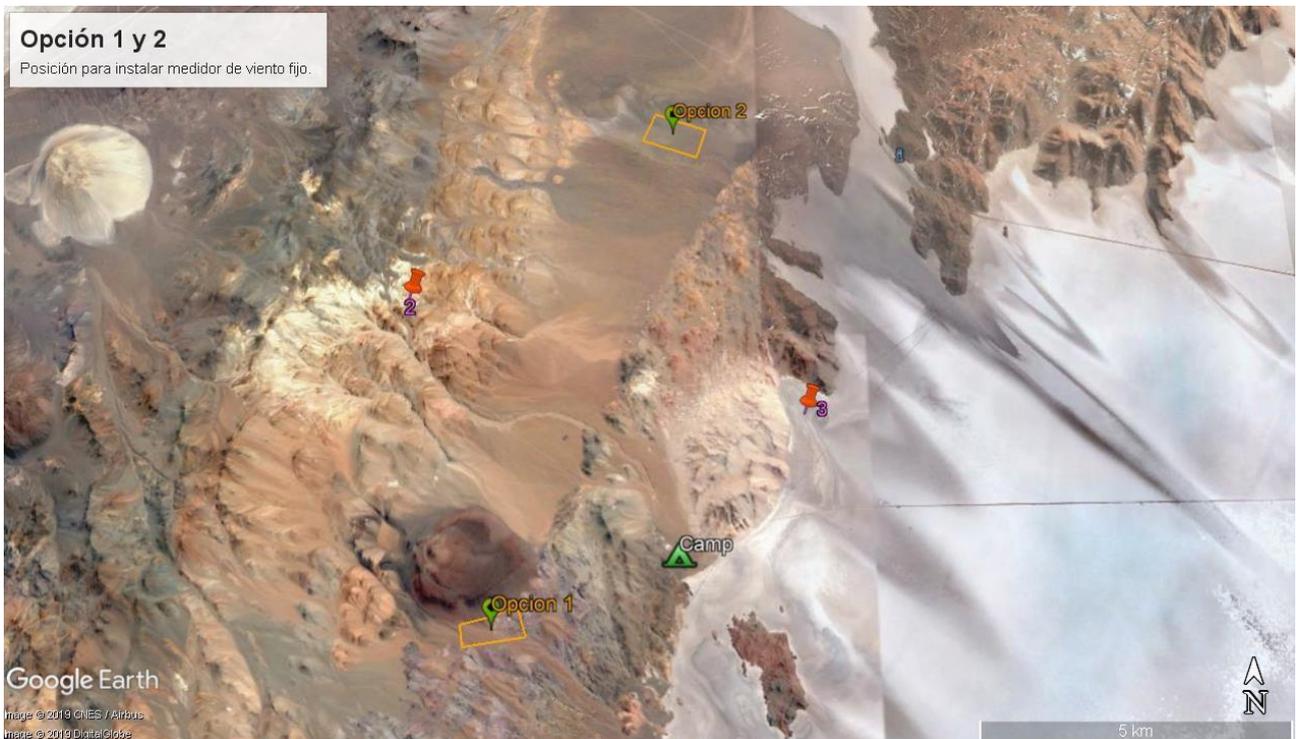
during the austral winter can range between 20 km/h and 40 km/h. The speed is less during the austral summer, although pronounced after mid-day and calming during the night.

Figure 6-2 Average wind speed (km/h) and direction, measured at the Taca Taca weather station



Two wind stations (anemometers) were installed in 2019 at potential locations for the permanent campsite. These are denoted as Options 1 and 2 on Figure 6-3. The site labelled as “Camp” is the location of the existing exploration camp.

Figure 6-3 Wind monitoring stations



6.4.2 Solar radiation

M&A (September, 2018) advise that the daily solar radiation is extreme in the Project area, averaging from 180 Watts/m² in July to 380 Watts/m² in December. During December, instantaneous solar radiation can exceed 1,000 Watts/m² at mid-day.

6.4.3 Evaporation rates

From available meteorological data, M&A (September 2018) estimated the potential open water evaporation rates as 11 mm/day in January ranging to 6 mm/day in June, with an annual average of 8.6 mm/day. The estimated evapotranspiration rates were 9 mm/day in January ranging to 4 mm/day in June, with an annual average of 7 mm/day.

6.5 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 evapotranspiration.

The Salar de Arizaro is reportedly the sixth largest salt lake in the world and the second largest in Argentina. It covers an area of about 1,600 km². 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 also in the overlying evaporate deposits (M&A, November 2018).

The process plant will be the largest water consumer for the Project. More than two thirds of its requirements will be met with brine water sourced from the Salar de Arizaro and to a lesser extent, from depressurisation of the open pit slopes. It has been assumed that the volume of saline brine water available is unlimited, and that brine water at up to 300,000 mg/L TDS (total dissolved solids) could be used for milling and rougher flotation.

Table 6-1 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 6-1 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. 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 the brine level 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.

6.6 Access to the Project by road

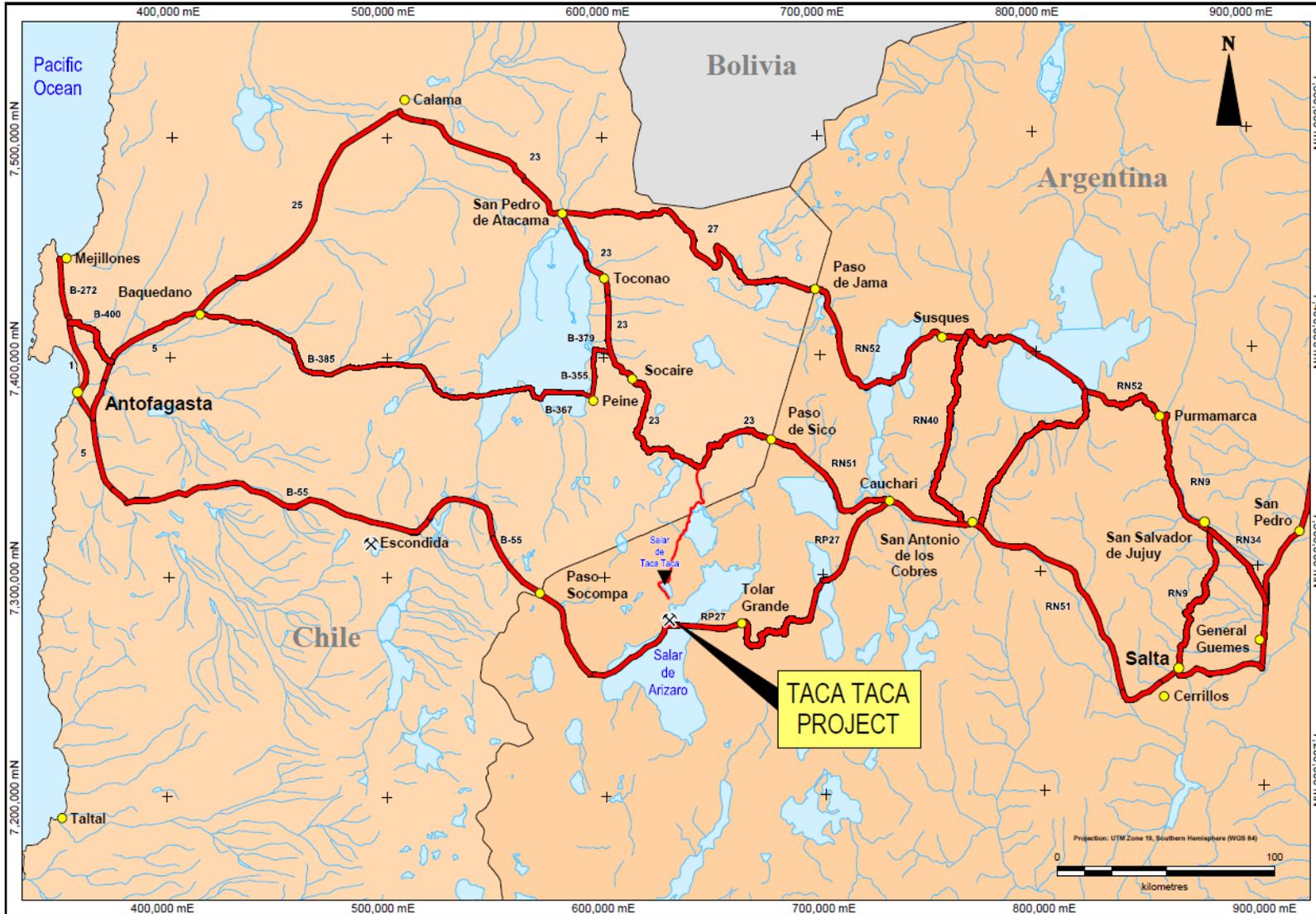
The Project site is readily accessible by road from the city of Salta, which is approximately 230 km to the east and is the nearest major population centre in Argentina. The road distance from Salta, via San Antonio de los Cobres, Cauchari and Tolar Grande, is approximately 400 km. Figure 6-4 shows the access route on the Argentine side of the border, along Provincial Route (RP) N°27, turning off National Route (RN) N° 51 at Cauchari.

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) N°52 to Cauchari and onwards to Tolar Grande and the Project site (Figure 6-4). Alternatively, Route 23 continues from San Pedro de Atacama and the Argentine border can be crossed at Paso de Sico before continuing on to Cauchari, Tolar Grande and the Project site (Figure 6-4). 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 (Figure 6-4).

6.6.1 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 N° 193 and 8, then to Pergamino and Melincué (RN N° 93), onwards to Chabas and Rosario (RN N° 33), and then to Rafaela, Santiago del Estero and eventually to Salta (RN N° 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.

Figure 6-4 Road access to the Project site



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For overweight loads (i.e. 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 N° 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 N° 34 and 9), and from there south to San Antonio de los Cobres (Figure 6-4).

Beyond San Antonio de los Cobres, at Cauchari, the Project access road branches off onto RP N° 27, continues to Tolar Grande and the Project site, and then onwards to Socompa on the Chilean border.

6.6.2 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 6-4). 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 6-4). 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 6-4 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 6-4). The advantage of this particular route on the Chilean side is a lower altitude terrain.

In regards to customs clearance for road transport entering/leaving Argentina, this can be done at Paso de Sico. This enables direct logistical access to/from site via Baquedano, Peine and Socaire in Chile (Figure 6-4). For the route approaching the Project site through the border post at Paso Socompa, there is currently only a police control point and no customs clearance facility.

6.7 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 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. 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 6-5 shows the railway access route on the Argentine side of the border, between Salta and Taca Taca, whilst Figure 6-6 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 tonne/axle.

6.7.1 Rail access through Argentina

On the Argentine side of the border, the railway is operated by state-owned *Belgrano Cargas y Logística 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 engines and rolling stock are located at Güemes, east of Salta.

Figure 6-5 Rail access to the Project site, through Argentina

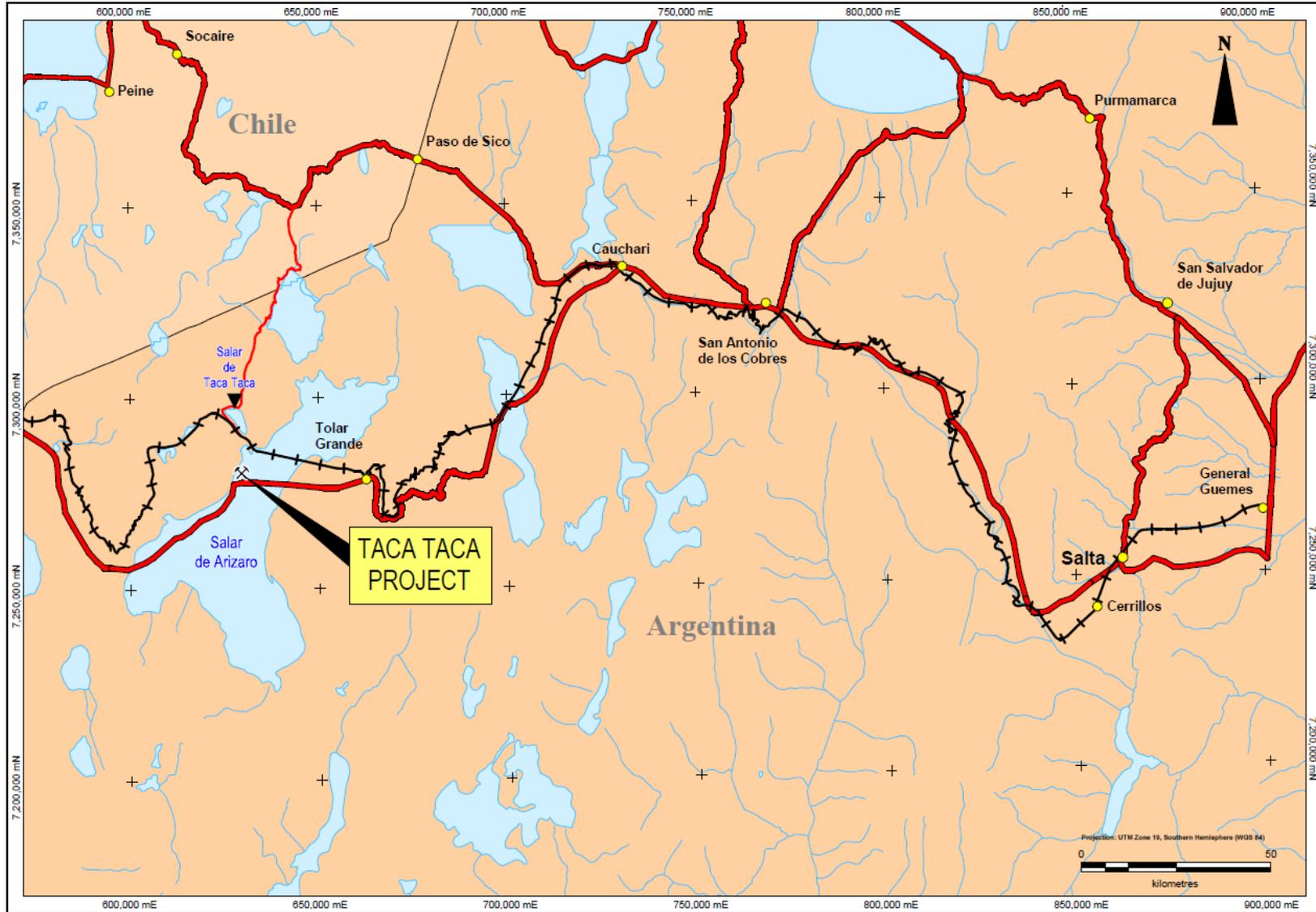
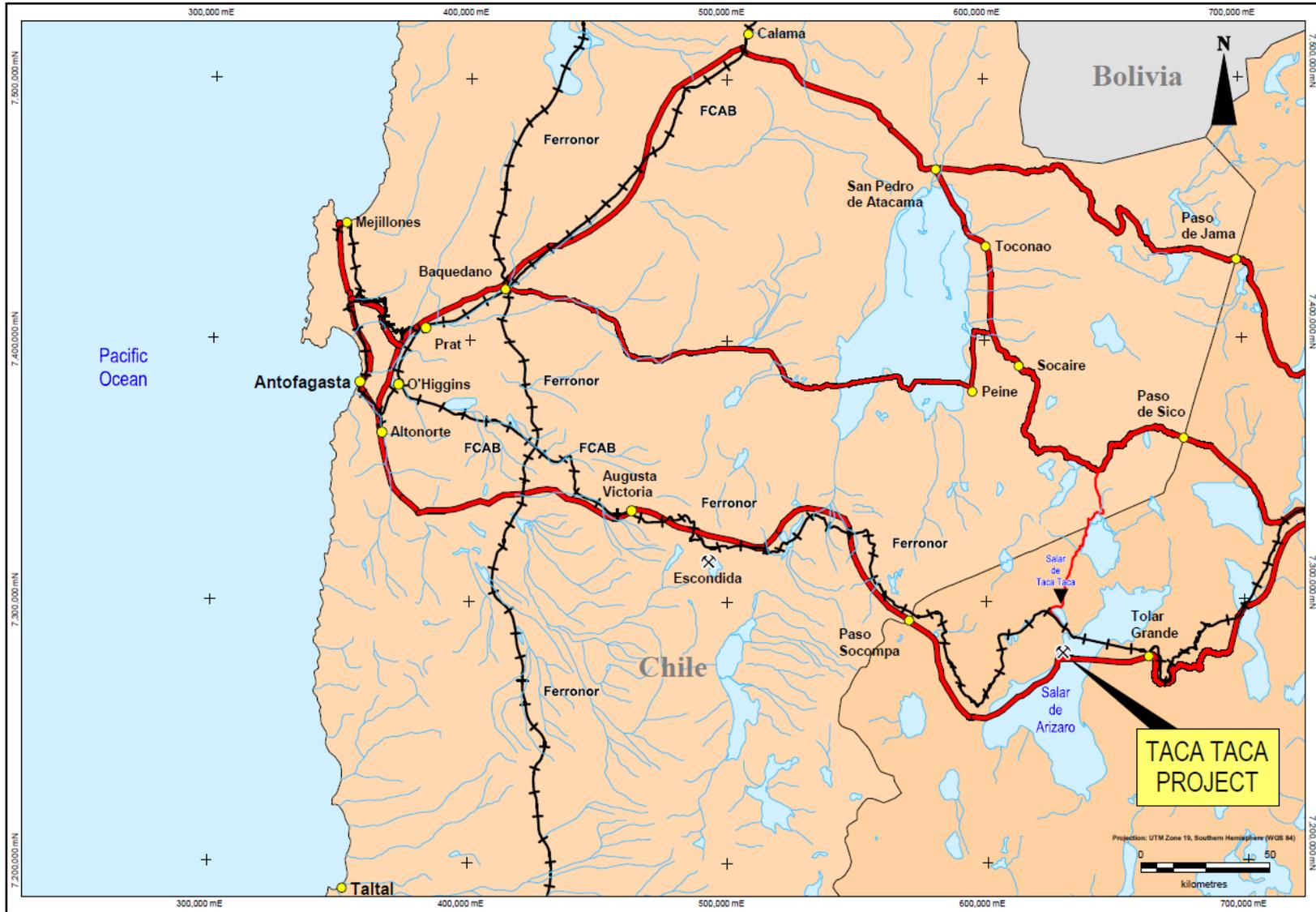


Figure 6-6 Rail access to the Project site, through Chile



6.7.2 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* (Feronor 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 of these companies have a track access agreement, each allowing the other to use its track, for payment of a fee.

6.8 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. The Company has been granted an easement and has received approval 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. This airstrip would enable small propeller powered planes (including medical evacuation planes) to provide services to the Project.

6.9 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 (Chile) and Salta (Argentina); the village has a population of around 150 people. With approximately 535,000 people, 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

6.10 Availability of power, water, personnel and areas for Project infrastructure

6.10.1 Power supply

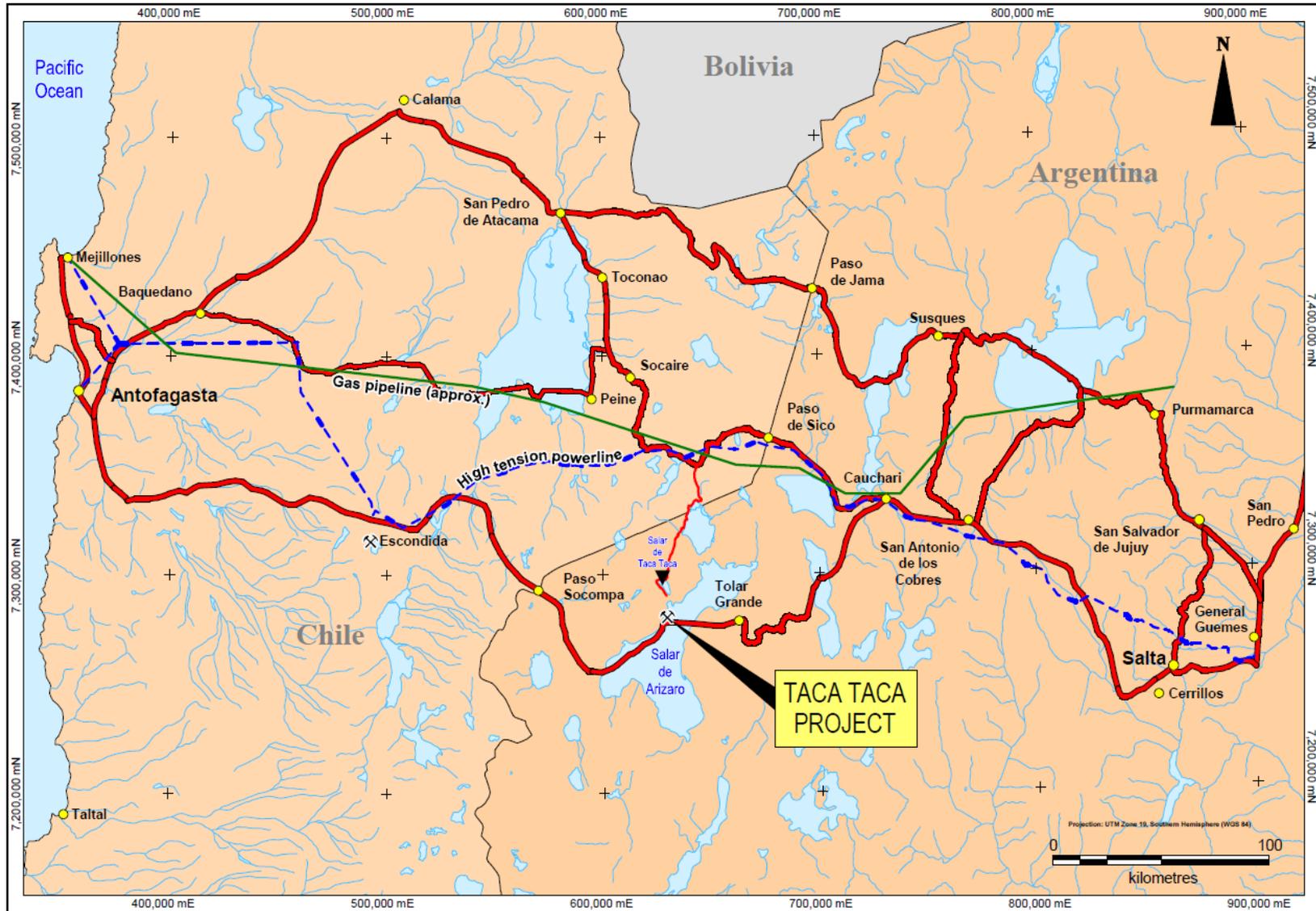
The nearby town to the Project site, Tolar Grande, generates its own power from diesel fuel. The nearest power transmission line to the Project site is to the north in the vicinity of Olacapato, near Cauchari (Figure 6-7). This is a 345 kV line from the Güemes generating station in Salta Province, extending to Los Andes in Chile. The line is privately owned and operated by *Termoandes SA* (*Termoandes*). Consultants to the Company have undertaken technical studies indicating that a straight forward interconnector to this existing power line is feasible, without compromising the existing transmission.

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 co-ordinating power generation by a number of separate entities, including *Termoandes*, and for regulating the supply and wholesale market for electric power.

CAMMESA also administers a renewable energies programme referred to as RenovAr, which includes the provision of financial stimulus packages and the granting of taxation benefits. Arising from this programme, a 300 MW photovoltaic solar power generation plant has been constructed in the Project region at Cauchari and is anticipated to begin commercial operations in 2020.

The Atacama Gas Pipeline (AGP) extending between Salta Province and Mejillones in Chile passes approximately 210 km to the north of the Project site (Figure 6-7). This pipeline has a capacity of 8.5 Mm³/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 Mm³/day.

Figure 6-7 Power line and gas pipeline routes

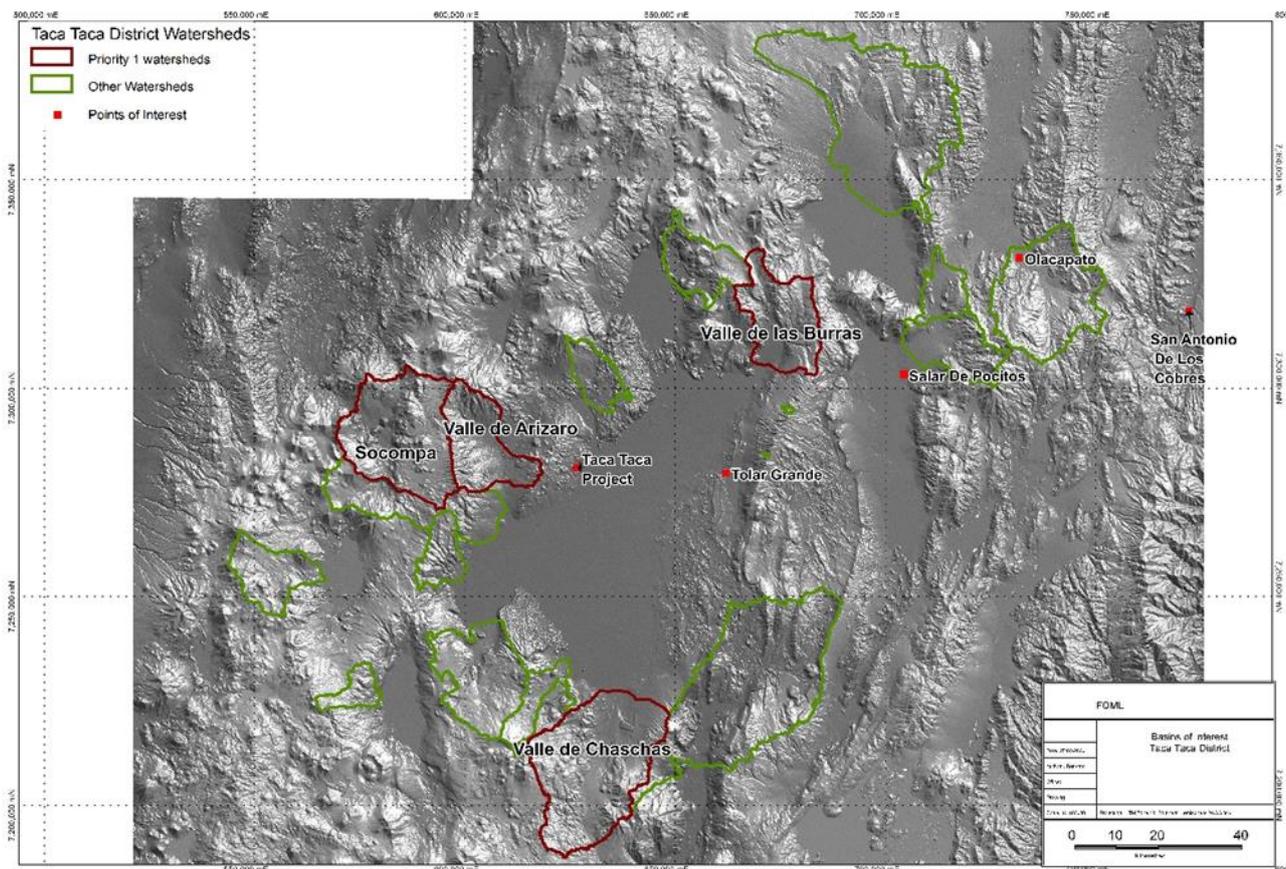


6.10.2 Water supply

Error! Not a valid bookmark self-reference. shows the major catchment (watershed) areas within the Siete Curvas basin, four of which (Valle de Arizaro, Valle de las Burras, Valle de Chaschas and Socompa) have been selected for fresh water supply to the Project. The supply of water to the Project will be from borefields, yielding freshwater from these sedimentary basins, in addition to high salinity brine water from the adjacent Salar de Arizaro.

There are additional fresh water basins further afield than the four mentioned above. Investigation drilling and modelling will continue as the Project engineering phase proceeds, in an effort to confirm the sustainability of the various supply sources

Figure 6-8 Regional catchment areas near to the Project



6.10.3 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, there is little direct work experience with metallic mines, although a new project at nearby Lindero is currently under construction. 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

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there is considerable experience in gold and silver mining in the country, particularly in Catamarca and San Juan Provinces. The Company may also recruit experienced personnel from lithium brine projects in the country. Neighbouring provinces including Jujuy and Catamarca, and further south the San Juan Province, may also have suitably experienced labour pools.

1.1 Communications

There is limited communications infrastructure in the area due to the remote location. Mobile telephone coverage is currently limited to 2G, with limited internet availability.

6.11 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 properties.

A small, north western segment of the open pit crosses onto the Francisco-2 joint venture concession. A small area in the centre of the TSF is covered by concessions held by third parties for lithium exploration. The affected area is considered small enough for non-protracted agreements with those parties.

ITEM 7 HISTORY

7.1 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 1 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.

7.2 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.

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Table 7-1 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 porphyry deposit	-	Mapping	Discovery of porphyry copper mineralisation
1975	Falconbridge	-	Cu porphyry 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

7.3 Previous Mineral Resource estimate

Table 7-2 lists the Taca Taca Mineral Resource estimate that was included in the PEA report (Ausenco, May 2013). The copper equivalent cut-off grade was calculated based on a copper price of \$2.00/lb, a gold price of \$800/oz and a molybdenum price of \$12.00/lb. The inventory was constrained by a notional pit design and is summarised in terms of supergene (secondary sulphide) and hypogene (primary sulphide) resources.

Table 7-2 Mineral Resource statement for Taca Taca, as at 30th October 2012

Classification	Tonnes (Mt)	Cu (%)	Cu _{eq} (%)	Au (g/t)	Mo (%)
Total Measured					
Supergene	-	-	-	-	-
Primary Sulphide	-	-	-	-	-
Subtotal	-	-	-	-	-
Total Indicated					
Supergene	701.0	0.60	0.70	0.08	0.009
Primary Sulphide	1,453.0	0.37	0.50	0.08	0.015
Subtotal	2,165.0	0.44	0.57	0.08	0.013
Total Meas. plus Ind.					
Supergene	701.0	0.60	0.70	0.08	0.009
Primary Sulphide	1,463.0	0.37	0.50	0.08	0.015
Total	2,165.0	0.44	0.57	0.08	0.013
Total Inferred	921.0	0.37	0.47	0.05	0.012

Note: The Mineral Resource Statement was reported using a 0.3% Cu_{eq} cut-off grade.

7.4 Previous Mineral Reserve estimate

At the time of Project acquisition, a formal Mineral Reserve had not been produced by Lumina for the Taca Taca Project.

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For the PEA report (Ausenco, 2013), a notional open pit mining inventory, inclusive of Inferred Mineral Resource, was determined as listed in Table 7-3. This inventory was derived from an optimal pit shell corresponding to a copper price of \$2.00/lb, a gold price of \$1,100/oz and a molybdenum price of \$12.00/lb. The associated waste was 2,606 Mt, yielding an overall strip ratio of 1.6 : 1.

Table 7-3 Mineral inventory for Taca Taca (Ausenco, 2013)

Mineral Inventory	Tonnes (Mt)	Cu grade (%)	Mo grade (%)	Au grade (g/t)	Cu metal (kt)	Mo metal (kt)	Au metal (koz)
Indicated	1,545.0	0.46	0.013	0.09	7,107	201	4,471
Inferred	106.0	0.43	0.005	0.09	456	5	307
Indicated & Inferred	1,651.0	0.46	0.012	0.09	7,563	206	4,777

7.5 Production from the property

To date there has been no production from the property.

ITEM 8 GEOLOGICAL SETTING AND MINERALISATION

8.1 Regional geology

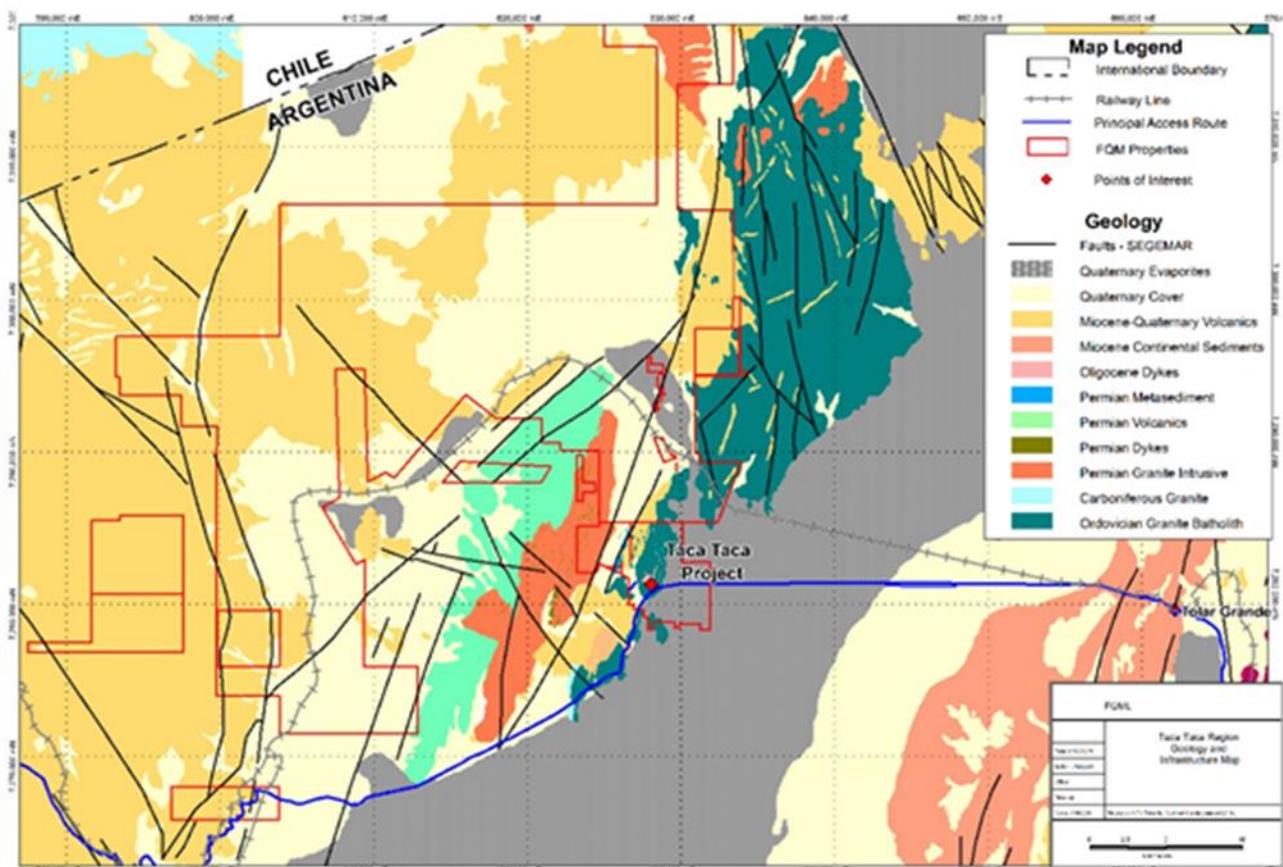
The Taca Taca deposit is located within the Puna region of Argentina. The Puna region has undergone multiple continental collision and extension events during its geological history. Today, the region comprises a back-arc basin bound by high angle reverse faults, formed during the uplift of the present-day Andean mountain chain.

The geology of the area can be summarised as a series of granitic-composition batholiths and dykes intruding the crystalline and metasedimentary basement of the Puna, associated with coeval volcanics (Figure 7-1). These were overlain by back-arc basin sediments and volcanics of Miocene to present day age, related to the uplift and erosion of the present-day Andean mountains.

The porphyry mineralisation is hosted in the southern half of a long (>50 km) Ordovician aged batholith. This batholith forms part of the north-west trending intrusive and volcanic arc that stretches over 700 km through northwest Argentina. Later Permian intrusives, volcanics, and sediments are related to the continental magmatism and back-arc basin formation during a period of passive-margin tectonics.

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

Figure 8-1 Regional geology



8.2 Project geology

The following summarises the geological history of the Taca Taca deposit area (Figure 7-2):

- During the Ordovician period (~440 to 463 Ma) the Taca Taca batholith, a large body of igneous granite rock with a surface area of >50 km², intruded into the surrounding basement of the Puna region.
- Aplite dykes and minor dolerite dykes intruded into the coarse-grained granite during the late stages of batholith emplacement.
- During the Permian, another granitic batholith (263 Ma) intruded along the western margin of the Taca Taca batholith, alongside coeval volcanic rocks from approximately 257 to 272 Ma. Permian aged, steeply dipping, dykes later intruded into the volcanic and intrusive packages.
- During the late Permian period (268 Ma and younger) a mixed sedimentary package was deposited. These comprise shale, sandstone, and a basal conglomerate, suggesting a small structurally controlled basin against the western side of the batholith.
- During the Oligocene epoch (29.3 Ma), steeply dipping rhyodacitic dyke intrusions were responsible for introducing the porphyry copper mineralisation and alteration at the Taca Taca deposit. Mineralisation is thought to have been introduced in three pulses associated with dykes of distinct textural variation.
- Regional evidence suggests that these intrusive rocks were then uplifted during the Oligocene and Miocene epochs, as part of the creation of the modern Andes, to form the Sierra de Taca Taca mountain range.
- During the Miocene-Pliocene-Pleistocene-Holocene epochs, 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 Taca Taca deposit area. The region is still seismically and volcanically active, with basaltic plugs and flows of <1 Ma age.
- Quaternary to present day deposition of salts and sands in intermontane basins form the evaporitic salars of the region.

8.2.1 Rock types

Mineralisation detailed in this Technical Report is associated with the Taca Taca Bajo deposit and is hosted within the Ordovician granite batholith and the co-magmatic aplite and dolerite intrusives. A smaller, less explored deposit, known as Taca Taca Alto, occurs 4 km to the west and is not within the Company's concession holdings.

Ordovician Taca Taca batholith

The batholith of granite to granodiorite composition outcrops on the northwest margin of the Salar de Arizaro and forms a prominent range over 50 km in length.

It is a medium to coarse grained, equigranular to moderately porphyritic rock, with phenocrystic plagioclase, quartz, k-feldspar, biotite and amphibole (Figure 8-3). The batholith is cut by several co-magmatic aplite sills/dykes (Figure 8-4) and less common, steeply dipping, dolerite dykes (Figure 8-5).

The batholith's western margin is in contact with a northeast trending Permian aged granite body. The granite is partially obscured by thin (<50 m) recent lava flows adjacent to the Aracar Volcano.

Figure 8-2 Taca Taca Bajo area geology

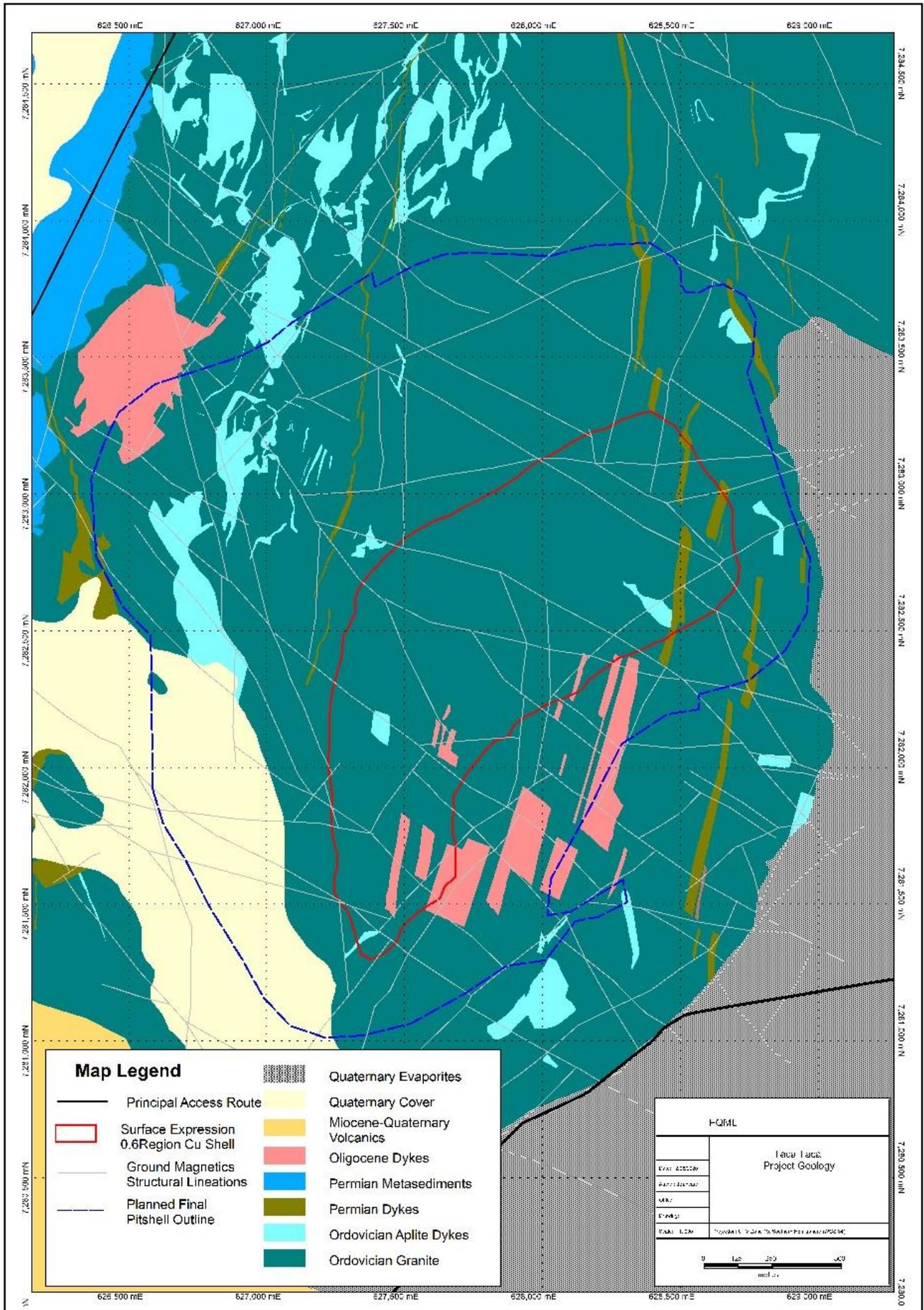


Figure 8-3 Drill core example of Ordovician granite host rock

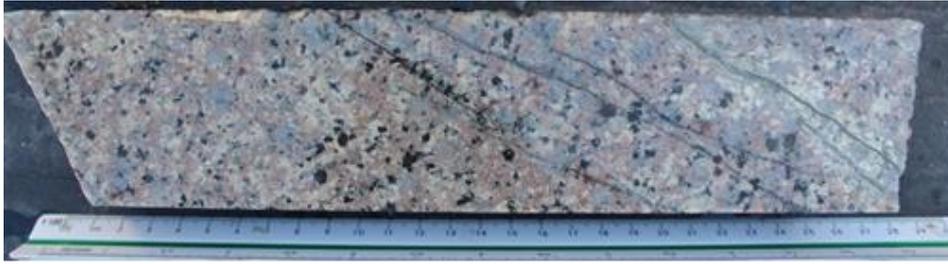


Figure 8-4 Drill core example of aplite sill



Figure 8-5 Drill core example of dolerite dyke



Permian Granite

A medium to coarse grained, pink granite crops out to the west of the Ordovician granite. It measures over 20 km in strike length and up to 5 km in width. This granite forms the Taca Taca Massif, which rises to heights of 4,300 m 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

The volcanic rocks in the Taca Taca area are a dacite to rhyodacite dominated suite (Figure 8-6) coeval to the Permian intrusive granite. The package contains volcanoclastics, lava, crystal tuff, and ignimbrite subaerial components, with lesser dykes, domes, and other intrusive subvolcanic material.

Permian metasedimentary rocks

Metasedimentary rocks outcrop to the west of the deposit. The sequence is dominated by dark purple shales and siltstones (Figure 8-7), transitioning to a volcanic breccia at the base. The basal volcanic breccia has a transitional conformable contact with an underlying dacite crystal tuff that yielded a U-Pb zircon age of 268 Ma.

Figure 8-6 Taca Taca outcrop example of Permian dacite lava



Figure 8-7 Bedded Permian siltstone and shale in a trench 1 km west of Taca Taca



Permian dykes

Permian age dykes occur across the Project area within the Permian volcanic package and have varied associations. The dykes around the Taca Taca deposit are of rhyolite composition, are typically several metres wide, and north-south trending (Figure 8-8).

Oligocene dykes

North-northeast striking, steeply dipping dacite-rhyodacite dykes (29.3 Ma) occur locally at the Taca Taca Bajo 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, based on drilling and mapping.

Figure 8-8 Drill core example of Permian aged rhyolite dyke



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 recognized (Figure 8-9):

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, large areas are covered by recent andesite to basalt lava flows, volcanoclastics, and pyroclastic deposits from the Aracar and Arizaro volcanoes. These flows cover the Ordovician and Permian granite, as well as Permian volcanics.

Quaternary evaporites

Salar de Arizaro, a salt lake spanning 1,600 km², lies in a structurally-controlled closed basin immediately east and southeast of the Project and partially covers the Taca Taca host granite. Proximal to the deposit, the upper level (10 m) of the salar is predominantly halite with interspersed sands, capped by a 2 m to 3 m thick surface salt crust.

8.2.2 Alteration

Alteration at Taca Taca Bajo is broadly typical of an Andean porphyry copper-gold-molybdenum system. Large zones of hydrothermally altered rocks grade from a central potassic core to peripheral phyllic and argillic zones, although there is a limited propylitic alteration zone compared to similar size porphyry deposits.

At the deposit scale, a pervasive phyllic (quartz-sericite-phengite-pyrite) alteration often overprints the original alteration assemblages. With the benefit of geochemical modelling of multi-element assay data (Figure 8-10), a remnant potassic (biotite-K feldspar) alteration is observed at the centre of the system and coincident with a series of rhyodacite porphyry dykes. Laterally, around the edges of the deposit, the phyllic altered zone grades into propylitic (chlorite ± epidote) alteration. Supergene argillic (kaolinite-alunite-chalcedony-chalcocite) alteration impacts the upper parts and to depth along structures.

Figure 8-9 Drill core examples of at least three rhyodacite porphyry phases associated with mineralising events (shown from earliest to latest)

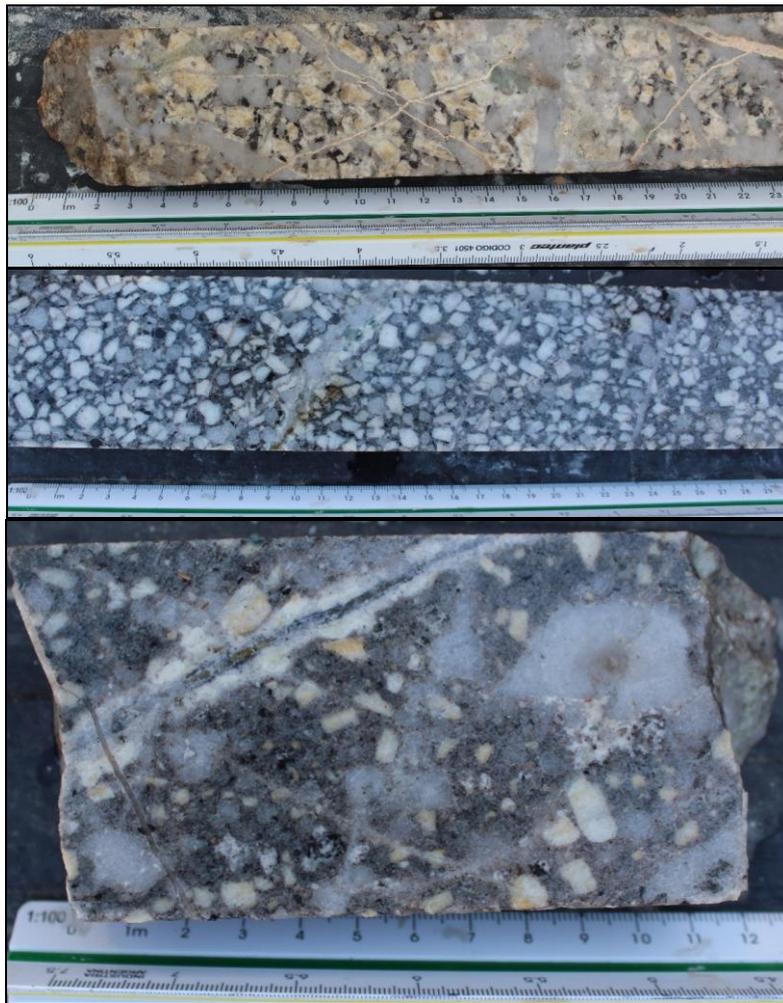
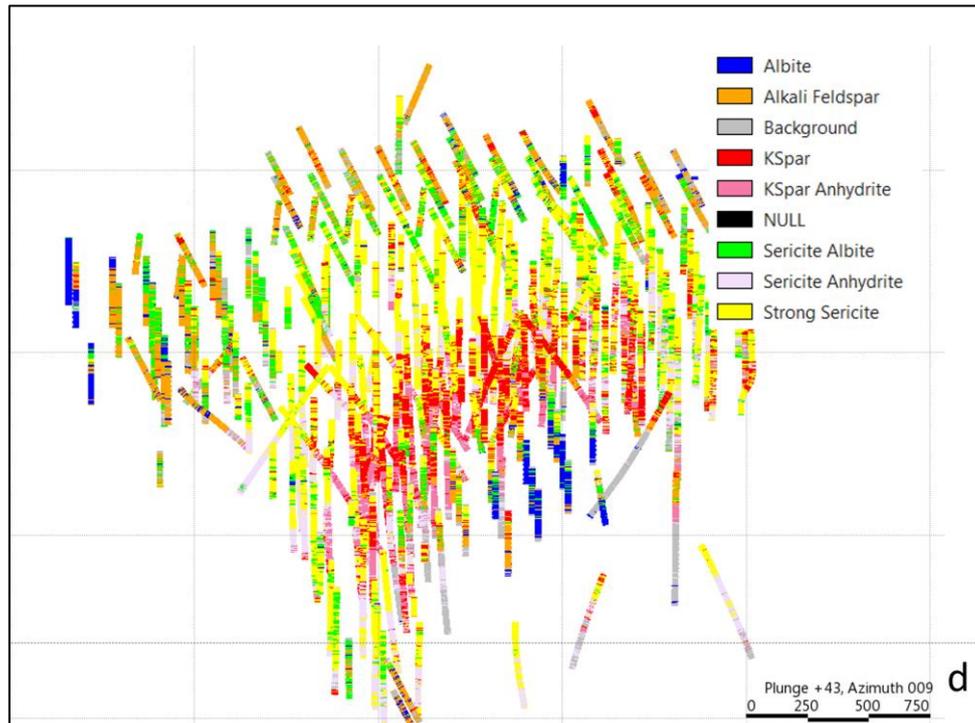


Figure 8-10 Alteration zonation interpreted from multi-element assay data



Potassic alteration

The innermost potassic altered core is characterised by the abundant coarse secondary biotite replacement of mafic minerals and by rare potassium-feldspar (K-feldspar) in vein selvages. This zone is largely overprinted by phyllic alteration but is identified by a distinct geochemical signature (Figure 8-11) and visible as remnant rafts.

Potassic alteration is associated with a weakly mineralised pervasive ‘A-type’ quartz vein stockwork. ‘B-type’ quartz-molybdenite veinlets are common around the outer edges of this quartz-rich core as phyllic alteration overlaps and becomes dominant (Figure 8-11).

Figure 8-11 Drill core example of potassic altered granite showing quartz-molybdenite vein



Phyllic alteration

Phyllic alteration is the most widely distributed and pervasive alteration in the deposit and is associated with the bulk of the mineralisation. Alteration occurred in two stages:

1. Early pale green phengite typically found in sericite-andalusite±anhydrite selvages to quartz-copper sulphide ‘D-type’ veinlets. It is associated with an intermediate sulphidation mineral assemblage,

where chalcopyrite and bornite are significantly more abundant than pyrite. Higher copper grades and above-average gold grades are related to this alteration phase.

2. A late phase characterised by more pervasive white sericite and quartz overprints the potassic, earlier phyllic, and propylitic alteration zones (Figure 8-12). A change in sulphidation state of the mineralising fluid from intermediate to high sulphidation resulted in pyrite becoming more abundant as disseminations and veinlets. Pyrite-bornite and pyrite-chalcocite-covellite sulphide assemblages are observed.

Though there are two distinct phases, phengite and white sericite are found broadly intermixed within the deposit. Pyrite is found throughout the mineralised zone but shows a broad zonation outward in intensity.

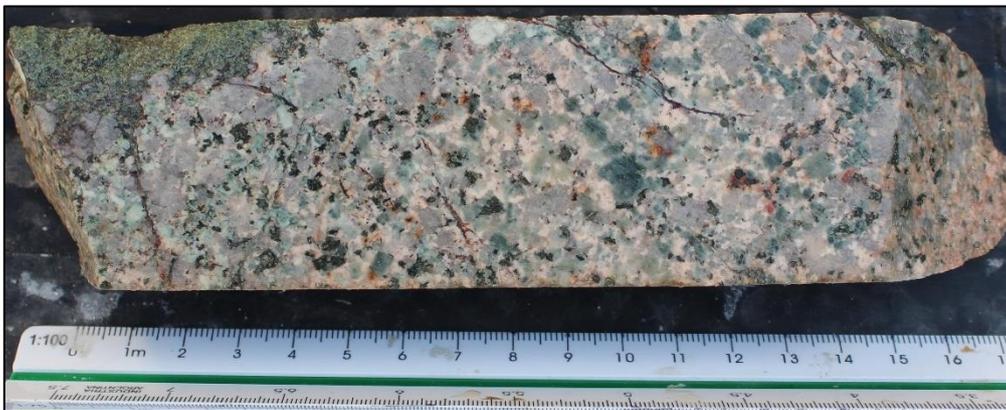
Figure 8-12 Drill core example of pervasive white sericite alteration of granite



Propylitic alteration

The propylitic altered zone (Figure 8-13) is observed at the periphery of the deposit and is largely overprinted by the late phase of phyllic alteration. It is characterised by illite-chlorite mineral assemblages with minor epidote. This zone is also associated with the strong pyritic halo rimming the outer edges of the deposit.

Figure 8-13 Drill core example of propylitic alteration of granite

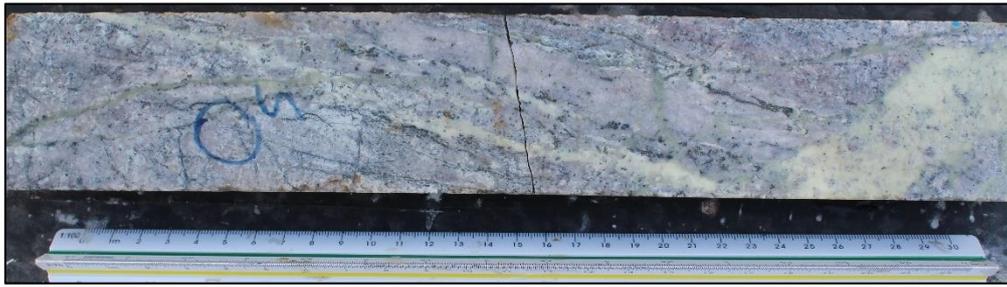


Supergene argillic alteration

A well-developed 150 m to 300 m thick leach cap overlies the mineralised zone. It is characterised by abundant secondary kaolinite and hematite-jarosite fractures that replaced sulphide veins. Rare lenses of copper oxides exist and a perched horizon of secondary sulphide mineralisation is present in the eastern side of the deposit.

Copper from the weathered leached cap was remobilised and-deposited directly underneath, and to depth along structures and within the host rock, as secondary sulphides in zones of supergene enrichment. Steeply dipping structures allowed for localised supergene alteration to depths exceeding 1 km below surface. Secondary kaolinite, chalcedony, alunite, and chalcocite veins are associated with these structures (Figure 8-14).

Figure 8-14 Drill core example of alunite veining with chalcocite related to supergene alteration



Alteration and metallurgy

Pyrite occurs throughout the deposit although the hypogene mineralogy is zoned, grading outwards from the centre to increased pyrite and decreased chalcocopyrite-bornite (Figure 8-15). A strong pyritic halo exists on the outer rim (up to 10% sulphur). In the supergene zone, chalcocite has precipitated as overgrowths on pre-existing sulphides (Figure 8-16). Relative pyrite content within the plant feed will need to be monitored to minimise the impact on recovery.

Figure 8-15 Sulphide sulphur to copper ratio – approximation of pyrite zonation

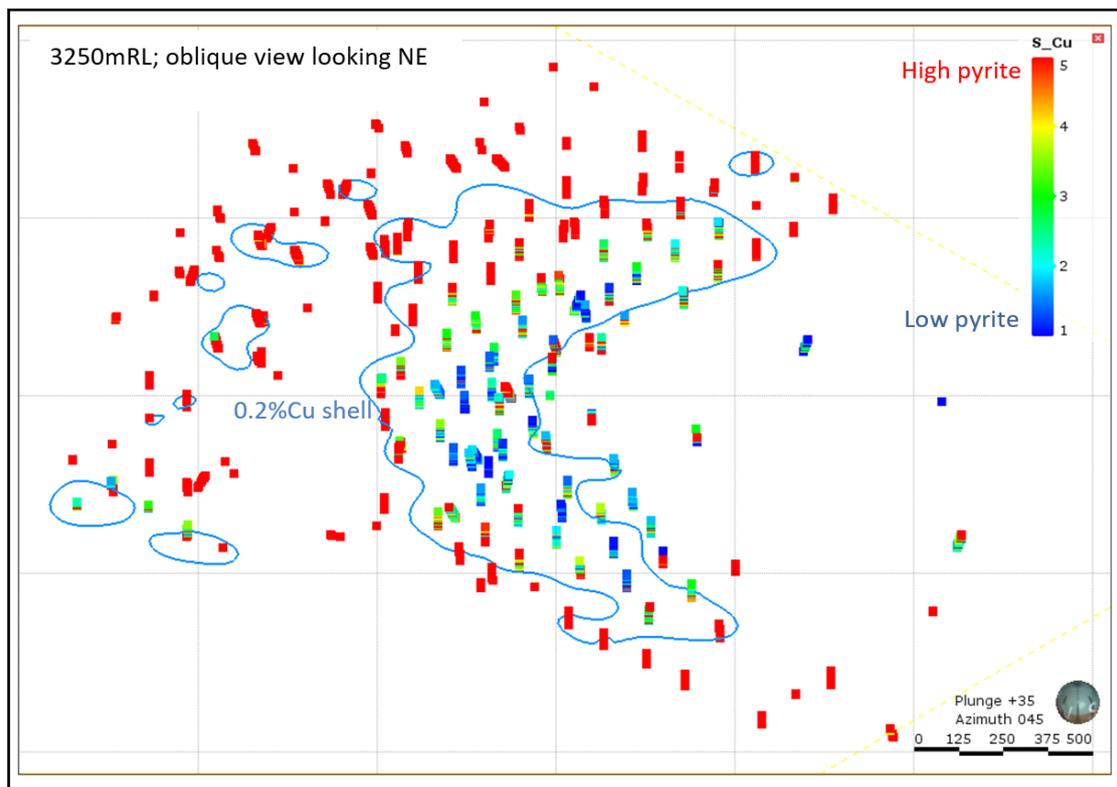
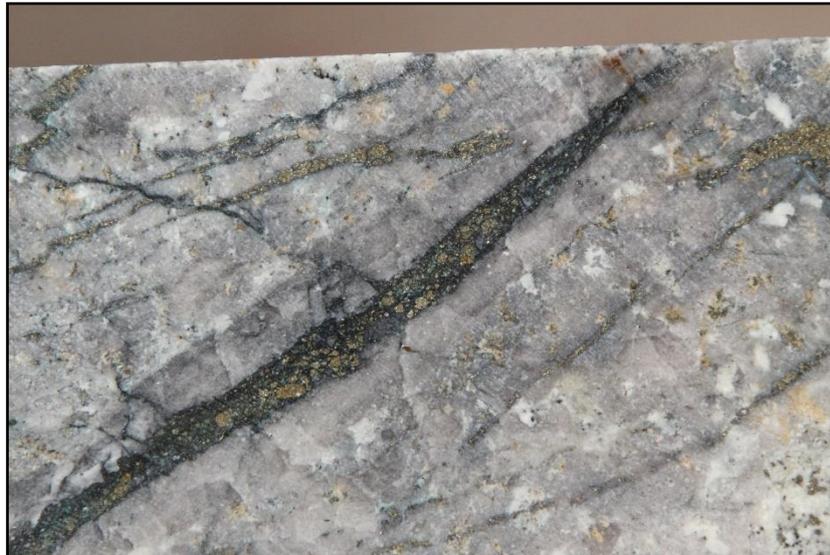
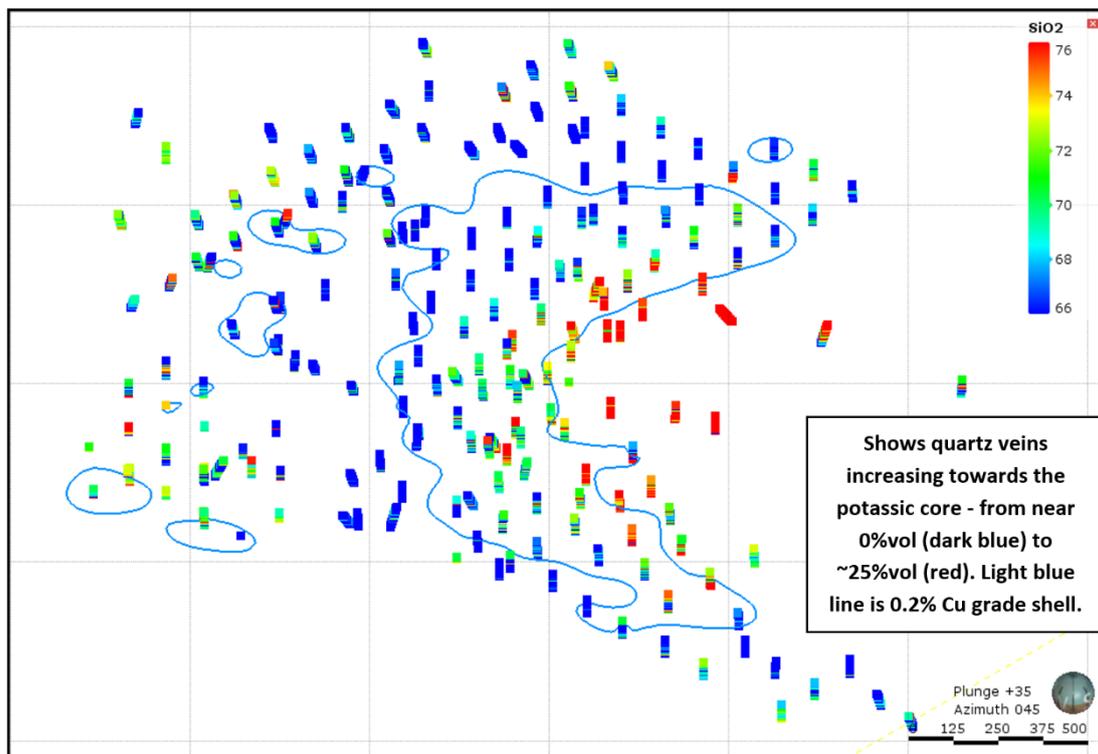


Figure 8-16 Chalcocite rims on pyrite



Along the east to south-east margins of the proposed pit, some feed to the plant is likely to contain elevated quartz veining. Without SiO₂ assay data, quartz vein intensity can only be approximated into broad relative zones (Figure 8-17). When feeding material to the processing plant from the potassic altered area, quartz vein intensity will need to be factored into grinding and power consumption rates.

Figure 8-17 Estimated SiO₂ values at 3,250 mRL



Phyllic alteration within the mineralisation contains intermixed white sericite (muscovite) and green sericite (phengite). White sericite appears pervasive and is associated with pyrite being more abundant than copper sulphides. Conversely, green sericite, typically occurring in selvages to quartz-copper sulphide veins, is associated with copper sulphides being more abundant than pyrite. Where phyllic alteration is less intense, secondary biotite is preserved.

It is not possible to estimate the relative abundance of white, green, or black micas using the existing data. Since metallurgical testwork shows no clear correlation between mica speciation and performance, there is

likely negligible risk associated. As mining proceeds, however, it is recommended bulk mineralogical composition and possible associations with performance be continually assessed with the view to optimising metal recovery at plant scale.

8.2.3 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 saprolitic material exists in or around the deposit. Alluvial and colluvial fans of gravel regolith are common in the deposit surrounds. Although gravel dominant, these also contain mud, silt, sand, and sometimes boulders.

Observations from drill core logging suggest that the topmost 30 m of leached saprock is often more broken, or crumbly in texture. This material is the source of locally derived small-scale alluvial and colluvial deposits.

8.2.4 Mineralisation

Most of the mineralisation is hosted by phyllic-altered Ordovician granite and associated aplite and minor dolerite dykes. Dolerite dykes have relatively higher copper grades owing to the abundance of ferrous iron from mafic minerals facilitating the precipitation of copper from hydrothermal fluids. Mineralisation can be divided into the upper leached zone and the underlying supergene and hypogene mixed mineralisation.

Leached horizon

A leached zone (also referred to as the “leach cap” or “leached cap”) ranging from 150 m to 300 m thick (Figure 8-18) is almost completely depleted of copper mineralisation and is dominated by limonite assemblages consisting of hematite, jarosite, and goethite (Figure 8-19).

Figure 8-18 Plan showing thickness contours of the leached cap

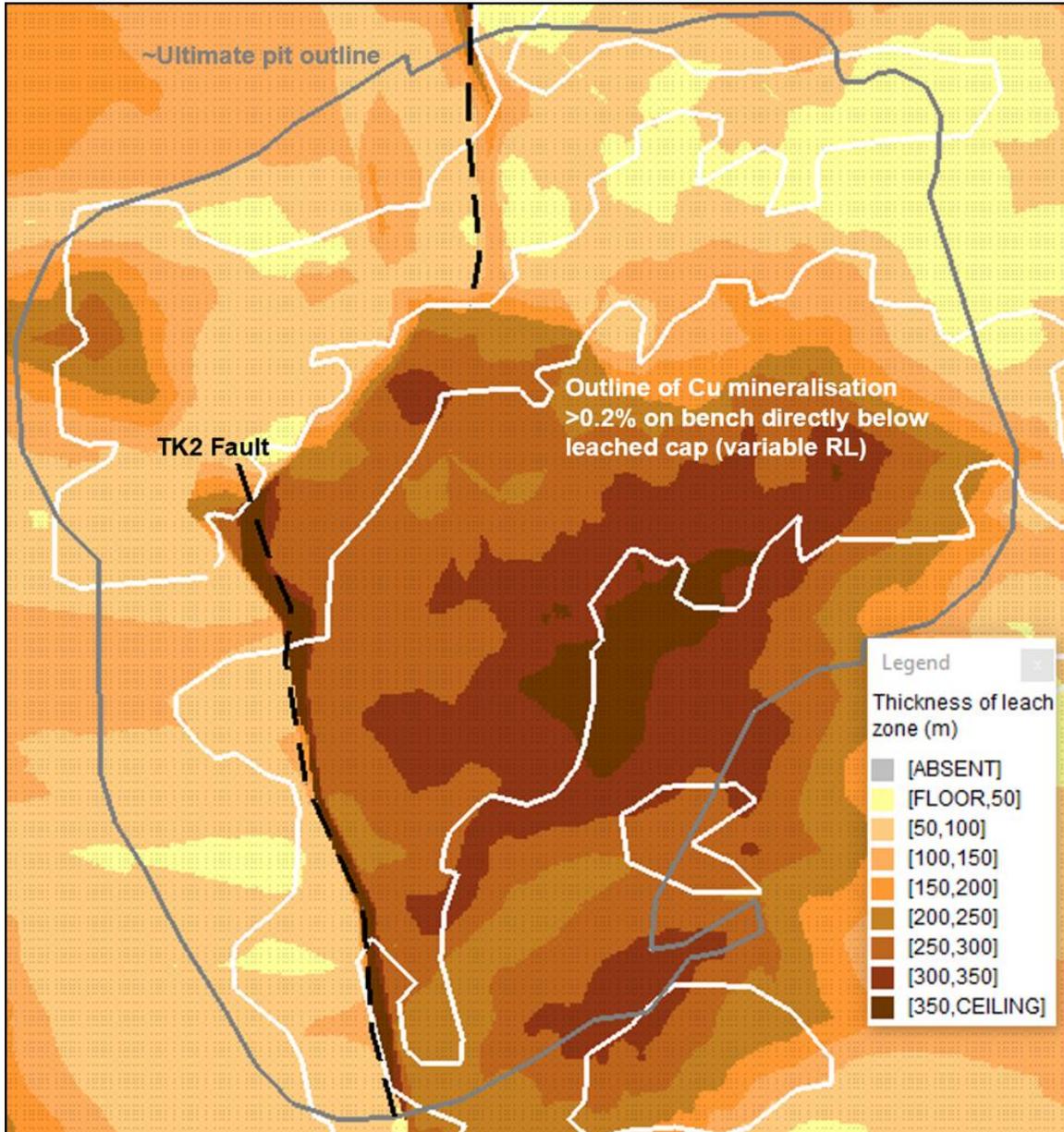


Figure 8-19 Drill core example of granite from the leached cap showing abundant iron oxides



Refractory copper and remnant zones of copper oxide mineralisation are limited to sporadic metre-scale sub-horizontal lenses. A decimeter scale perched horizon of copper mineralisation is in the thicker part of the leached zone to the east and contains predominant chalcocite with minor copper oxides. Supergene gold mineralisation is also enriched near the surface above the thickest portions of the leached cap.

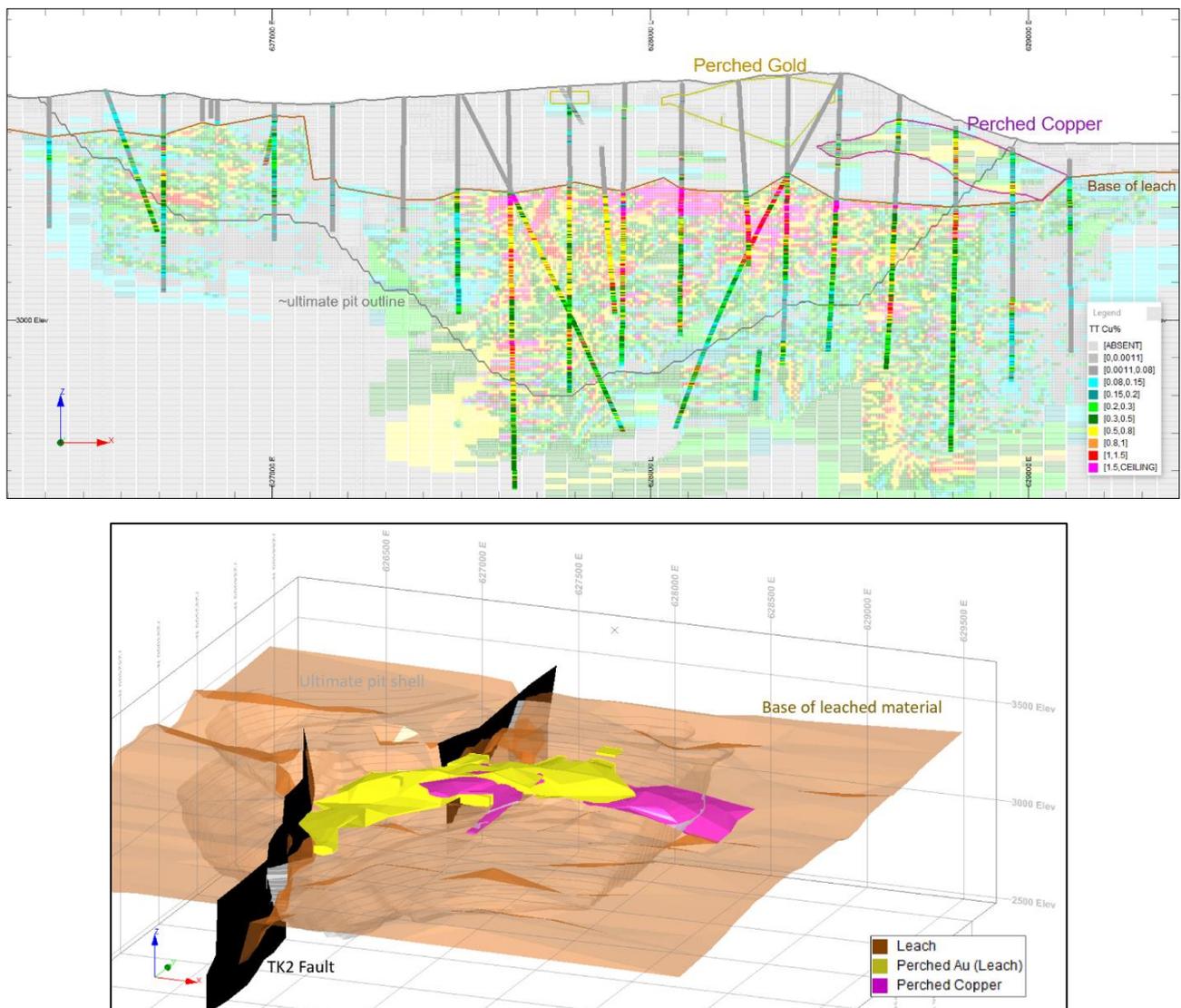
A perched horizon of copper mineralisation is located in the thicker part of the leached zone. This mineralisation is characterised by dominant supergene chalcocite with minor copper oxides and primary

sulphides. Since this zone will be mined during initial stripping to reach the main body of mineralisation, and subject to further evaluation during the engineering phase, this material could provide an initial source of plant feed.

Supergene gold mineralisation is enriched near the surface above the thickest portions of the leached zone. 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 gold-bearing material for later treatment.

The location of the perched copper and perched gold zones relative to each other is presented in Figure 8-20.

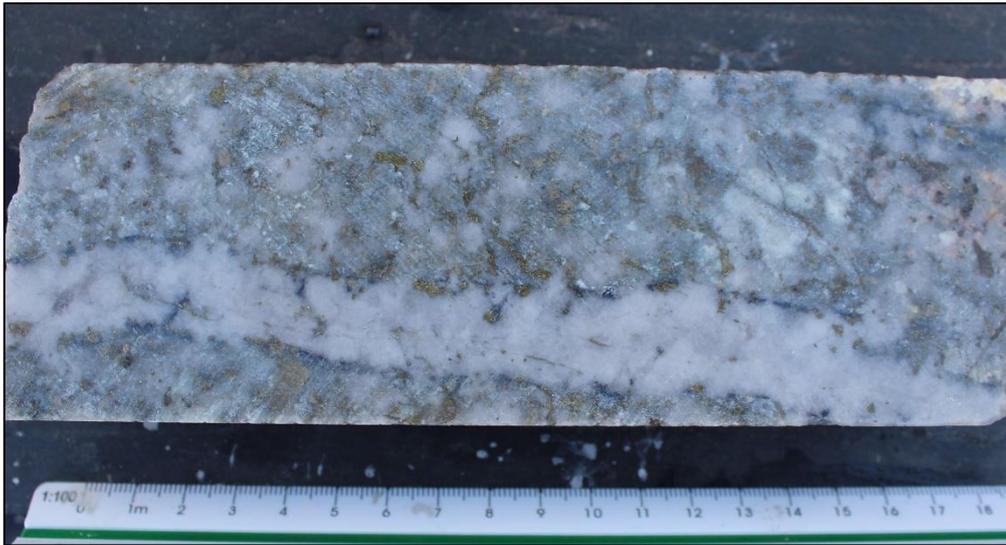
Figure 8-20 Top: Section 7,282,775 mN showing the perched copper and gold horizons within the leached zone (looking north). Bottom: perched copper and gold horizons represented by 3D wireframe volumes.



Supergene-hypogene mineralisation

Hypogene copper mineralisation is dominated by chalcopyrite (cpy) with lesser bornite (bn), chalcocite, covellite, and digenite. Copper sulphides mostly occur as disseminations in sericitic vein selvages, in microfractures, and intergrown with quartz veins (Figure 8-21).

Figure 8-21 Drill core example of sericite altered granite with hypogene mineralisation (chalcopyrite, pyrite and quartz-molybdenite vein)



In the potassic altered zone, minor chalcopyrite and trace bornite is associated with secondary biotite. Most copper mineralisation sits within the phyllic-altered zone and is related to two phyllic alteration stages. Early green sericite alteration is associated with the chalcopyrite-bornite sulphide assemblage, higher copper grades, and above-average gold grades. Late white sericite and quartz alteration is associated with pyrite-bornite and pyrite-chalcocite-covellite sulphide assemblages. Item 7.2.2 provides more detail on phyllic alteration phases and associations.

Molybdenite typically occurs as disseminations and in quartz vein stockworks. Molybdenite-rich quartz veins are more common in the potassium-feldspar altered granites and the aplite dykes.

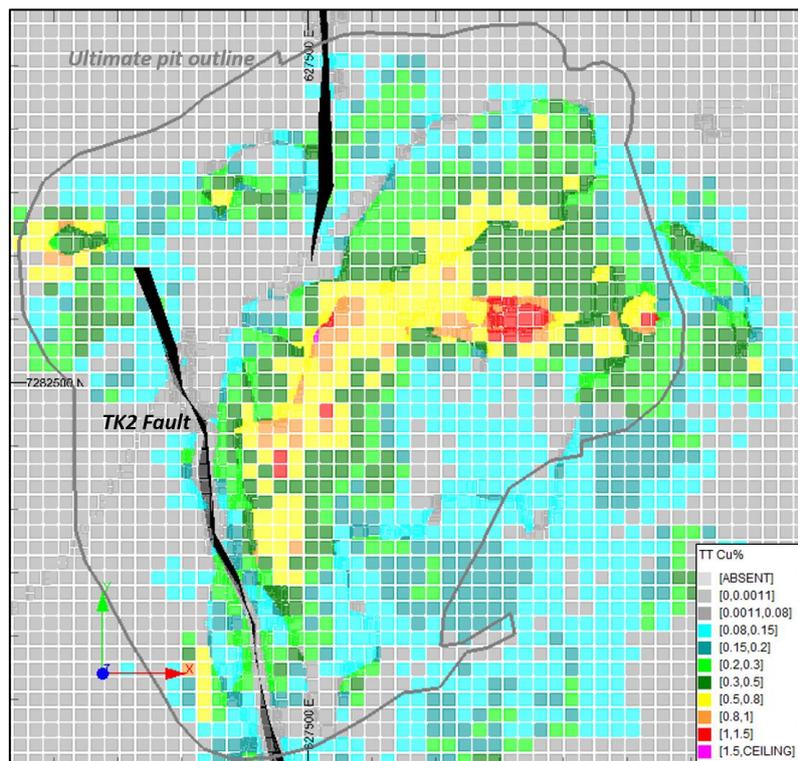
Fine-grained black chalcocite and lesser covellite are the main secondary copper sulphides associated with supergene alteration of the mineralisation (Figure 8-22). A discontinuous horizontal zone of supergene mineralisation occurs directly beneath the leached cap and contains an elevated copper tenor relative to the hypogene zone. This enrichment blanket is best developed in the north-eastern part of the deposit and proximal the TK2 Fault. In addition, supergene alteration associated with steeply dipping structures facilitates alteration and enrichment to depth. Pockets of supergene mineralisation internal to the hypogene zone are common. Though copper mineralisation is observed as zones where hypogene or supergene mineralisation is most dominant, the overall mineralogy of the deposit is mixed with a variable chalcopyrite to chalcocite ratio.

The mineralisation has an overall arcuate shape, reflecting the morphology of the igneous intrusion and the alteration effects of the mineralising fluids (Figure 8-23).

Figure 8-22 Chalcocite in supergene altered granite, with minor hypogene mineralisation



Figure 8-23 Plan view slice of the parent model at 3,250mRL, showing arcuate shape of copper mineralisation



Mineralisation and metallurgy

Feed to the plant, from both the perched copper horizon (within the leached zone) and the topmost portion of the supergene zone, will mix with surrounding leached material during mining. In some areas, the upper portion (0 m to 80 m) of the supergene horizon is also characterised by discrete interspersed metre-scale leached lenses. Mixing of this nature is likely to introduce atypical levels of iron oxide minerals into the earlier feeds. Although this material represents a small volume within the mine life, iron oxide contamination may need to be managed during the plant commissioning phase.

Earlier feed will also contain variable amounts of soluble copper (localised oxidation within the supergene dominant zones). Depending on the degree of oxidation and consumption of acid by gangue minerals, this material may require some mining and processing management. Further discussion on the metallurgical performance of the mixed mineralisation is provided in Item 13.

Supergene enriched pockets internal to hypogene zones are common and observed at depth. In certain areas, primary sulphide dominant material will contain lesser but variable quantities of secondary sulphides (chalcocite).

Studies undertaken to investigate the rate of oxidation of exposed coarse-crushed material suggest no significant increase in soluble copper content over a nine-month period under on-site conditions. However, drill core stored under cover at the Salta core shed does show frequent examples of oxidation with copper staining rimming chalcocite (Figure 8-24). It is suspected hotter and more humid conditions at the core shed have promoted oxidation of rock material exposed to air and moisture over extended periods (>12 months) when compared to Project site conditions. Long-term stockpiling or ponding of pit water over broken mineralised material should be avoided.

Figure 8-24 Granite with disseminated chalcocite and pyrite, showing chalcocite and oxide rims (from Salta core storage shed)



Rare trace quantities of nickel mineralisation have been observed in drill core. Nickel assay data suggests that quantities will be insignificant at a mining scale.

8.2.5 Structural geology

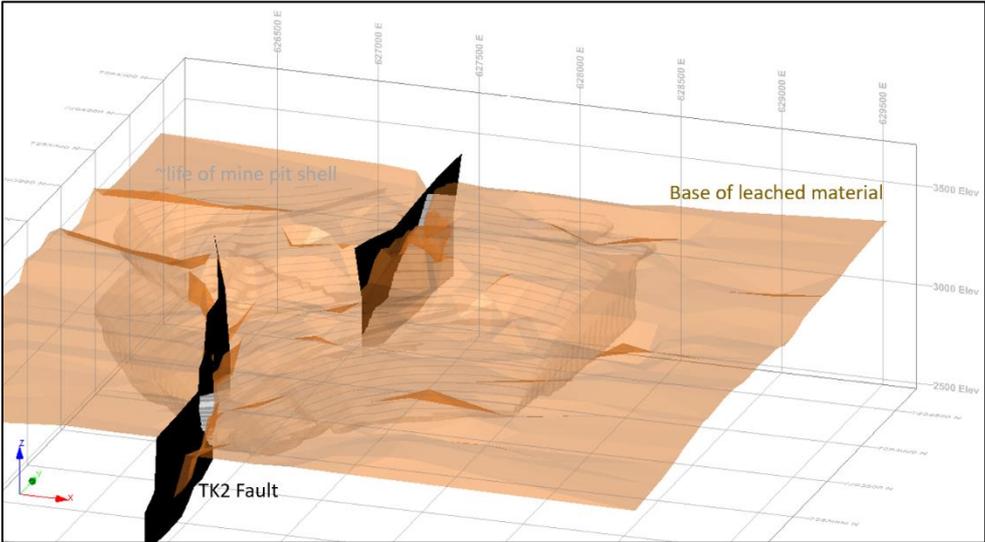
North-northeast and northwest trending, steeply dipping discrete mylonite zones are widespread within the Ordovician granite host rock. These pre-existing zones of structural weakness control the emplacement of Oligocene rhyodacitic dykes, porphyry related quartz veining, fractures, and small-scale faults.

Evidence from mapping, drill core, and geophysical surveys indicates that the deposit is structurally influenced by two main sets of steeply dipping faults. The main set trends northwest-southeast with a subordinate set oriented north-south. These structures act as conduits for supergene alteration and are commonly associated with secondary quartz, gypsum, alunite, and chalcocite.

The TK2 (West) fault

A deposit scale north-south trending fault, known as the West or TK2 fault, runs along the western edge of the design pit and dips steeply to the west. The base of the leached cap and underlying copper mineralisation is displaced across this fault, with the leached-mineralised contact significantly deeper on the eastern side (Figure 8-25). Where exposed on surface the fault comprises breccia zones up to 5 m wide. Centimetre-thick chalcocite rich veins are observed proximal to the fault.

Figure 8-25 Location of TK2 fault in relation to pit design



ITEM 9 DEPOSIT TYPE

9.1 Mineral deposit type

Taca Taca is a porphyry copper-gold-molybdenum deposit hosted by plutonic rocks of granitic composition with lesser dacite, dolerite, and rhyolite intrusions. Kilometre-scale zones of hydrothermally altered rocks grade from a central potassic core to phyllic and argillic zones. There is a limited propylitic altered zone compared to similar size porphyry deposits. Phyllic alteration is most pervasive across the deposit and is associated with the bulk of mineralisation. Late argillic supergene enrichment has improved the tenor of mineralisation.

Mineralisation can be divided into the upper leached horizon and the underlying supergene and hypogene mixed zone. Extensive low-grade mineralisation is found disseminated and in fractures, veinlets, and quartz vein stockworks. The sulphides are broadly zoned with a chalcopyrite-bornite-molybdenite core yielding to a strong pyritic halo around the outer rim. Overall, mineralisation is of mixed sulphide species atypical to this style of deposit.

Surface weathering and oxidation leached copper from oxide and hypogene copper minerals which led to the development of a 150-300m thick copper depleted leached cap. Remobilised leached copper deposited as secondary sulphide underneath the leached horizon creating zones of supergene enrichment. Fine-grained black chalcocite and lesser covellite are the main secondary copper sulphides associated. The supergene mineralisation occurs as discontinuous zones located directly beneath the leached cap. Below this, the contact between hypogene dominant and supergene dominant mineralisation is highly variable and reflects alteration to depth along structures and within host rocks. The copper tenor of supergene mineralisation is relatively higher than the hypogene zones.

9.2 Guiding principles for exploration and modelling

The extent and geometry of the granite host is well defined by drilling, mapping, aerial photography, and geophysical surveys. Drilling was initially guided by outcrop geology, followed by drilling which targeted mineralisation extents. Drillhole alignments were guided by the overall orientation of the mineralisation, to ensure a high angle of intersection to mineralised intervals.

Detailed drill core logging and analysis of multi-element assay data allowed for identification and modelling of deposit scale geological domains. These domains were based on a combination of weathering, alteration, lithological, and mineralisation characteristics.

Modelling of geology and mineralisation domains was by means of interpretation and wireframe linking of interpreted string envelopes on deposit cross sections.

Mineralisation was introduced with the intrusion of Oligocene-aged rhyodacitic porphyries. The distribution of copper sulphides is controlled by the vein stockworks related to dyke intrusions and by igneous contacts. Hydrothermal breccias, intrusive breccias, intersecting fracture sets, and veining proximal to fault zones often coincide with the highest metal concentrations. The combination of geology logging data, copper grades and multi-element geochemistry has assisted in delineating mineralisation domains.

The 150 m to 300 m thick leached cap contains narrow discontinuous lenses of copper-oxide mineralisation throughout with discrete perched chalcocite-rich and gold-rich horizons to the east. The basal limit of the leached horizon was guided by absence of copper grades together with elevated hematization and logged weathering data.

Secondary sulphides, formed by supergene enrichment, are dominant within a discontinuous blanket directly underneath the leached cap. Supergene mineralisation is also found intermixed with hypogene

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mineralisation to variable amounts throughout the deposit owing to deep-seated alteration along structures and host rocks. Generally, supergene zones have a notable increase in copper grades.

Modelling of mineralisation within geological domains was guided by an analysis of sequential copper assay data for defining dominant copper species within the different altered rock-types. The TK2 fault position was modelled using integrated outcrop mapping, geophysical survey, drill-core logging, core photographs and assay data.

Mineralisation remains open at depth and in some areas surrounding the deposit.

ITEM 10 EXPLORATION

10.1 Historical exploration

Copper mineralisation at Taca Taca was first reported by *Fabricaciones Militares* in the late 1960s. Historical exploration included multiple drill programs and geophysical surveys across the Project area. An overview of the historical drill campaigns by each company is provided in Table 7-1. The following points summarise the geophysical surveys that were completed:

- Transient electromagnetic (TEM) and Induced Polarisation (IP) surveys were completed by BHP Minerals in 1997, aiming to delineate sulphide mineralisation extents. The TEM survey was over the porphyry deposit and totalled 36.8 km (10 lines at 500 m spacing)
- Ground magnetic and gravity surveys were undertaken by CASA in 1999, targeting exotic copper on the northern edge of the Salar de Arizaro.
- A 38.5 km radiometric survey was completed across the property by Rio Tinto in 1999 to aid with shallow mineralisation targeting. This survey was overlapping and coincident with the line spacing that CASA had covered with ground magnetics.
- A Titan 24 survey (combined DCIP and magnetotelluric data) was conducted on behalf of Lumina during 2010. Results from this survey provided several targets of deeper sulphide mineralisation for Lumina's early drilling programme.

Surface outcrop mapping was active during most of the exploration phase, supported by excavator trenching and road cuts. CASA and Rio Tinto also undertook comprehensive geochemical sampling of soils and rock outcrops over and peripheral to the deposit, resulting in a dataset with approximately 100 m by 100 m spatial coverage.

Much of the property geological information, as described in Item 7, has been derived from drillhole logging, interpretation of assay data, geophysical surveys, and the mapping of outcrop and trenches.

10.2 Recent exploration by the Company

Following Project acquisition, the Company completed several small-scale data collection programmes to ensure that supporting datasets were complete and of high quality.

In 2014, New-Sense Geophysics carried out a helicopter-borne magnetic and radiometric survey on behalf of the Company. A total of 4,424.1 survey line kilometres of data was collected at a 300 m spacing across the property. Results were used to support anomaly delineation, structural evaluation, and the identification of lithological trends.

Geochemical sampling campaigns of in-situ soils at a 500 m by 500 m grid spacing were also completed around the outer extents of the concessions.

In 2019, a high-resolution topographic survey was acquired via WorldView3. It was taken over a 12 km by 23 km area of the deposit at 0.5 m resolution, and over the wider surrounding areas at 3 m resolution.

ITEM 11 DRILLING

Most of the Project drilling activities were carried out prior to the Company's acquisition of the Project. Information is provided herein in the context of it being relied upon for the current Mineral Resource estimate. The issuer has verified the drillhole core logging data by check re-logging and by check assaying.

11.1 Drilling overview

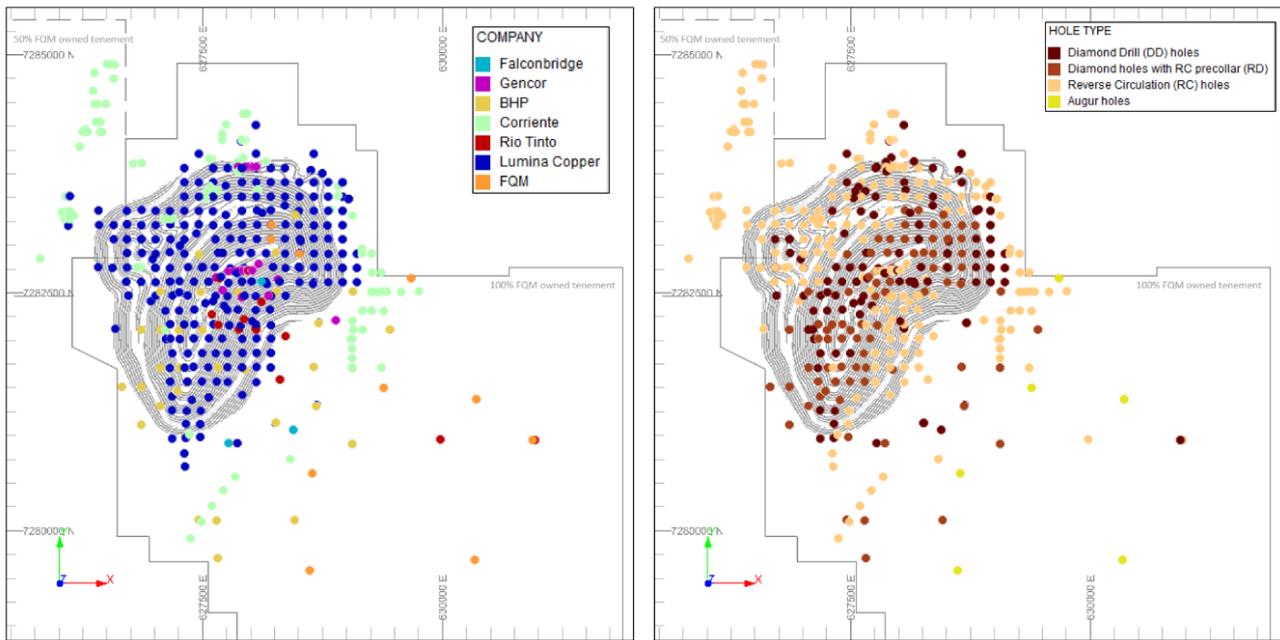
A total of 484 drillholes, for 172,031 metres, have been drilled in support of defining the mineralisation extents (Table 11-1). Of these, 44 holes were drilled outside of Company owned concessions for the purpose of freshwater exploration and for collecting geotechnical data for proposed infrastructure sites.

Figure 11-1 shows the location of the 440 holes drilled within Company held concessions. Approximately 28% of diamond (DD) holes were pre-collared using reverse circulation (RC) drilling.

Table 11-1 Summary of drillholes in the Taca Taca Project database

Year	Company	Drilling method	Number of drill holes	Total number of drilled metres
1975	Falconbridge	DD	3	529
1994	Gencor	RC	18	1,606
1996	BHP	DD	4	1,651
1997	BHP	DD	31	9,950
1998	Corriente	DD	14	3,325
1999	Corriente	RC	80	4,428
1999	RioTinto	RC	9	3,338
2008	RioTinto	DD	8	4,877
2010	Lumina	DD	5	3,437
2011	Lumina	DD	74	48,069
2011	Lumina	RC	21	4,866
2012	Lumina	DD	76	47,357
2012	Lumina	RC	107	33,942
2018	FQM	DD	14	3,041
2019	FQM	DD	4	1,455
2019	FQM	Augur	16	160
SUBTOTAL		RC	155	43,752
		DD	313	128,119
		Augur	16	160
TOTAL		Holes	484	172,031

Figure 11-1 Drillhole collar locations coloured by company (left) and hole type (right), relative to life of mine pit area and concession boundaries



11.1.1 Historical drilling

Prior to acquisition by Lumina, five different companies had carried out exploratory drill campaigns on the property (Table 11-1).

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.

Between 1996 and 1997, BHP drilled 35 diamond holes (including two re-drills) at an approximate 400 m by 400 m grid spacing into the porphyry. 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. 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 and Rio Tinto, and also 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.

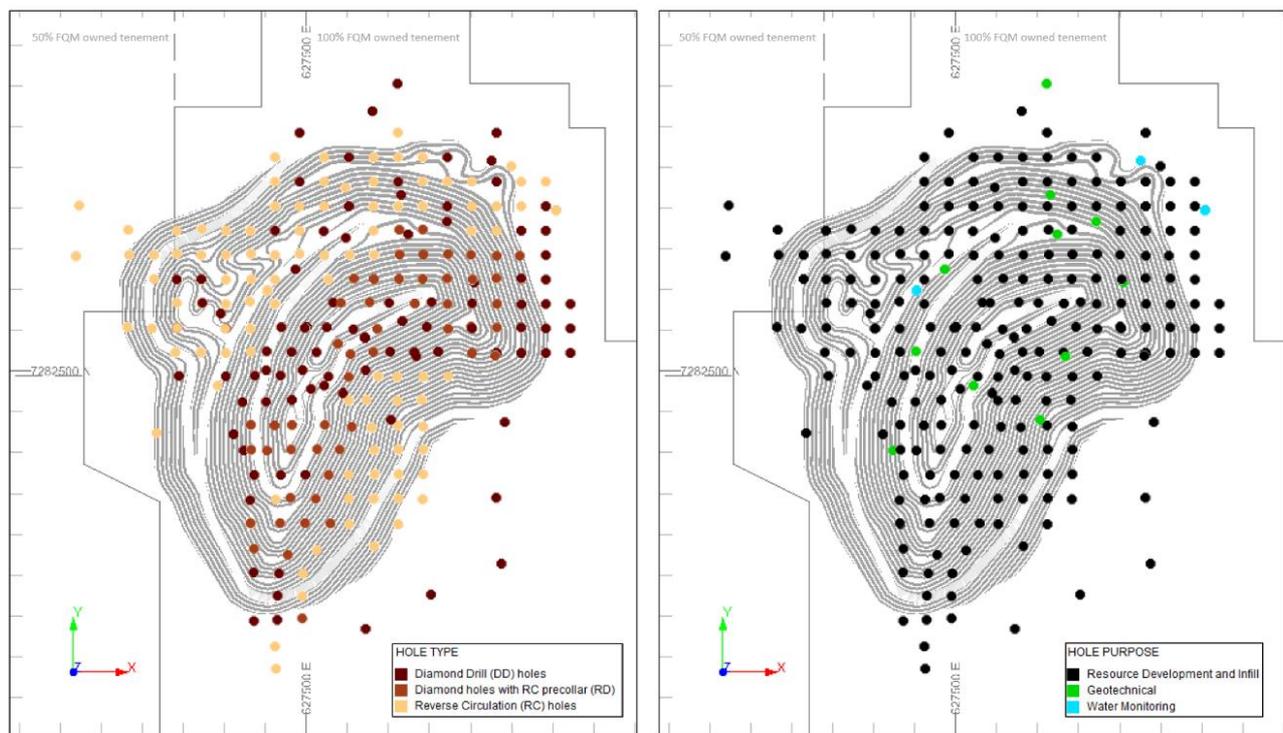
11.1.2 Drilling by Lumina

Lumina completed a total of 283 drillholes (137,671.5m) across the Project area during a 2010 to 2012 drill campaign. This comprised 155 diamond drilled (DD) holes and 128 reverse circulation (RC) holes, and included fifteen geotechnical holes and four water monitoring holes (Figure 11-2).

Most drillholes targeted the deeper porphyry and were completed along a set of east-west trending sections, on a nominal 150 m by 150 m grid spacing. Diamond holes were collared with a standard HQ sized core barrel and drilled to a standard run length of 3 m. Most diamond holes were cased to a NQ core diameter at depth. Shorter runs of 1 m to 1.5m were sometimes used in poor ground conditions to maximize recovery. 52 diamond holes in the central part of the deposit were pre-collared using reverse circulation drilling to depths just above the leached-mineralised contact.

As standard procedure, drill core was logged for lithology, weathering, alteration, mineralisation and structure. Diamond holes were also logged for geotechnical data, including core recovery, rock quality, fracture frequency and vein density, and vein angles. Samples were taken every 10 m for point load index tests and for density measurements.

Figure 11-2 Lumina drillhole collars coloured by hole type (left) and hole purpose (right), relative to life of mine pit area and concession boundaries

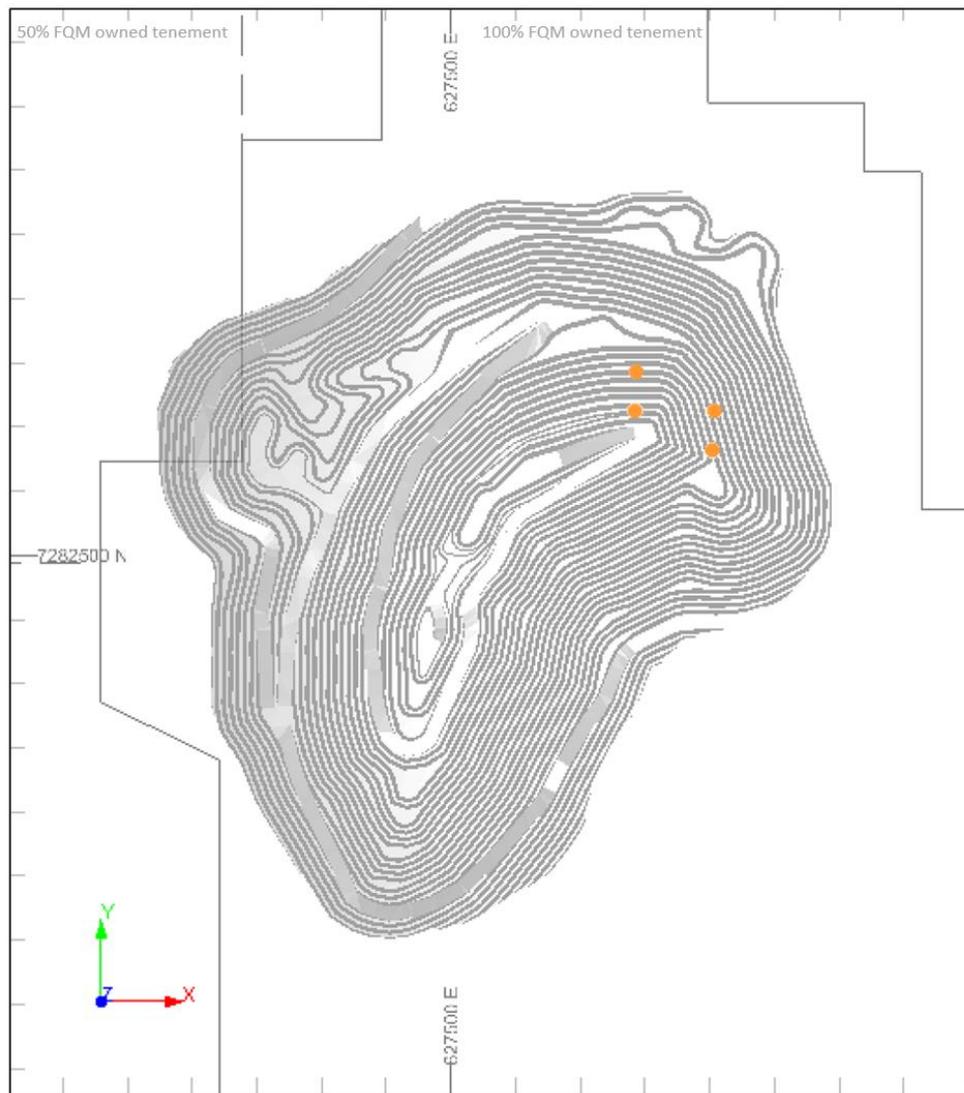


11.1.3 Drilling by the Company

During 2019, an additional four diamond holes were drilled by the Company as twins to Lumina drillholes. Their purpose was to provide additional samples for metallurgical testwork from material representing early plant feed.

Drilling and sampling procedures for these four drillholes were aligned to those used by Lumina. Samples were analysed using Inductively Coupled Plasma-Mass Spectrometry (ICP-MS) for 48 elements, including copper and molybdenum, and fire assay with AAS finish for gold. Results from these drillholes were included in the database for the Mineral Resource estimate.

Figure 11-3 Location of four holes drilled by the Company and used in the Mineral Resource estimate and for metallurgical test work, relative to the life of mine pit area and concession boundaries



Further drilling was undertaken to explore for fresh water sources and to collect geotechnical data at proposed infrastructure sites. The location and data collected from these drillholes was not relevant to the Mineral Reserve estimate.

11.2 Drilling database used in the Mineral Resource estimate

For Mineral Resource estimation purposes, the drilling database was clipped to include drilling information in the immediate vicinity of the deposit. The clipping limit coordinates used are listed in Table 11-2 and all drilling statistics referenced in this Technical Report are based on the clipped data.

Of the 484 drillholes in the database, 435 fell within the clipping limits, with 395 having assay data relevant to the Mineral Resource estimate (Table 11-3).

Table 11-2 Clipping limits applied to drillhole data

Constraint	Easting (mE)	Northing (mN)
Minimum	625,500	7,279,500
Maximum	630,000	7,285,000

Table 11-3 List of drill holes with assay data used in the Mineral Resource estimate

Year	Company	Number of holes	Hole Purpose	Hole Type*	Hole ID Sequence	Meters Drilled	Samples with assays	Main analytical method**
1996	BHP	4	Exploration	RD	TK01-TK04	1,651.45	806	Unknown
1997	BHP	29	Exploration	RD	TK05-TK25, TK27-TK33	9,422.60	4,398	Unknown
1998	Corriente	2	Exploration	DD	TK34, TK44	606.20	175	AAS
1999	Corriente	80	Exploration	RC	TK48-TK126	4,428.00	1,543	AAS
1999	Rio Tinto	8	Exploration	RC	CCR001-CCR007, ARI001	3,029.50	1,478	ICP-OES/MS
2008	Rio Tinto	8	Exploration	DD	TTBJ0001-TTBJ0008	4,876.70	1,572	ICP-OES
2010	Lumina	5	Exploration	DD	TTBJ10-01 - TTBJ10-5	3,436.85	1,422	ICP-OES
2011	Lumina	67	Resource Development	DD/RD	TTBJ10-06, TTBJ11-07 -TTBJ11-73	44,168.50	21,241	ICP-OES
2011	Lumina	1	Exploration	RC	TTEX01	200.00	100	ICP-OES
2011	Lumina	3	Exploration	DD	TTEX02, TTEX03, TTEX07	1,496.40	746	ICP-OES
2011	Lumina	4	Geotechnical	DD	TTGT01-TTGT04	2,404.35	1,200	ICP-OES
2011	Lumina	20	Resource Development	RC	TTRC11-01 - TTRC11-17, TTRC12-88/94/95	4,666.00	2,333	ICP-OES
2012	Lumina	6	Exploration	RC	FR12-05 - FR12-10	2,196.00	1,098	ICP-OES
2012	Lumina	2	Water Monitoring	RC	AVSP4S, AVSP5D	260.00	130	ICP-OES
2012	Lumina	11	Geotechnical	DD	TTTV1-TTTV11	5,905.47	2,865	ICP-OES
2012	Lumina	65	Resource Development	DD/RD	TTBJ11-72-TTBJ11-76, TTBJ12-77-136	41,451.95	20,468	ICP-OES
2012	Lumina	76	Resource Development	RC	TTRC12-18 -TTRC12-97	26,988.00	13,492	ICP-OES
2019	FQM	4	Metallurgical	DD	TTBJ19-137-TTBJ19-140	1,455.00	736	ICPMS
TOTAL		395				158,642.97	75,803	

*DD=Diamond drilled holes, RC= Reverse Circulation holes, RD=Diamond drilled hole with a RC pre-collar.

**Main method refers to method used for majority of elements including copper. All gold analyses were by fire assay with AAS finish.

Holes drilled prior to 2008 typically have incomplete procedural records, with limited to no QAQC data available. Checks of these drillholes against adjacent drillhole samples, via visual comparisons to remaining core and interrogation of historic reports, suggest sample data is similar and therefore of adequate quality. Any risk of these holes misrepresenting the Mineral Resource estimate is considered minimal since most were informing shallow areas, within the leached cap or peripheral to the deposit. These holes were therefore used to support the Mineral Resource estimate.

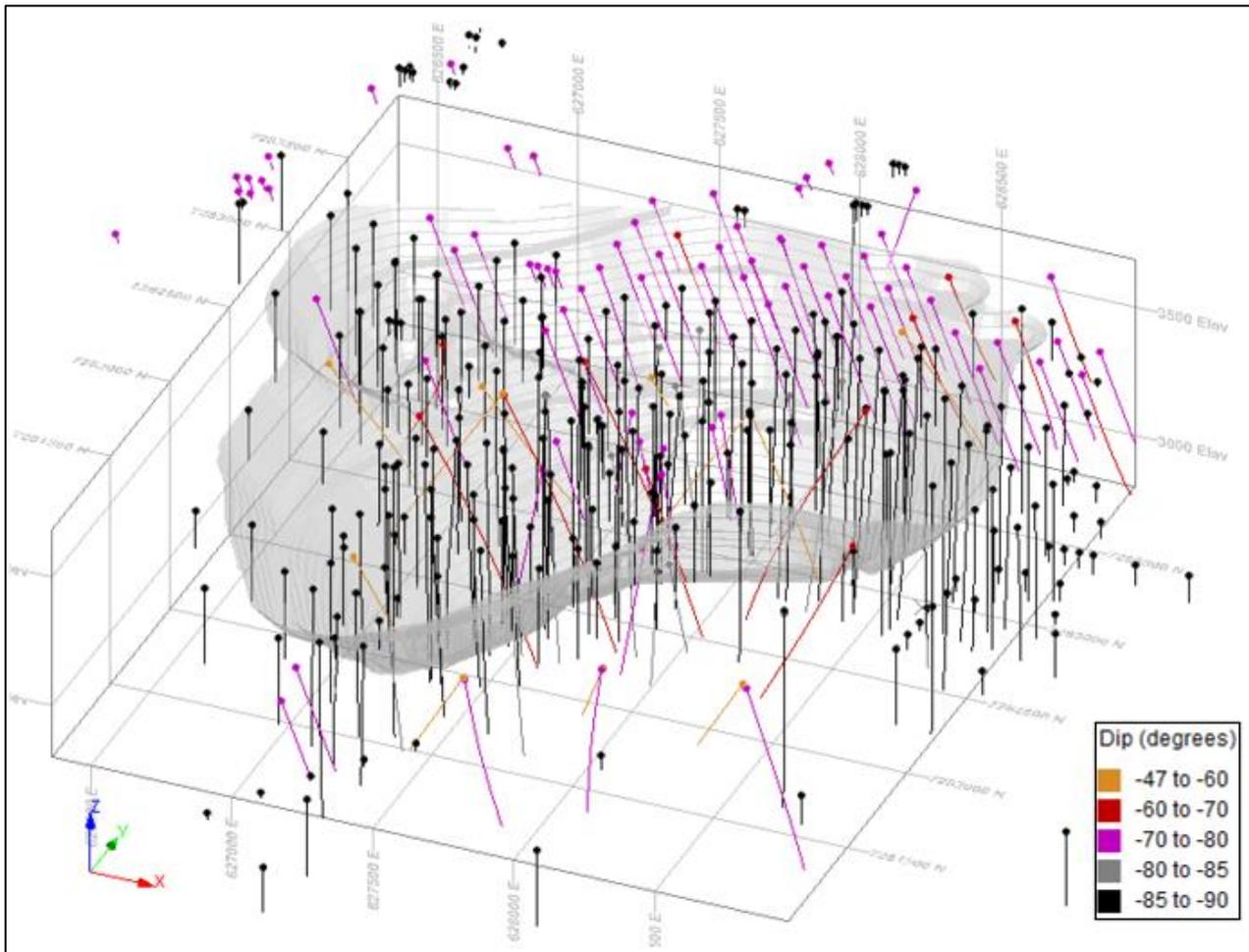
11.3 Drilling database

Drilling data is stored in the Company's single, secure SQL database. All historical data was migrated into this database with validation checks completed for consistent quality data. Collar coordinates were validated against a topographic surface together with field spot-checks completed by a Company geologist. A representative portion of assay records stored in the database were compared with original assay certificates for any data translation errors. During the estimation process, data was further validated using built-in geological software tools, with no significant issues noted.

11.4 Drilling orientation

More than half of the drillholes were drilled vertically (Figure 11-4). Shorter RC holes in the north of the deposit were typically inclined at -70° to the east to maximise the angle of intersection to mineralisation related to sub-vertical north-south trending structures. Geotechnical holes were drilled at variable inclinations and directions according to the orientation of their target structure. Most diamond holes range between ~350 m and ~900 m in depth, with shorter RC holes typically located around the periphery of the deposit. The deepest hole on the property was drilled by Rio Tinto in 2008 to 1,153 m.

Figure 11-4 Drillhole traces coloured by range of dip



11.5 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 formulae applied to convert these coordinates to WGS84 and to adjust ellipsoid elevations to orthometric elevations is summarised in Table 11-4.

Table 11-4 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

11.6 Collar surveys

For holes drilled since 2010, collar co-ordinates were initially located using a handheld GPS and then surveyed using a differential Trimble GPS after hole completion. 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 and no discrepancies were identified.

11.7 Downhole surveys

Downhole survey data only exists for diamond holes drilled since 2008. Earlier drillholes and all RC holes have a single planned hole orientation recorded. Samples from holes with no downhole survey data comprise approximately 11% of the total samples used in the Mineral Resource estimate. Checks were conducted to compare samples from historical holes to Lumina samples within 30 m of each other. Results suggest that the respective grade distributions are similar and that survey records compared with historical report data were similarly reliable.

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.

11.8 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 159 diamond holes drilled since 2010, 88% of the core has a recovery greater than 85%. Core with recovery lower than this is predominantly from the weathered leached cap. Overall median core recovery was 98%.

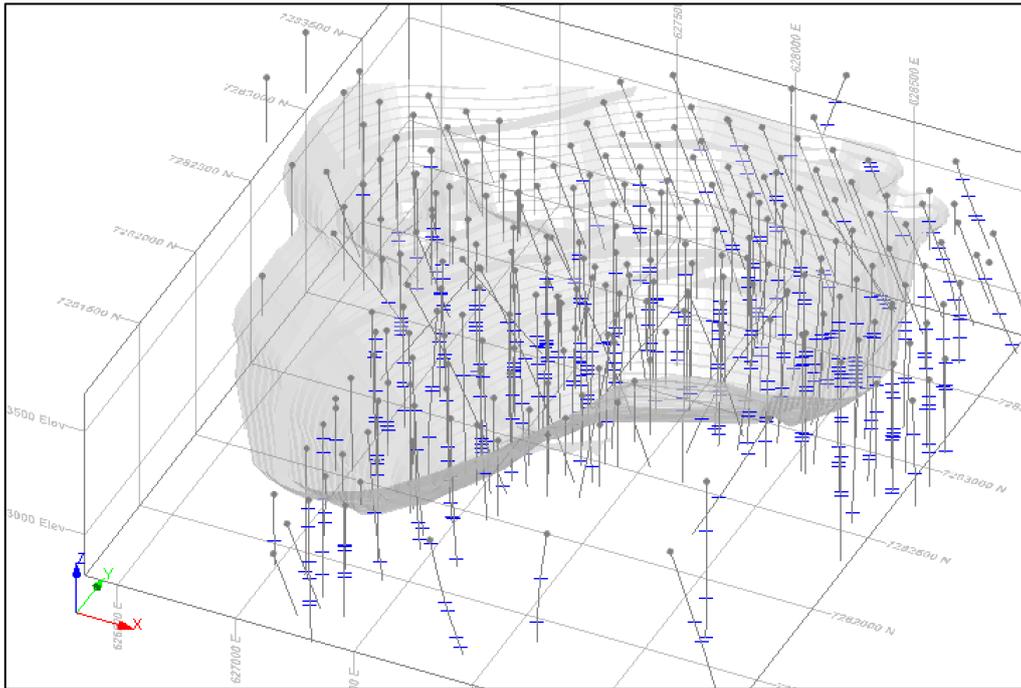
There are no drilling or sampling recovery concerns that are material to the 3D position, accuracy, and quality of the logging and assay grades obtained from drillhole samples.

11.9 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 11-5). There were sufficient samples per weathering and key lithology domains for a reasonable density estimate.

Figure 11-5 Spatial distribution of samples, shown in blue, taken from Lumina drillholes for density measurement



11.10 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 Company owned warehouse, as a permanent record. Storage facilities are covered and secured.

In contrast, not all RC chip samples remain available for RC holes drilled prior to 2010. Most of these holes are shallow and have limited impact on the estimate within the main mineralised zone. All RC holes drilled after 2010 have chips preserved on 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.

11.11 Geological data

Qualified geologists logged all drill core and RC chips for lithology, weathering, alteration, vein density and mineralisation. Several drill core re-logging campaigns were completed by Company geologists to validate earlier logging and to collect extra information on mineralisation styles and associations. It was determined that original logging records are reliable.

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.

11.12 Factors materially affecting the Mineral Resource estimate

It is the opinion of the QP that diamond and RC drilling was reasonably undertaken and the holes were satisfactorily oriented. Drillhole spacing provides good coverage of the available mineralisation, with limited areas of clustered drilling evident. Drillhole orientations are sufficiently variable across the deposit for reducing potential biasing risks arising from preferred orientation drilling. Drilling depths have provided sufficient extent and sampling of the prevailing geology and mineralisation for the Mineral Resource estimate. Together with safe and secure data management, the drill data supports accurate 3D downhole sample positions.

On the basis of the above, there are no known drilling factors that could materially affect the accuracy and reliability of samples used in the Mineral Resource estimate for this Technical Report.

ITEM 12 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following information describes the sampling activities for drill holes, the data from which was used in the Mineral Resource estimate. This includes all holes drilled between 1996 and 2019. No written record of sample preparation, analytical methods, or analytical results is available for holes drilled prior to this, by Falconbridge or Gencor.

12.1 Diamond core and RC chip logging

Geological logging of all diamond and RC holes was completed and recorded using the standardised logging code systems of each previous owner.

Diamond core drilled between 2010 and 2012 had structural, recovery, and RQD measurements recorded. Drill core was photographed, and point magnetic susceptibility and SWIR spectrometer readings were taken. Prior to sample preparation for elemental analyses, centimetre-scale samples were taken every 10 m down-hole for density measurements and for point load testing.

Historic data was predominantly stored in MS Excel workbook templates. These were later imported, formatted, standardised and validated by the Company into the Maxwell DataShed SQL datashed.

12.2 Core sampling

Written descriptions of the sampling procedures followed by BHP (1996-1997) and CASA (1998) are incomplete, however, visual checks and database data suggests the processes and resulting data were similar to industry standard practice. The procedures followed by Rio Tinto (2008), Lumina (2010-2012), and the Company (2019) are detailed below.

Upon completion of logging, core was typically marked out to 1 m sample lengths and a line was drawn down the centre. A diamond core saw was used to cut core in half along the centre-line mark and at a nominal 2 m sample length. One half was bagged for dispatch and the other retained in the core box as a permanent record. Several samples per hole were taken from the core cutting table sludge (rock dust plus water) to monitor any loss of metal to fines.

As a standard, the full length of each hole was sampled. However, in several earlier drilled holes, the topmost portion of the hole was not sampled in zones where weathering and oxidation were most intense.

All bagged samples were assigned a sample ticket. In 2008, a lab-generated sample ticket was inserted into the plastic bag with a second one stapled onto the bag closure. Since 2010, barcoded sample tags were included in each sample bag. Sample bags were secured, and sample numbers were written on each. The bags were placed into a secured larger mesh sack for transport to the sample preparation laboratory and dispatched once the full hole was sampled.

The remaining core was placed into storage at covered, secure facilities in Salta.

12.3 RC sampling

Descriptions of the sampling procedures followed by BHP (pre-collar of diamond holes, 1996 to 1997), CASA (1999), and Rio Tinto (1999) are not available, although evidence suggests practices in place were similar to those of the more recent RC drilling. Most of the historic RC holes are shallow or outside of the planned pit extents and, as such, will have a limited impact on the Mineral Resource estimate. RC sampling procedures followed by Lumina (2011 to 2012) were confirmed and are detailed below.

A drill cuttings (chips) sample was retrieved from the drill-rig cyclone at every 2 m interval. The full length of each hole was sampled. The sample was split three times using a Gilson adjustable sample splitter, to

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produce two samples weighing approximately 6 kg to 10 kg each. The sample splitter and tools used in the sampling process were cleaned with compressed air after every sample to control cross-contamination.

Samples were secured into bags, with one sample dispatched to the laboratory and the other stored for reference or future sampling. An additional <100 g sample of coarser chips was fixed to a chip sheet for logging.

If samples retrieved were wet, samples were split using a rotary wet splitter. One half of the material was collected in a bucket and left to decant. Once the sample appeared dry, the sample was divided in two with one half sent to the laboratory and the other placed in storage.

12.4 Sample preparation

Records of sample preparation methods for BHP and CASA drillholes are no longer available, although evidence suggests similar industry standard processes were likely to have been followed.

12.4.1 Rio Tinto 1999 RC programme

Chip samples were prepared by Bondar Clegg's laboratory in Mendoza using the 'Large Pulp Preparation' procedure. The method involved crushing the whole sample to -80 mesh (177 µm) and obtaining a 1 kg split to be pulverised.

12.4.2 Rio Tinto 2008 programme

Core samples were prepared by the Alex Stewart preparation facility in Mendoza, as follows:

- samples were weighed upon receipt and sent for drying
- each sample was crushed to 80% passing 2 mm using a jaw crusher
- crushed sample was split in a riffle splitter until a 1.2 kg sample mass was retrieved
- sample was pulverised to produce a pulp sample with 85% passing 75 µm

Approximately 200 g of pulp was retrieved for use by the analytical laboratory.

12.4.3 Lumina 2010 to 2012 programme

Lumina used both ALS Minerals (ALS) and the Alex Stewart laboratories in Mendoza. The sample preparation method for both drill core and RC chips was as follows:

- samples were weighed and barcoded sample ticket was scanned upon receipt
- each sample was crushed to 70% passing 2 mm using a jaw crusher
- crushed sample was split in a riffle splitter until a 1 kg sample mass was retrieved
- sample was pulverised to produce a pulp sample with 85% passing 75 µm

Sample was split to around 200 g for use by the analytical laboratory.

The chain of custody for Lumina and for Rio Tinto's 2008 drill campaign had all samples sealed and packed onto a covered pick-up truck. They were transported by road, with a driver employed by the relevant company, to the chosen laboratory. The laboratory was notified of the sample numbers in advance of delivery and a copy of the sample list was kept on site. Upon delivery, the laboratory sent confirmation of the samples received, their condition, and any discrepancies.

No irregularities were noted during these programmes.

12.4.4 FQM 2019 programme

Core samples were prepared at ALS Mendoza using identical methods to those of the Lumina 2010 to 2012 programme.

12.5 Sample analysis

A summary of drill campaigns and associated analytical laboratories used for sample analyses is provided in Table 12-1. Original assay certificates were not available for samples submitted prior to 2008; the previous owners' results were in spreadsheet format only.

Table 12-1 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 values with 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 values with AAS Fire assay (AAS) for Au Sequential leach Cu for a 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 FQM	ALS, Lima	4 acid digest with ICP-OES/MS finish Fire assay (AAS) for Au Sequential leach Cu	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, Sc, Se, Sn, Sr, Ta, Te, Th, Ti, Tl, U, V, W, Y, Zn, Zr

12.6 QAQC protocols

Reasonable QAQC procedures have been verified and resulting data is available for the 2008 sample programs to date. This represents approximately 89% of samples used in this Mineral Resource estimate.

References to QAQC protocols prior to 2008 were sourced from reports and information provided by the previous owners. Results from samples prior to this mostly inform areas peripheral to the main mineralised zones and demonstrate good agreement with proximal samples.

12.6.1 BHP 1996 to 1997 programme

Historic reports indicate that BHP submitted a quarter core duplicate sample for every 20 samples, to the original laboratory, and 300 coarse residual duplicates to Bondar Clegg (location unknown), for check assaying. Results from this quality control programme were not available for review.

In 2003, a representative from AMEC collected eleven samples from archived NQ core at matching sample intervals to the original assay programme and dispatched them for check assaying. Historic documentation reports show good agreement between these sample results.

The Company completed checks by comparing the BHP 1996 - 1997 samples to Lumina samples located within 30 m of each other. Comparisons demonstrated similar values and grade distributions. Results were deemed of sufficient quality for use in the Mineral Resource estimate.

12.6.2 CASA 1998 to 1999 programme

Whilst historic reports indicate pulp duplicates were submitted routinely to an umpire laboratory for check assaying, details and results for this programme are not available.

Most of this holes from this programme are shallow or outside the planned pit extents and therefore will have limited impact on the Mineral Resource estimate. Sample values within 30 m of Lumina samples were compared. The comparison suggests sample results are of adequate quality to support the estimate.

12.6.3 Rio Tinto 1999 RC programme

Information provided by Rio Tinto indicates systematic QAQC procedures were followed, as below:

- one field duplicate was inserted for every 12 samples
- pulp duplicates, Certified Reference Material (CRM), and blanks were added, but at an unknown rate.

Whilst results are not available, historic reports suggest acceptable quality data. Although these sample results were used in the Mineral Resource estimate, most were located in shallow zones with limited impact on main mineralised zones.

12.6.4 Rio Tinto 2008 DD programme

Information provided by Rio Tinto indicates systematic QAQC procedures were followed, as below:

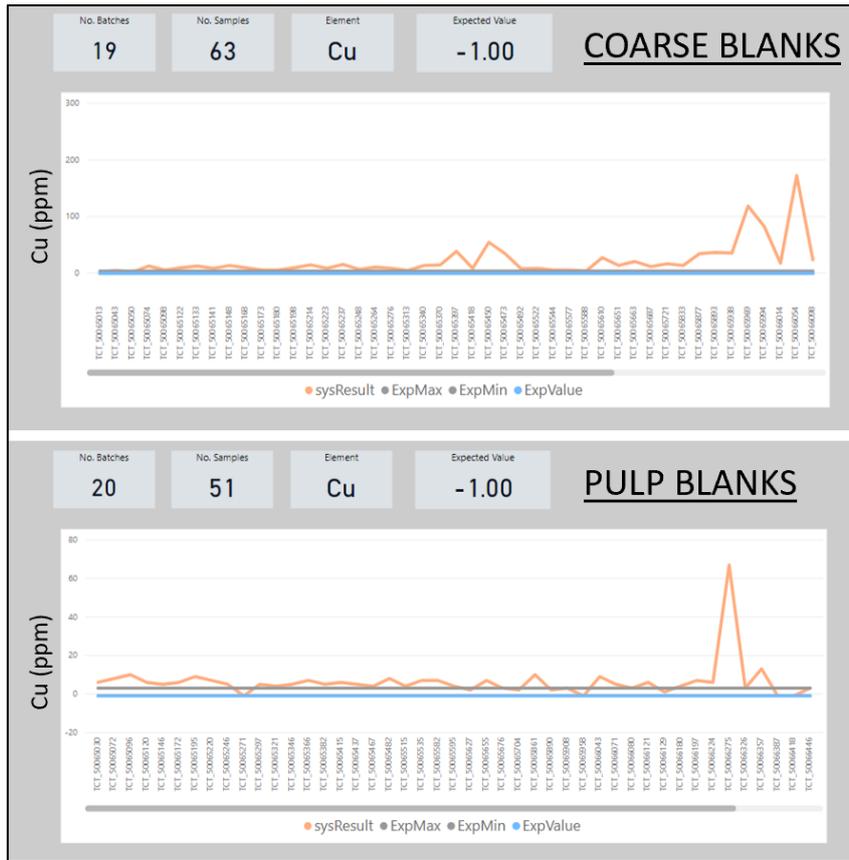
- CRM samples, certified for Cu and Au, at a rate of 3% to 4%
- blank quartzite samples, both chipped and pulverised, at a rate of 2% to 3% each
- coarse and pulp duplicate samples taken at a rate of 2% to 3% each
- seven half core field duplicates samples submitted from four drill holes

A total of 133 pulp duplicate samples were also taken from hole TTBJ0003 and sent to ALS in Lima. These were analysed using 4-acid digestion with AAS as a check on the primary laboratory. Sludge samples from the diamond saw table used to cut core were also routinely analysed to monitor for any disproportionate loss of metal to fines.

Only the results from submitted blank and duplicate samples are available to the Company and are detailed below. Although CRM and other check samples results are not available, historic reports and proximal samples suggest data quality is acceptable.

Returned blank sample values for copper analyses demonstrate that contamination was mostly controlled during sample preparation (Figure 12-1).

Figure 12-1 Blank samples control chart for Rio Tinto 2008 diamond drill campaign, both coarse crush (top) and pulp material (bottom)



Duplicate samples show acceptable precision for copper, gold, and molybdenum analysis, with an expected increasing scatter for higher sample values (Figure 12-2, Figure 12-3, Figure 12-4). Field duplicates for gold analysis show poor correlation against the original results, due largely to low values and the inherent nugget effect of gold mineralisation.

Figure 12-2 Scatter plots showing original copper analysis values against field, coarse, pulp, and laboratory duplicate results for Rio Tinto 2008 diamond drill campaign

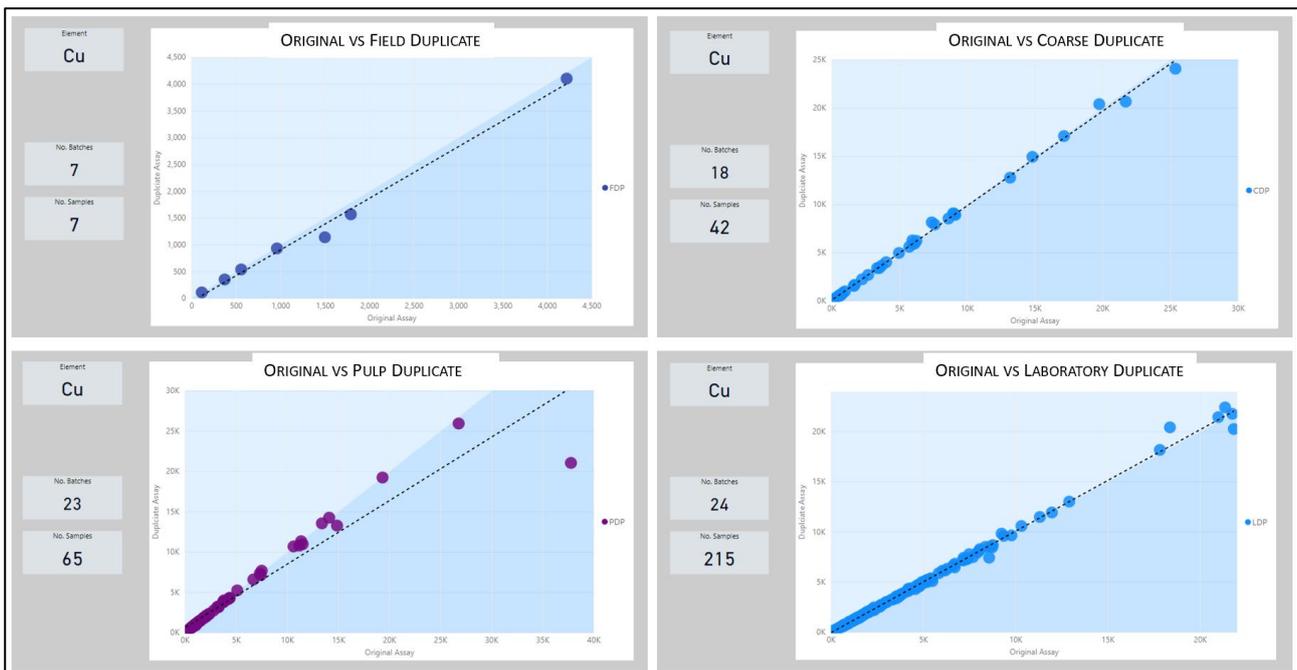


Figure 12-3 Scatter plots showing original gold analysis values against field, coarse, pulp, and laboratory duplicate results for Rio Tinto 2008 diamond drill campaign

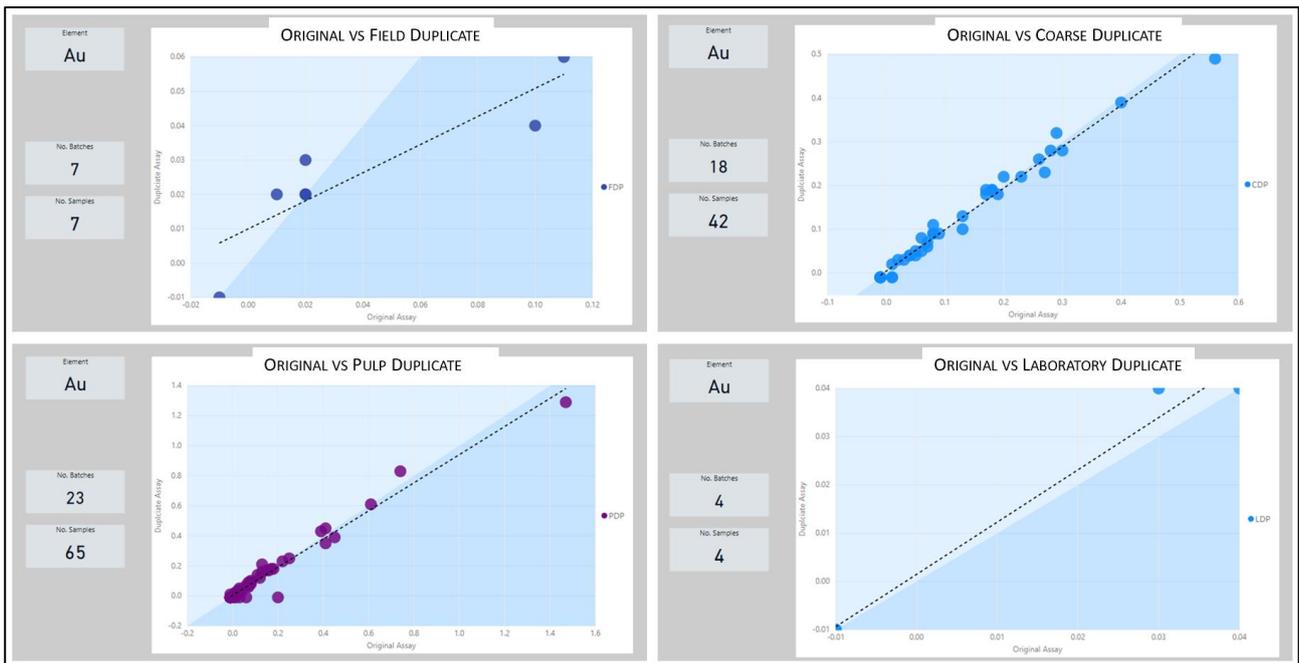
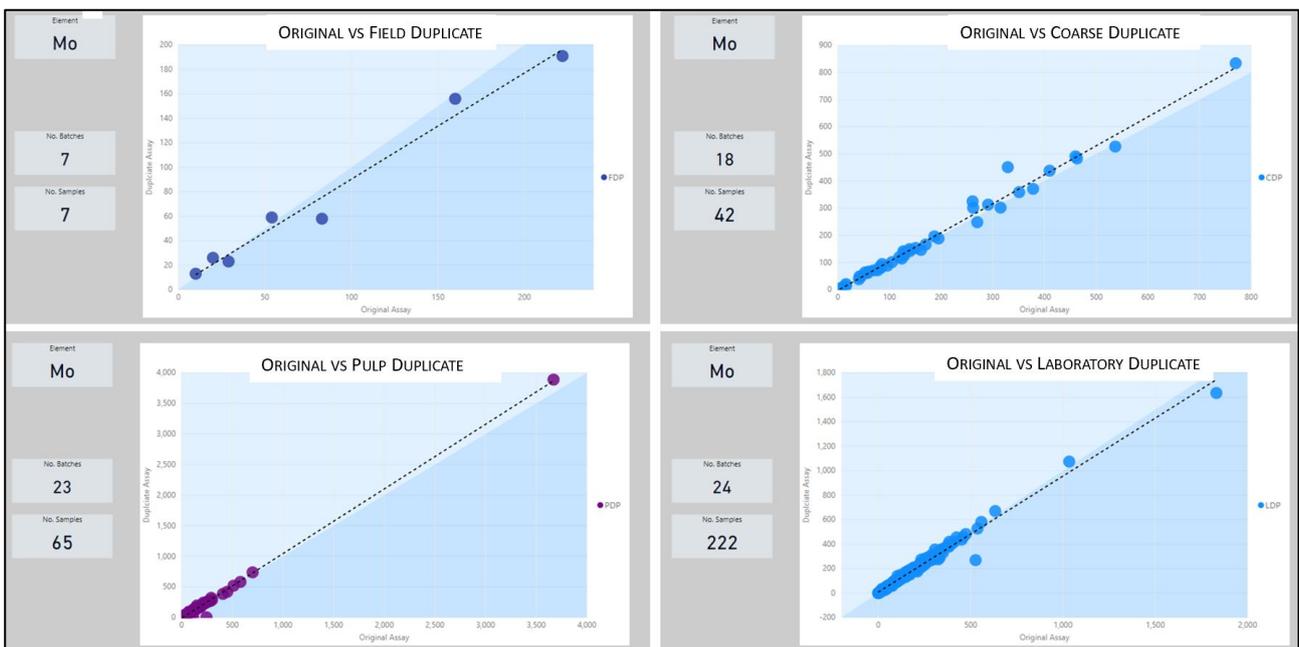


Figure 12-4 Scatter plots showing original molybdenum analysis values against field, coarse, pulp, and laboratory duplicate results for Rio Tinto 2008 diamond drill campaign



12.6.5 Lumina 2010 to 2012 programme

Systematic industry standard QAQC protocols were in place for the full duration of the Lumina drill programme, as follows:

- CRM samples, certified for Cu, Au, and Mo, were inserted at a rate of approximately 2%
- three types of CRM were used to cover a range of metal concentrations
- coarse blank material and pulverised blank material were each inserted at a rate 2%
- both coarse duplicate samples and pulp duplicate samples were taken at a rate 2%

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Returned coarse and pulp blank sample values demonstrate that contamination was adequately controlled during sample preparation of both diamond core (Figure 12-5) and RC chip samples (Figure 12-6). Duplicate samples show acceptable precision for copper, gold, and molybdenum analysis of both the diamond core samples (Figure 12-7) and RC chip samples (Figure 12-8). The inserted CRM sample analysis results indicate acceptable primary laboratory accuracy during both diamond (Figure 12-9) RC (Figure 12-10) drill programmes. Most CRM assayed values fall within the expected certified values limits, with few showing evidence for sample mislabelling.

Figure 12-5 Blank samples control charts for Lumina diamond drill campaign, both coarse crush (left) and pulp material (right)

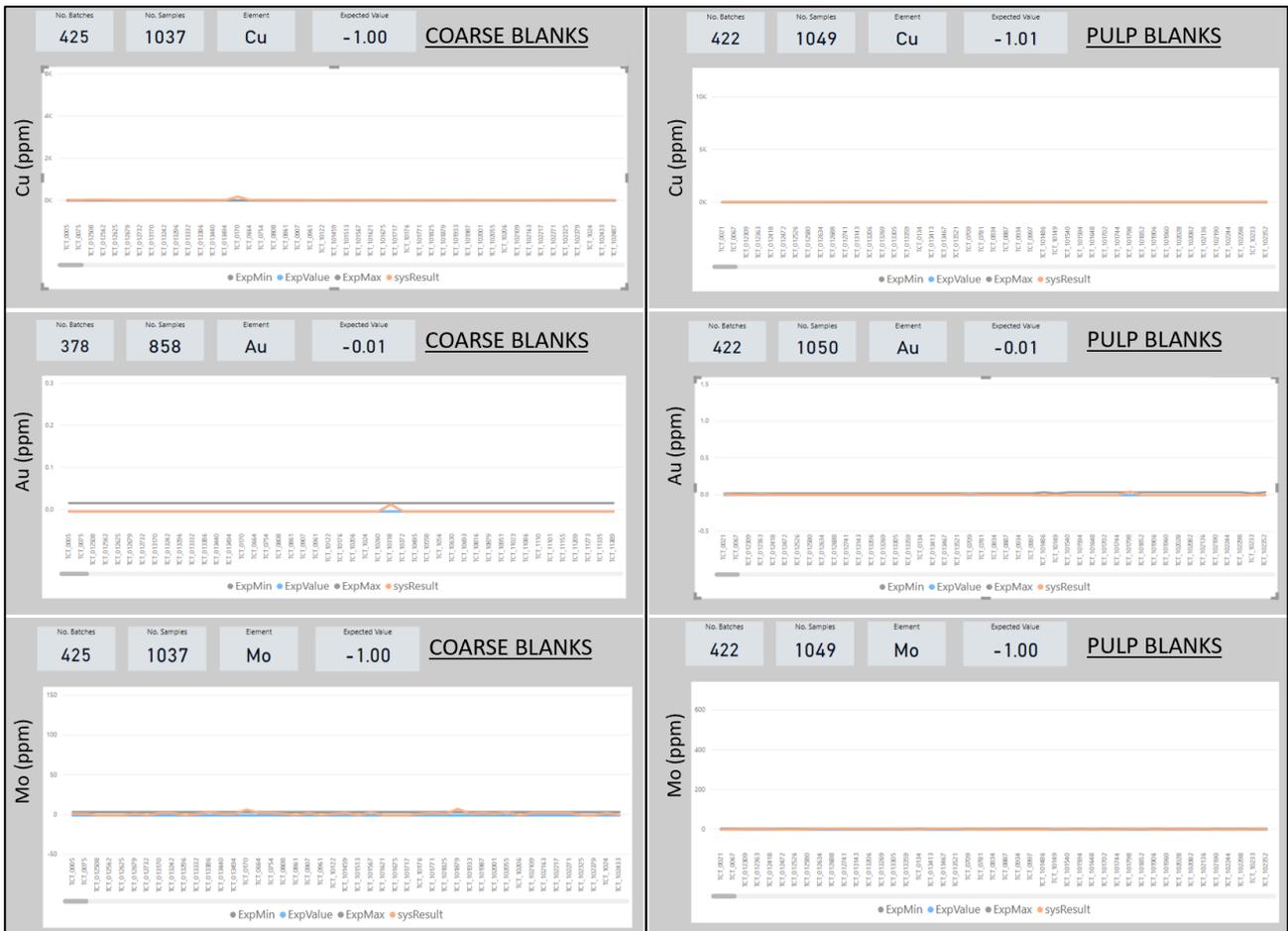


Figure 12-6 Blank samples control charts for Lumina RC drill campaign, both coarse crush (left) and pulp material (right)

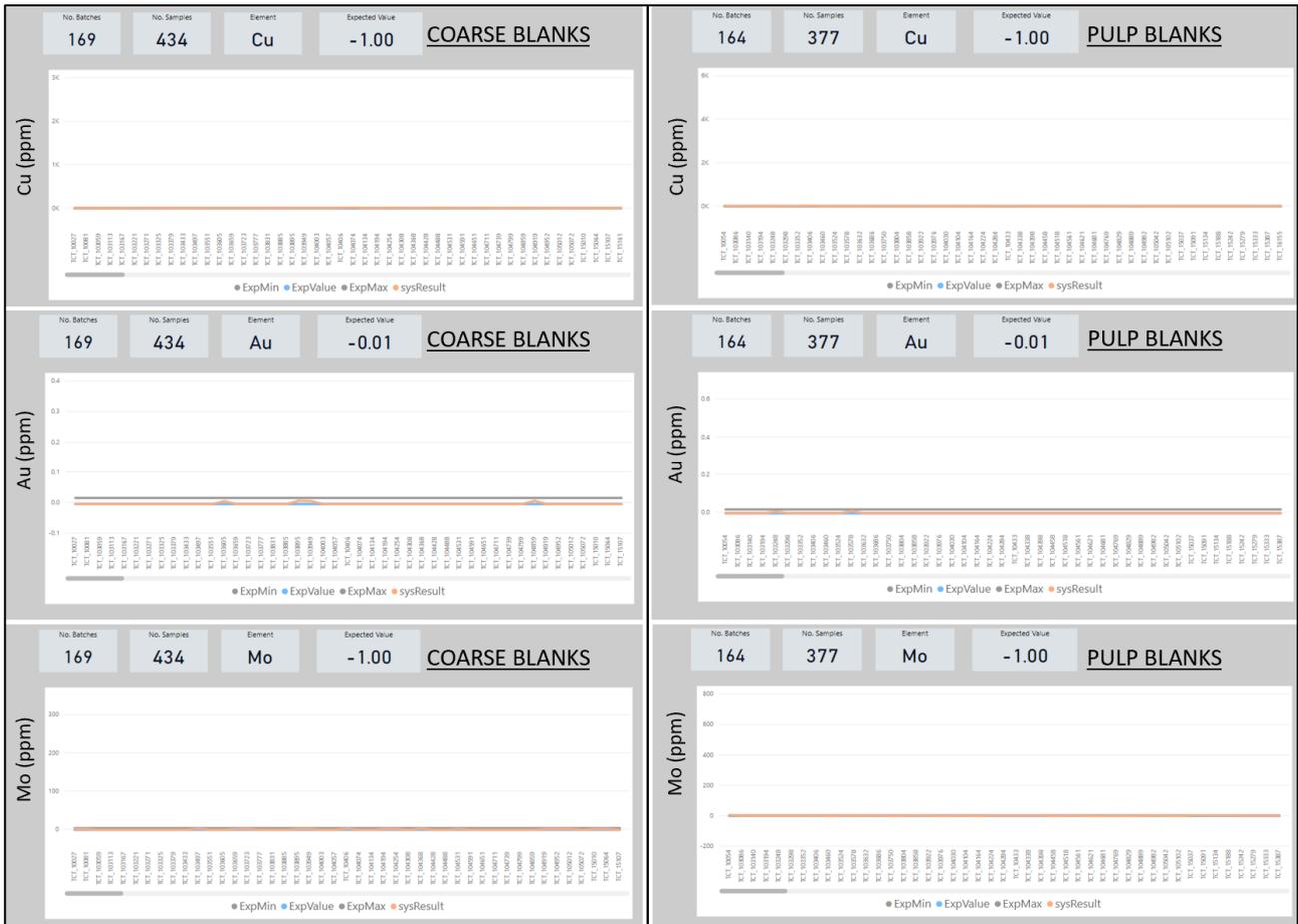


Figure 12-7 Scatter plots showing original Cu, Au, and Mo analyses values against coarse, pulp, and laboratory duplicate results for Lumina diamond drill campaign

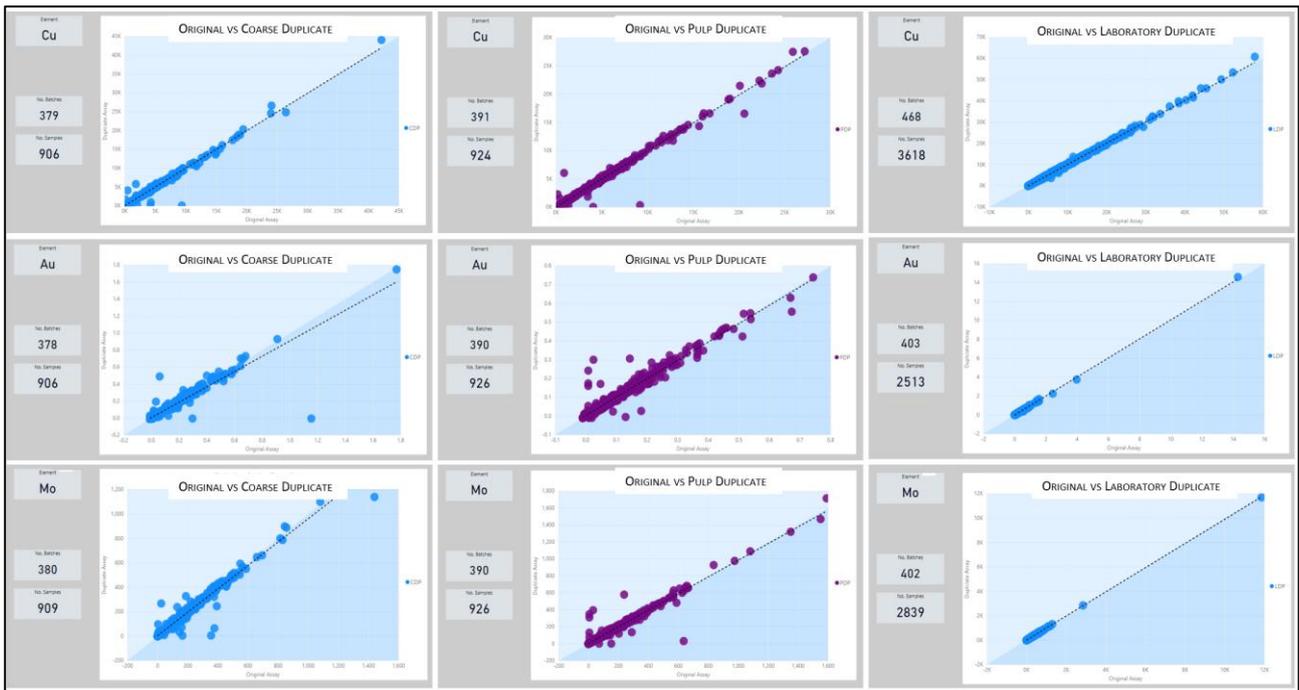


Figure 12-8 Scatter plots showing original Cu, Au, and Mo analyses values against coarse, pulp, and laboratory duplicate results for Lumina RC drill campaign

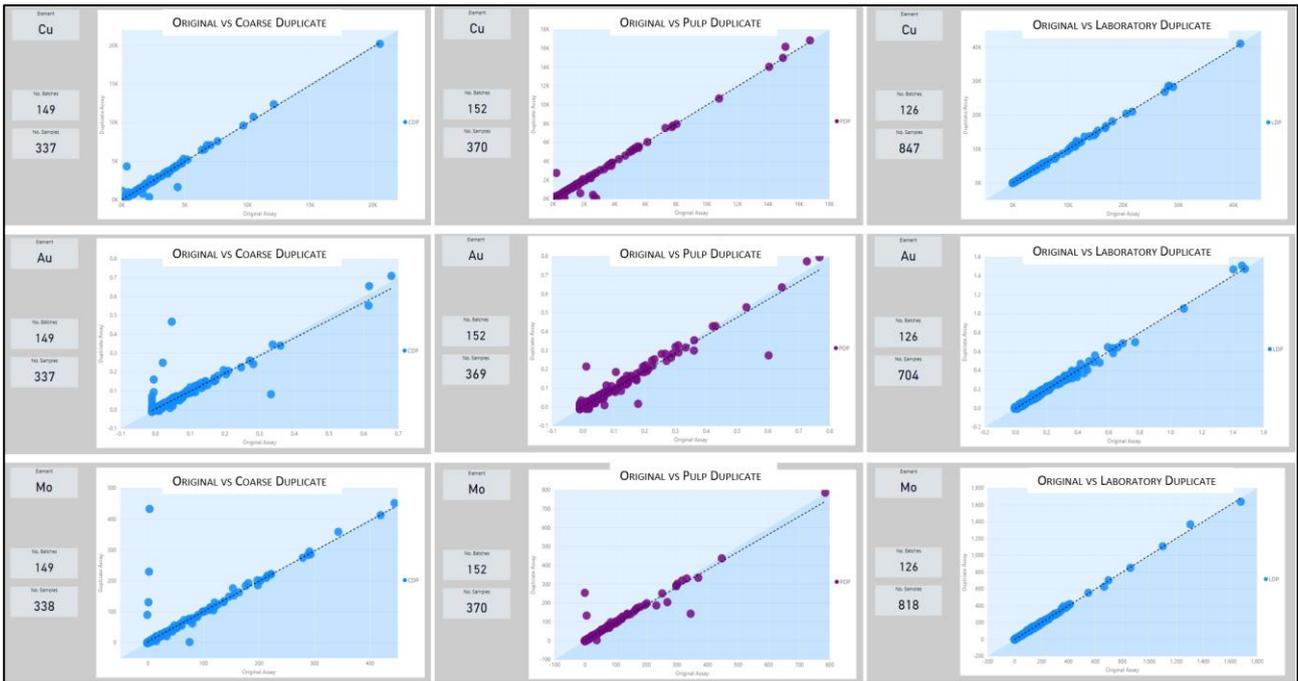


Figure 12-9 Control charts showing Cu, Au, and Mo results from the 3 CRM samples analysed during Lumina diamond drill campaign

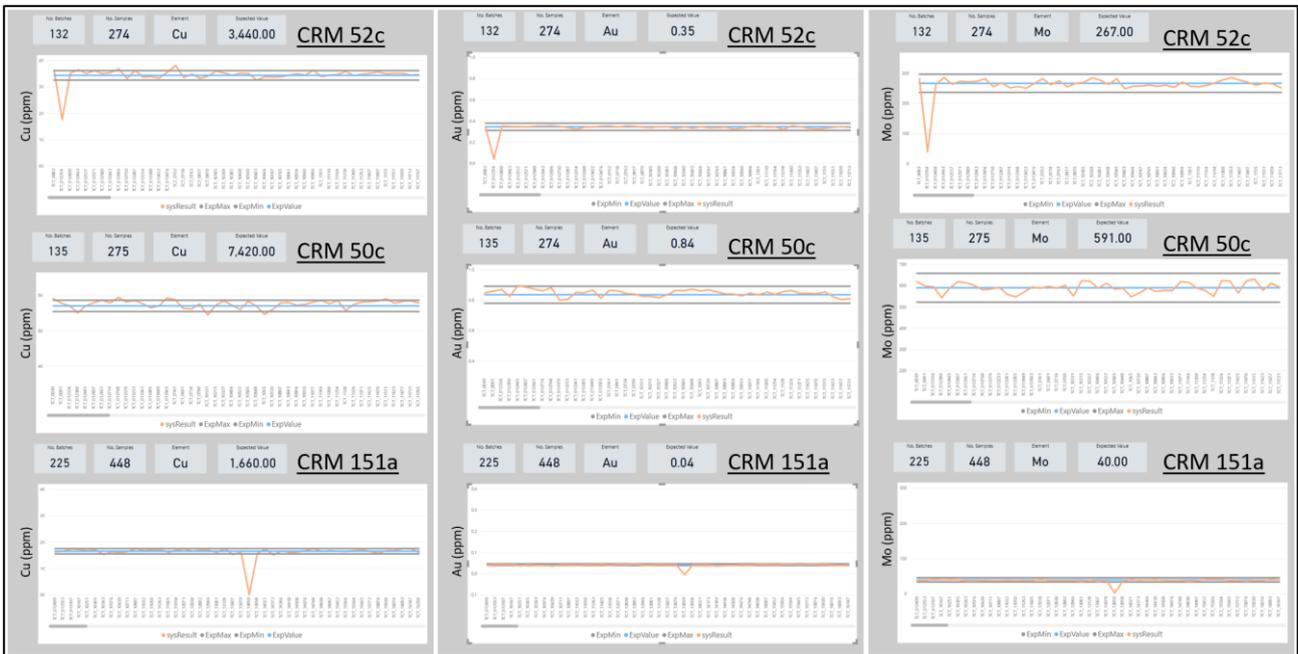
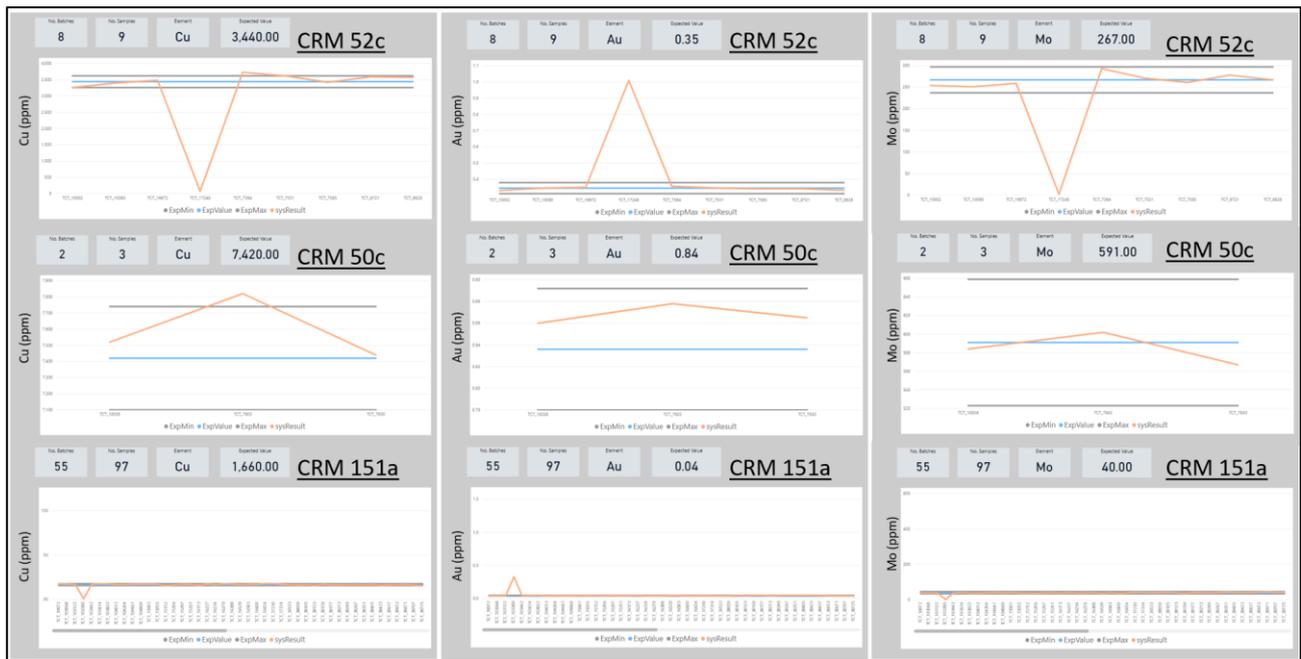


Figure 12-10 Control charts showing Cu, Au, and Mo results from the 3 CRM samples analysed during Lumina RC drill campaign



12.6.6 FQM 2019 programme

QAQC protocols for the four recently drilled metallurgical holes were as follows:

- CRM samples, certified for 48 elements including Cu, Au, and Mo, were inserted at a rate of 1 in 30
- coarse blank material was inserted at a rate of 1 in 50 samples
- coarse duplicates were taken at a rate of 1 in 50 samples

A visual check between the results and the Lumina twin core samples was performed and results showed expected agreement. Significant portions of these holes were dispatched as half core to ALS Kamloops (Canada) for metallurgical testwork with additional quarter core dispatched to ALS Mendoza for routine elemental analysis.

Returned coarse blank sample values demonstrate that contamination was adequately controlled during sample preparation (Figure 12-11). Duplicate samples show acceptable precision for copper, gold, and molybdenum analysis (Figure 12-12). CRM sample analysis results indicate acceptable primary laboratory accuracy (Figure 12-13). All CRM assayed values fall within the expected certified value limits.

Figure 12-11 Blank samples control charts for FQM diamond drill campaign analysed for Cu, Au, and Mo

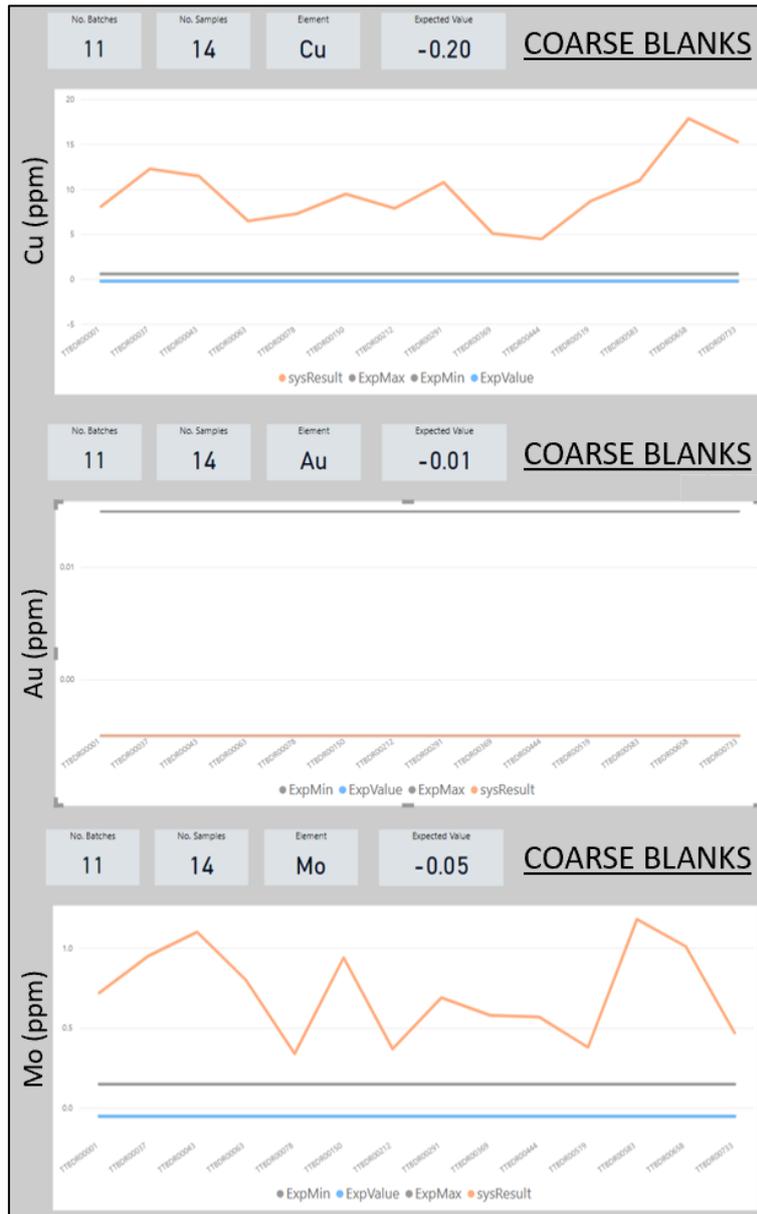


Figure 12-12 Scatter plots showing original Cu, Au, and Mo analyses values against coarse and laboratory duplicate results for FQM diamond drill campaign

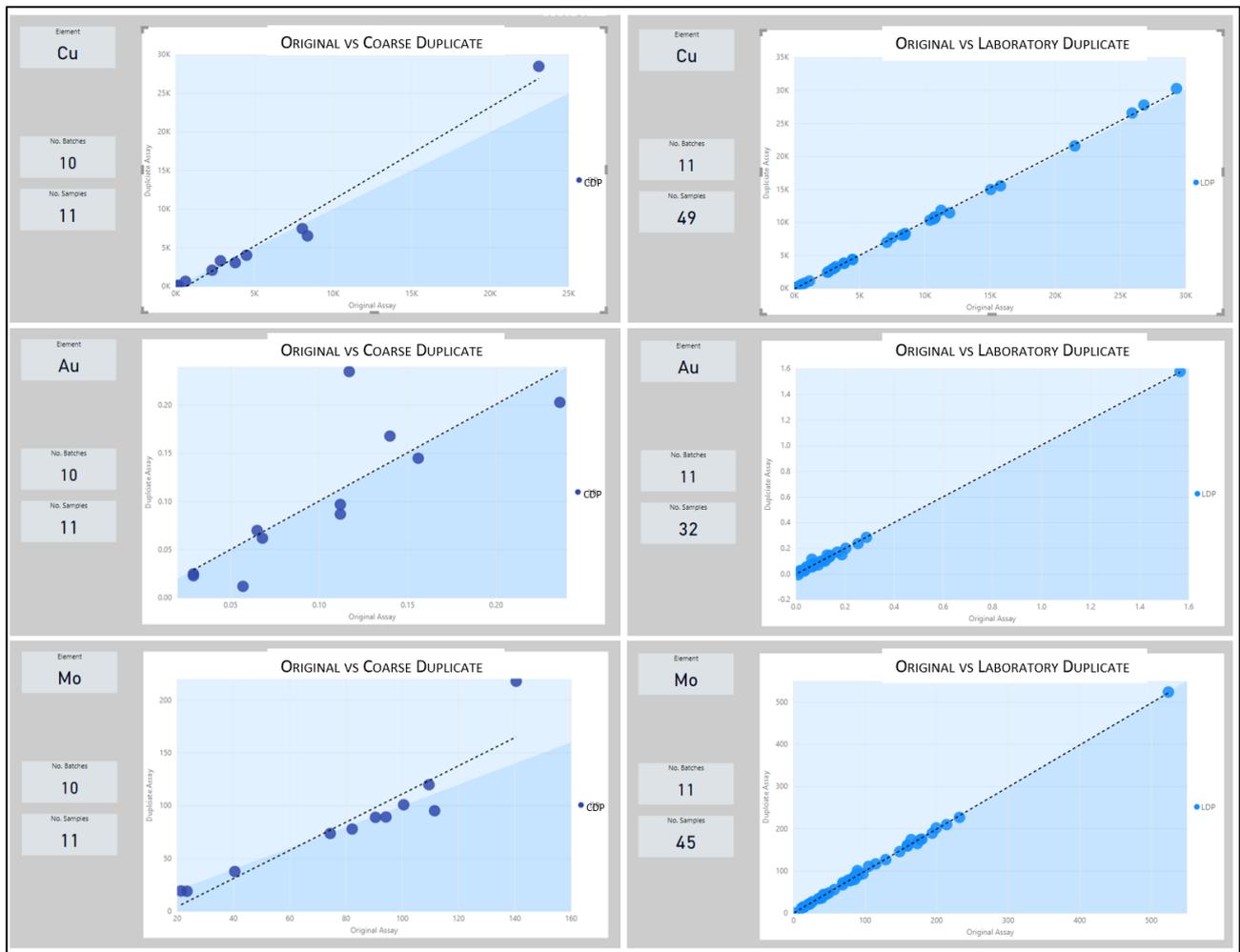


Figure 12-13 Control charts showing Cu, Au, and Mo results from the CRM sample analysed for the FQM diamond drill campaign



12.7 Sequential leach copper values

Sequential copper analysis results are useful for understanding and defining the style of copper mineralisation (i.e. refractory copper, oxide copper, secondary or primary sulphide copper). This form of analysis provides relative proportions of sulphuric acid soluble copper, cyanide acid soluble copper and residual copper per sample.

A total of 10% of samples (Figure 12-14) yielded sequential copper assay results.

Sequential leach copper analysis was conducted on pulp samples at ALS Mendoza or ALS Lima at several points in time, after analysis of the primary samples (Table 12-2):

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- In 2012, Lumina dispatched returned pulps from select portions of their drill holes for sequential analysis.
- In 2014, FQM sent a small batch of pulps from the 2012 Lumina drilling for sequential analysis, to further investigate the effect of time on oxidation of copper species within the pulp samples during storage. Pulp material stored in relatively humid storage conditions shows soluble copper values increasing and cyanide soluble copper values decreasing over time.
- The Company dispatched additional samples in 2016 and 2017 from Lumina pulp material in order to establish the viability of applying an oxidation correction factor to results from the 2012 and 2014 data. Results provided some resolution as to the degree of oxidation that pulp samples were exposed to over time.
- Samples dispatched from drill core recovered in 2019 were vacuum sealed prior to immediate transport to the laboratory in order to limit any oxidation of copper minerals. The results from these samples best represent the relative in-situ proportion of oxide, secondary and primary copper minerals.

Owing to the limited number of samples, similarly limited spatial coverage, and the variable impact of oxidation on most samples, sequential copper values were not estimated into the block model. Results were however, used to establish dominant copper mineralogy per geological domain. All sequential values were normalised to total copper values obtained from ICP (or ore grade AAS) analysis prior to any copper species/domain assessments.

Figure 12-14 Spatial distribution of samples (shown in red) with sequential leach copper assay results

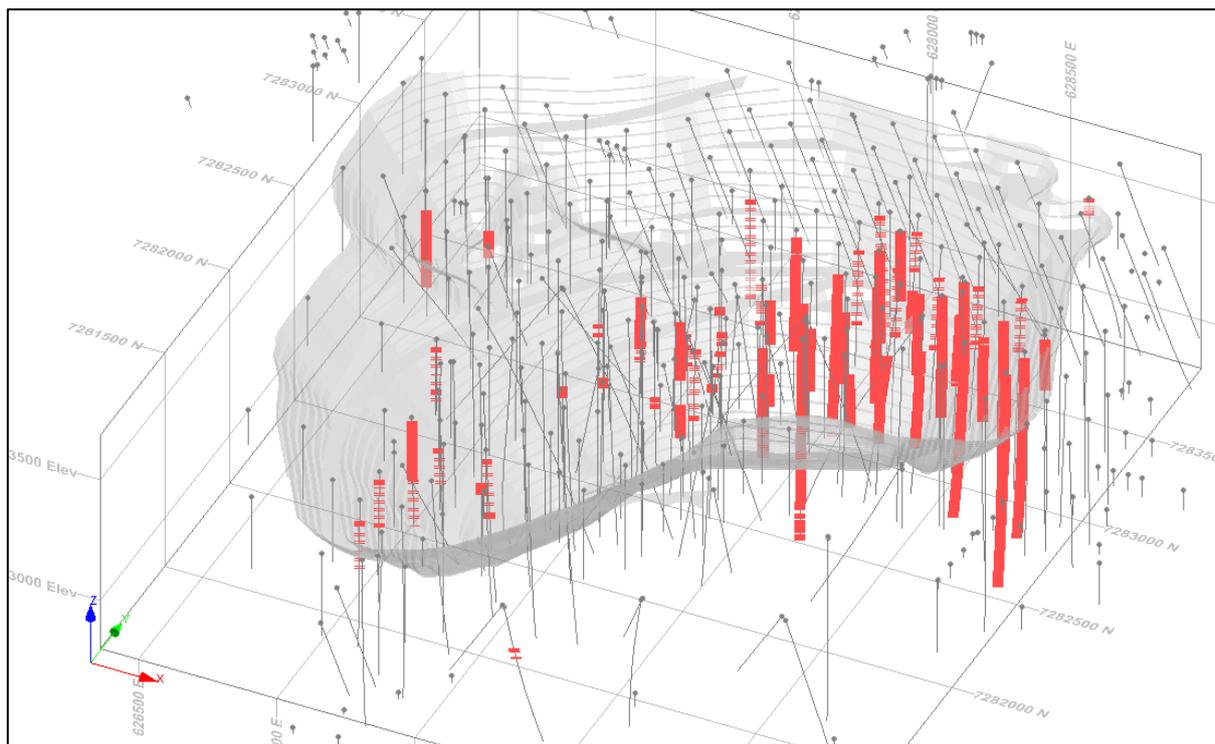


Table 12-2 Phases of dispatch for sequential leach copper analysis

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

A rate of oxidation study was initiated by the Company in 2019 using material from three of the metallurgical samples. Coarse-crushed homogenised material was left exposed at the deposit site and laboratory from which duplicate samples were sent weekly for sequential leach analysis. Results over a twelve month period demonstrate oxidation of copper minerals under both laboratory and site conditions is limited and therefore unlikely to impact on the material mined and processed.

12.8 Factors materially affecting the Mineral Resource estimate

It is the QP's opinion that sample preparation, analytical procedures, and secure data management have enabled consistent and repeatable sample analysis for most samples. Analysis of QAQC results indicate that adequate controls were in place and that assay results are reliable. Sample values are believed to be representative of the prevailing mineralisation and thus suitable for use in the Mineral Resource estimate. Historic data with limited records mostly provides additional information to the deposit peripheries and is not considered to pose a risk to the quality of the estimate.

ITEM 13 DATA VERIFICATION

David Gray (QP) has visited the Taca Taca site and the associated core storage facility in Salta on several occasions since project acquisition, with the most recent visit being in March 2019. During these visits, the QP has gained familiarity and confidence in the available data, geology, and prevailing mineralisation.

The following verifications were completed:

- Collar coordinates were validated against a high-resolution topographic surface together with field spot checks from hand-held GPS coordinates compared to database coordinates. No discrepancies were noted.
- Database sample and geological records were visually compared against the corresponding remaining core samples in storage. No discrepancies were noted.
- Bias checks were completed between sample types and analytical methods with marginal to no bias noted.
- Database validations were performed to:
 - ensure assay results in the database reflect original assay certificates (since 2008)
 - investigate outlier values of assay data fields
 - address errors in overlapping or duplicate sample and logging records
 - check orientations and relative magnitudes of downhole survey data
 - confirm relevant metadata was recorded consistently and accurately
- Analytical methods and QAQC results were assessed and verified as suitable to assure assay accuracy and precision, with sufficient controls on contamination.
- Residual pulp samples from Lumina drilling dispatched by the Company for sequential leach copper analysis have also served as check assays on total copper values. No significant issues were identified, apart from some pulp sample oxidation.
- 3D geological models were based on integrated datasets and interpretations were validated against relogs of stored drill core and core photography.

Review of drilling and logging procedures, sample preparation, analytical methods, and database security and management supports the assessment that data is good quality and representative of in-situ mineralisation. The QP is confident that the information available is of a suitable standard for use in estimating the Taca Taca Mineral Resource.

ITEM 14 MINERAL PROCESSING AND METALLURGICAL TESTING

Much of the metallurgical testwork performed by the previous owners was carried out by Plenge Laboratories in Lima from 2010 to 2012, under the supervision of Lumina personnel. This testwork was summarised by Pincock Allen & Holt in 2012 and included in the 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.

The salient points from these testwork campaigns are discussed below, with the results used to support the process designs.

14.1 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 of 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.

14.2 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 mineralisation⁷ was not considered as plant feed, but was metallurgically investigated for potential gold recovery to a scoping study level of detail.

Testwork was reported in numerous reports by Plenge and the findings summarised by Pincock Allen & Holt for the PEA.

14.2.1 Testwork summary

Comminution testwork indicated that the ore is soft (primary ore) to moderately soft (supergene ore) as defined in the JK Tech data base. Bond work indices are 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; secondary crushing of mill feed was considered unlikely to be required because of the high Axb numbers from the JK testwork.

⁷ Referred to as the “perched” gold horizon and also as Domain 102.

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Flotation testwork defined optimum conditions for flotation (grind, reagent suite, pH, slurry density, etc.), and provided recovery estimates and concentrate grades achievable.

The optimum primary grind was defined as 80% passing 150 µm, but a regrind of rougher concentrates to 80% passing 35 µm would be necessary to achieve high copper concentrate grades.

The majority of the work was conducted in Lima tap water. Several locked cycle tests were conducted using brine solutions from site, and these indicated similar recoveries to tap water in rougher flotation, but with increased mass pulls to concentrate. However cleaner flotation using brine resulted in significantly lower recoveries of all metals to concentrate and lower concentrate grades. Rougher flotation in brine, followed by cleaner flotation in tap water produced recoveries similar to those achieved using tap water alone, but at slightly reduced concentrate grades.

Preliminary separation testwork on a bulk Cu-Mo concentrate indicated that it was possible to achieve a molybdenum concentrate assaying 49% Mo and approximately 1.0% Cu.

Average recoveries over the life of mine were estimated to be 82.9% for Cu from supergene ores and 87% from primary ores. Fixed conservative recoveries over the life of mine of 55% for molybdenum and 60% for gold were assumed for both ore types.

Mineralogy

The copper mineralogy of two composite samples used for the original testwork are listed in Table 14-1.

Table 14-1 Mineral species % distribution MLA

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

Sequential copper analyses were conducted by Plenge on forty (15 supergene and 25 primary ore samples) provided for variability testing.

Figure 14-1 provides a graphical representation of the proportion of acid soluble copper, cyanide soluble copper and insoluble copper in the forty 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.

There appears to be no 'pure' primary ore in the Taca Taca deposit. All primary ore samples contained a minimum of 20% secondary copper minerals, and all contained acid soluble copper minerals (with 15 out of the 25 samples containing more than 10% As Cu). This may have a positive effect on concentrate grades achievable during processing.

The mineralogical evidence (where no oxide minerals were identified) is apparently incongruous with sequential copper analysis and similarly, the good flotation performance seen in the testwork results does not indicate the presence of oxide minerals.

Figure 14-1 Sequential copper analyses on supergene and primary ore samples



Comminution

SMC (SAG mill comminution) testing was completed on two samples from the deposit, one representing supergene, and the other primary ore. Ten samples (four supergene and six primary) were also sent for SAGDesign 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 these results is presented in Table 14-2.

Table 14-2 Comminution testwork results

Parameter	Units	Supergene Ores		Primary Ores	
		Ave Value	Tests	Ave Value	Tests
SG		2.7	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 A *b		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 is classified as moderately soft, whilst the supergene ore is 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.

Flotation

Initial flotation testwork was conducted on a composite sample of each ore type. The sample head grades were similar to the expected feed grades for the Project, and the results in Table 14-3 were obtained for the optimal conditions. The recoveries of gold and molybdenum were not optimised in these tests.

Table 14-3 Locked cycle test results

Ore Type	Head Grades			Bulk Cons Assays			Recovery %		
	Au ppm	% Cu	Mo%	Au ppm	% Cu	Mo%	Au	Cu	Mo
Supergene	0.12	0.57	0.018	4.20	33.9	0.55	51.3	86.9	45.8
Primary	0.13	0.40	0.018	4.86	34.5	0.95	33.7	90.1	54.1

The primary grind was 80% passing 150 µm; the results obtained as shown in Table 14-3 were achieved at a regrind size of 30 µm for supergene ores and 35 µm for the primary ore composite.

Copper recovery and grade appear to be sensitive to regrind size. Concentrate grade increases with finer regrind size, but recovery drops off – at 45 µm, the concentrate grade was only 22.6% Cu, whilst at 18 µm, copper recovery decreased by about 5% to 85% for both ore types. The optimum regrind size appears to be 30 to 35 µm.

Testwork on a bulk concentrate from the same composites indicated that it was possible to achieve a molybdenum concentrate assaying 49% Mo and approximately 1.0% Cu, at an overall Mo recovery of 44% from supergene ores and 53% from primary ores. The testwork was performed in open circuit; locked cycle operation would be expected to increase recoveries.

Optimisation of flotation reagents produced the recoveries and concentrate grades shown in Table 14-4.

Table 14-4 Flotation optimisation testwork

Ore Type	Head Grades			Bulk Cons Assays			Recovery %		
	Au ppm	% Cu	Mo%	Au ppm	% Cu	Mo%	Au	Cu	Mo
Supergene	0.08	0.75	0.033	1.90	31.4	1.15	60.8	89.9	72.2
Primary	0.16	0.45	0.028	5.4	30.6	1.91	62.4	92.8	84

Compared with the previous results, concentrate copper grades decreased by 3% to 4%, but recoveries were improved. The sample head grades were higher than for the previous tests, which may have contributed to the overall metallurgical recovery.

All of the above testwork was conducted in Lima tap water. Tests on the same composite samples using brine solutions collected from site gave the grades and recoveries in Table 14-5.

Table 14-5 Concentrate grades and recoveries in batch cleaning tests using brine solutions

Ore Type	Water	Bulk Cons 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

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Note that these were batch cleaner tests; locked cycle tests would be expected to give higher recoveries and lower concentrate grades.

Copper recoveries dropped significantly when using brine in the cleaners; concentrate grades were not seriously impacted apart from one test. In both samples, gold and molybdenum recoveries also decreased significantly when using brine for cleaner flotation.

On the basis of the above results, the process flowsheet will incorporate brine water for rougher flotation, followed by a dewatering stage and cleaner flotation in good quality water.

Testwork using brine for rougher flotation and tap water for cleaner flotation was repeated for confirmation purposes. Results are shown in Table 14-6.

Table 14-6 Concentrate grades and recoveries in batch cleaning tests

Ore Type	Water	Bulk Cons 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

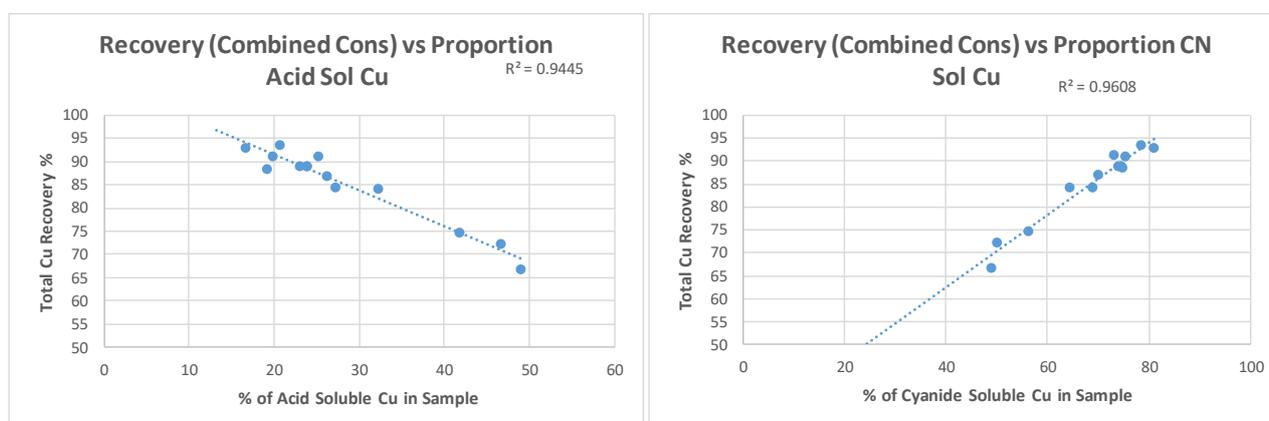
Variability testing

Forty samples (15 supergene and 25 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 is shown graphically in Figure 13-3.

The supergene samples had high copper grades, ranging from 0.51% Cu to 2.11% Cu, and averaging 1.23% Cu. Recoveries in batch cleaner flotation in the tap water varied between 66.8 and 94.5, with an arithmetic average of 85.8%, and 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 sample, and an inverse relationship between recovery and acid soluble copper in the sample (Figure 13-3).

Figure 13-2 Copper recovery from supergene ores vs cyanide soluble and acid soluble content



For the primary ore variability samples, the head grade varied between 0.31% Cu and 0.89% Cu, with an average of 0.49% Cu which is close to the planned production 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.

Multi element scans of concentrates produced from locked cycle testwork on supergene and primary composite samples indicated that the concentrates are clean, and do not contain high levels of penalty elements such as arsenic, bismuth, cadmium, antimony or mercury.

14.2.2 Copper-molybdenum separation

Initial copper-molybdenum (Cu-Mo) separation testwork was reported by Plenge in 2010

The separation tests were run in open circuit with six stages of cleaning for molybdenum. Results indicated that molybdenum concentrates assaying at 49% Mo can 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.

Further Cu-Mo separation testwork was carried out in 2012 on four composites representing the first ten years of operation.

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 in Lima tap water. Test results are listed in Table 13-7.

Table 13-7 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

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 conducted in 2019-20.

14.3 Testwork undertaken by FQM

An additional metallurgical testwork programme was completed in 2019, focussing on the starter pit plant feed types as defined by the mine plan. This testwork was undertaken in water collected from site, i.e. with brine from the salar for grinding and rougher flotation, followed by fresh and brackish water from site boreholes for cleaner flotation.

Four holes from the key copper domains, having representative grades and mineralisation within the starter pit, were drilled for metallurgical samples. These drill core samples were then composited (by depth) to provide ten distinct samples for variability testwork.

These samples were sent to the ALS laboratories in Kamloops, British Columbia, Canada, along with brine and brackish water samples sourced from the Project area at Salar de Arizaro (brine), Valle de Arizaro (brackish) and Valle de las Burras (brackish).

The locations of the sampling boreholes are shown in Figure 13-3 whilst details of the sample intervals are listed in Table 13-8.

Figure 13-3 Location of metallurgical sample boreholes, 2019 testwork

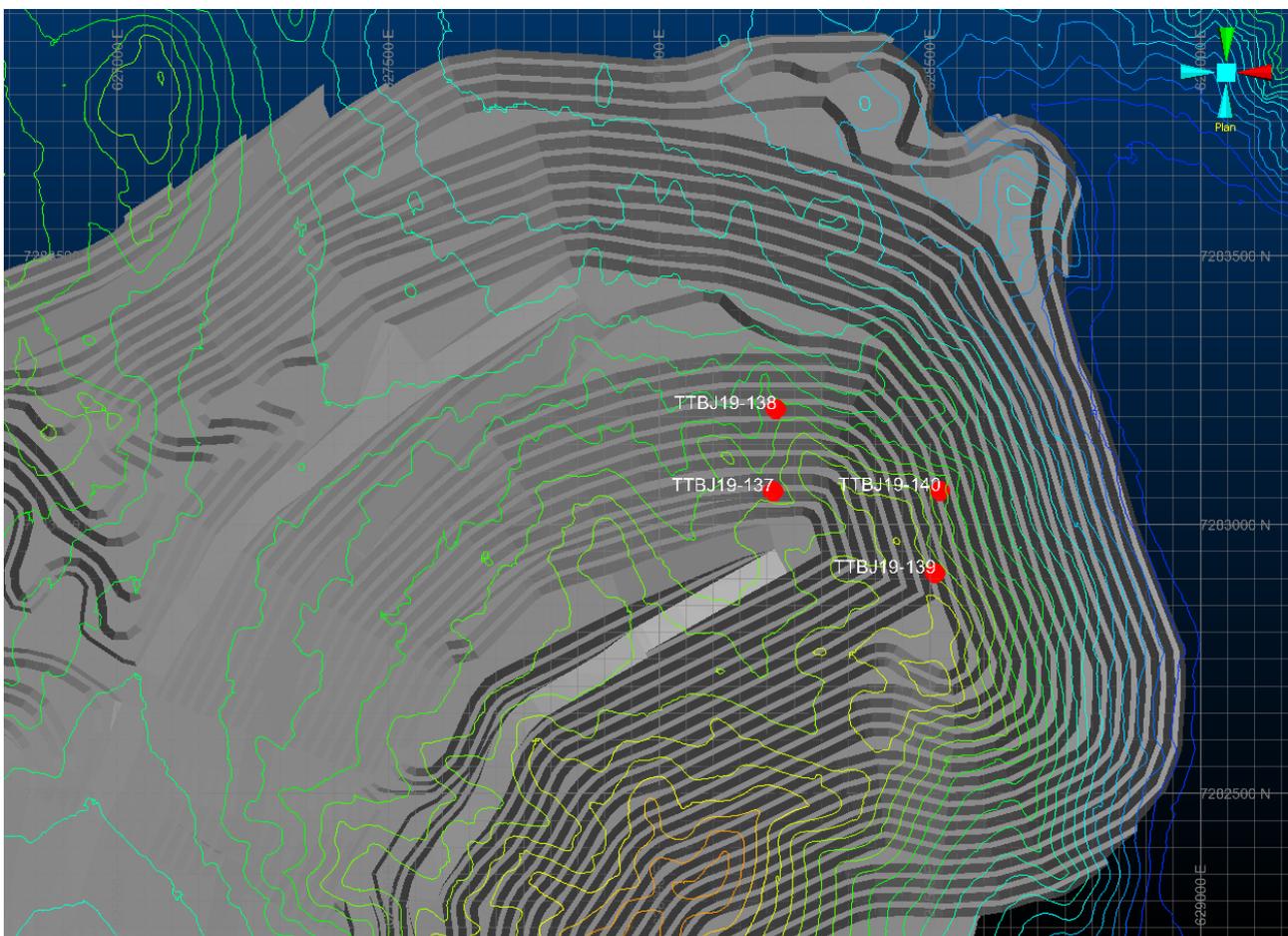


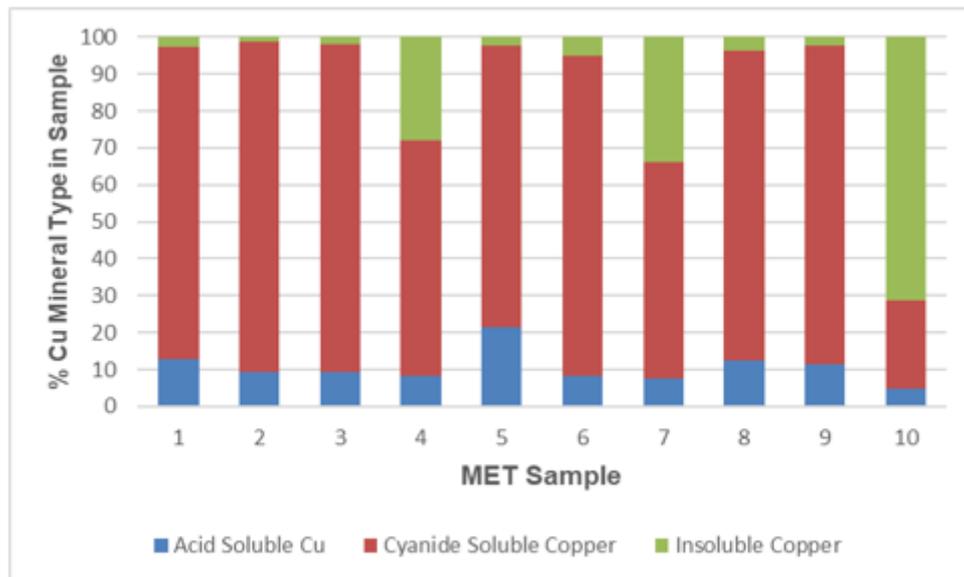
Table 13-8 2019 Metallurgical testwork, borehole sample intervals and assay grades

Sample	Intercept, m		Hole	Cu Domain	Pyrite	Lithology	Cu Assay, % Cu					Element Analyses			
	Start	End					Assay	Total	CuOx	CuCN	CuRes	Fe, %	S, %	Mo, %	Au, ppm
MET 001	134	164	137	CC > Sol + PL	Low	Granite	Ave	0.74	0.092	0.610	0.020	1.69	2.99	0.011	0.152
MET 002	175	205		CC > Sol	Low - Med	Granite	Ave	1.71	0.150	1.470	0.020	2.13	3.04	0.02	0.255
MET 003	278	308		CC > Sol	Med	Granite	Ave	1.68	0.150	1.420	0.030	1.98	2.59	0.021	0.216
MET 004	356	386		CC + FR	Med - High	Biotite - Granite	Ave	0.65	0.053	0.410	0.180	2.28	2.79	0.015	0.073
MET 005	240	270	138	CC > Sol	High	Granite	Ave	0.43	0.091	0.325	0.010	2.94	4.75	0.005	0.020
MET 006	261	290	139	CC > Sol	Low - Med	Granite	Ave	1.48	0.120	1.285	0.075	1.52	2.26	0.023	0.288
MET 007	317	348		CC + FR	Low - Med	Biotite - Granite	Ave	0.52	0.037	0.285	0.165	1.78	2.17	0.013	0.142
MET 008	202	232	140	CC > Sol + PL	Low - Med	Granite	Ave	0.90	0.112	0.745	0.034	2.20	3.62	0.013	0.102
MET 009	290	320		CC > Sol	Low - Med	Granite	Ave	0.96	0.109	0.835	0.021	2.34	2.71	0.015	0.090
MET 010	350	380		CC + FR	Low	Biotite - Granite	Ave	0.35	0.017	0.084	0.250	2.58	3.11	0.017	0.143
Arithmetic Averages of FQM & ALS Head Grades								0.94	0.09	0.75	0.08	2.14	3.00	0.015	0.148

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

The proportions of acid soluble Cu, cyanide soluble Cu and insoluble Cu in the ten composites are plotted in Figure 13-4.

Figure 13-4 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.

14.3.1 Comminution testwork

The comminution testwork was performed by SMC (JK Tech), and analysis of the results was carried out by Orway Mineral Consultants (OMC). Sample data information and test results are listed in Table 13-10. Historic samples, tested in 2010 & 2012, are included in the table for comparison purposes.

Table 13-10 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 13-10 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 13-10 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.

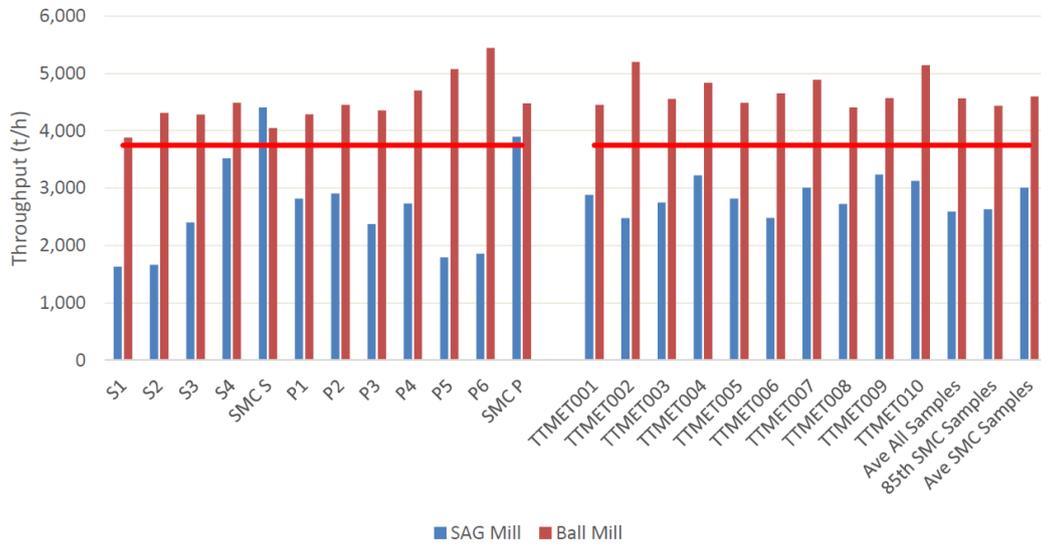
Using an average BWi of 15.6 (from the most recent ten samples), a required grind of 150 µm, and a throughput of 60 Mtpa (7,500 tph), indicates that ball milling would require a power consumption of between 69 MW and 74 MW. This suggests a requirement for four ball mills, two per milling train, and with 20 MW to 22 MW drives. Relaxing the grind size to 80% passing 180 µm would lead to a ball mill power consumption of 61 MW to 66 MW.

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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).

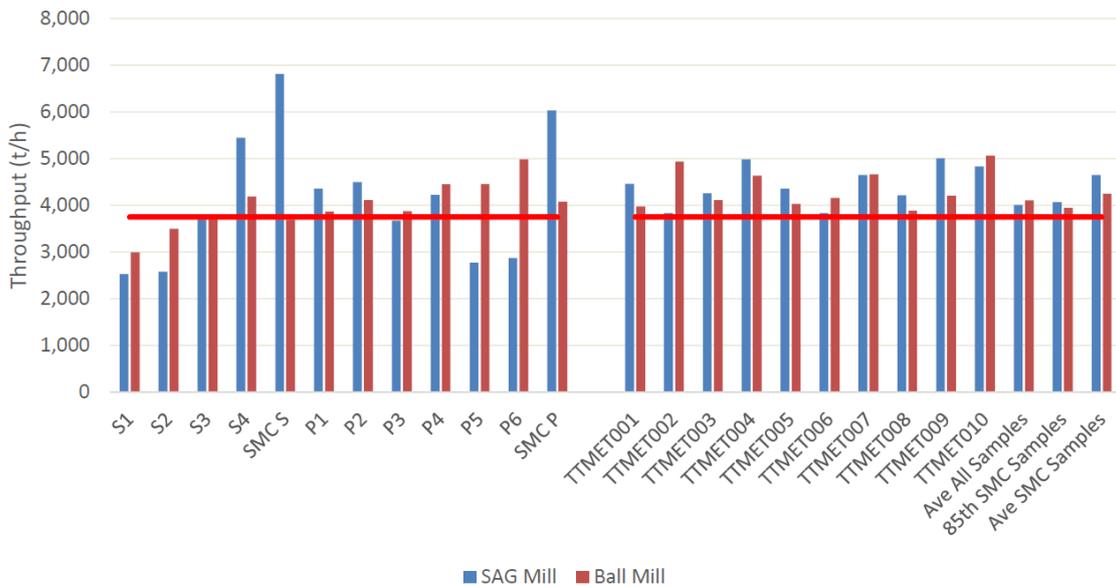
The throughput predictions from the individual samples, and the target 3,750 tph are graphed in Figure 13-5.

Figure 13-5 Mill circuit throughput predictions for SABC circuit



As can be seen from Figure 13-5, the proposed circuit is severely SAG mill limited, except in relation to the two historic SMC samples with high A_{xb} 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 13-6.

Figure 13-6 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 exactly 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.

14.3.2 Primary and concentrate regrind size requirements

The 2013 scoping study reported the optimum primary grind to be 80% passing 150 µm. This grind size was initially used in the most recent testwork, but coarser grinds were subsequently tested during the rougher flotation tests in tap water 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. The optimum conditions in terms of primary grind and concentrates regrind size were used in the locked cycle testwork, and are shown in Table 13-11, tabulated against borehole and sample depth.

Table 13-11 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	Biotite Granite	261	290	222	30
	MET 007		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. A primary grind of 80% passing 180 µm, and a regrind size of 80% passing 20 µm will be used for plant design.

14.3.3 Flotation testwork conducted in 2019-20

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.

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. This was 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).

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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. Dewatering testwork was conducted by Outotec.

Rougher flotation testwork

The first phases of flotation work, 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), but the grind was later coarsened to 80% passing 172 µm to 216 µm without any meaningful change in recoveries or mass pull. 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) was used, which together with the pH was aimed at minimising pyrite recovery to the rougher flotation concentrate.

The average recoveries from the total of 42 rougher tests in tap water was 93.7% for Cu, 77.4% for Mo and 78.8% for Au. The average mass pull was 9.2%.

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%. 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.

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 16.5%. Compared with rougher flotation in tap water, copper recoveries were reduced by about 4% using brine in the roughers and both molybdenum and gold recoveries were reduced by about 2%.

Cleaner flotation testwork

A single batch cleaner test was performed in tap water, in both the rougher and the cleaner flotation stages. The nine samples gave an average recovery of 85.5% Cu at a concentrate grade of 33.7% Cu.

The first batch cleaner tests using brine in the rougher float were run with lime addition to achieve a pH of greater than 10 in the roughers. This led to high lime addition, 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

Locked cycle testwork

The locked cycle testwork protocols attempted to mimic the proposed plant water circuits where 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.

Locked cycle tests were run for five cycles, with the average results for cycles 4 and 5 being reported. The locked cycle testing results are presented in Table 13-12

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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 samples have been re-ordered in the table so 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 last two samples is biotite – granite.

Table 13-12 Locked cycle test results

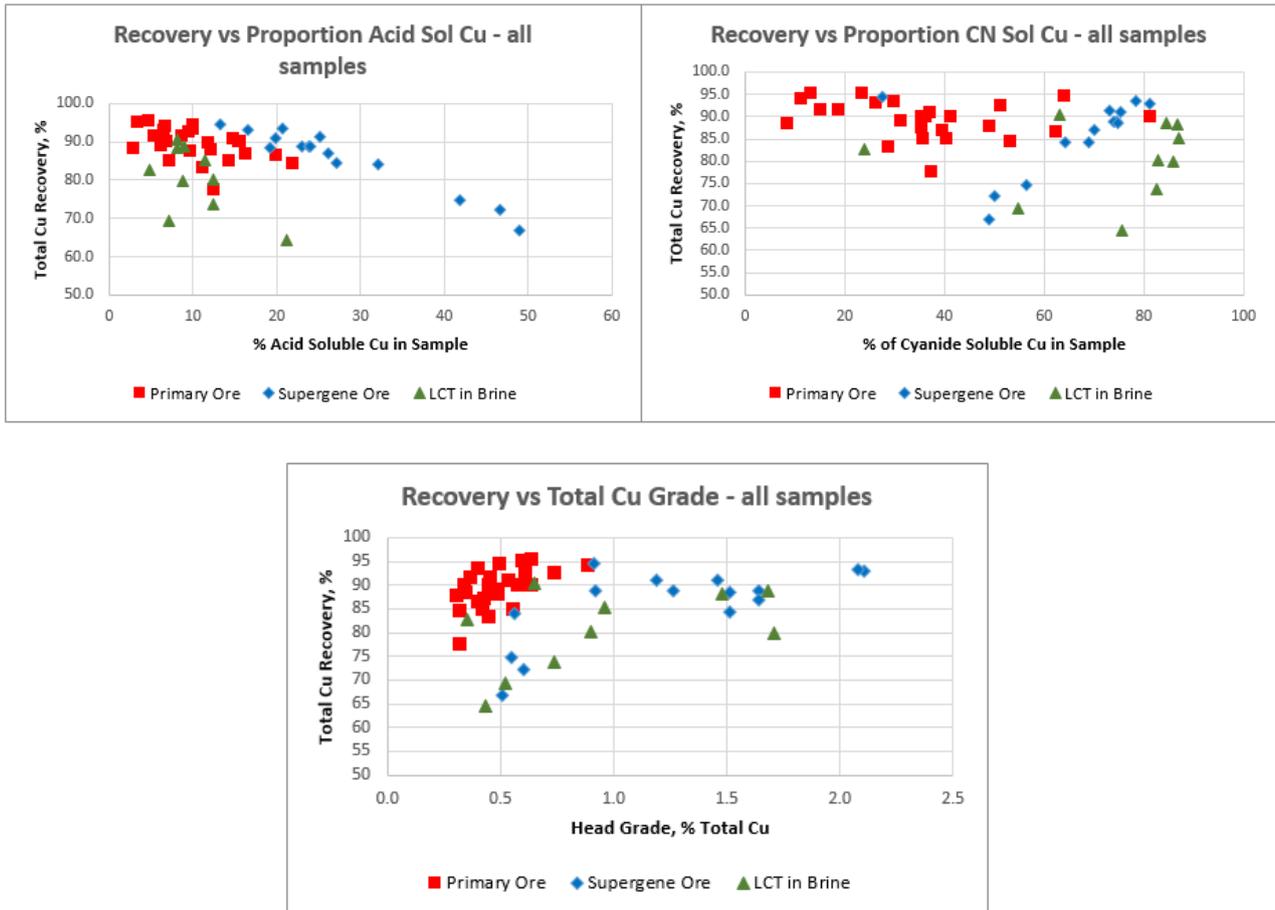
Sample	Grind, P ₈₀ µm		Weight %	Assay - percent or g/tonne					Distribution - percent				
	Primary	Regrind		Cu	Mo	Fe	S	Au	Cu	Mo	Fe	S	Au
MET 001	192	23	1.4	35.1	0.35	21.9	32.6	4.37	73.8	46.8	20.9	17.6	51.1
MET 008	202	18	1.9	37.3	0.29	21.7	35.5	2.82	80.3	52.4	21.6	19.6	59.0
MET 002	216	24	3.0	44.1	0.45	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

Results from a bulk flotation test are reported in Table 13-12 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.

Molybdenum recoveries averaged 50.6%, at a grade of 0.37% Mo in the bulk Cu-Mo concentrate, whilst gold recoveries averaged 58% at a grade of 4.85 g/t in the concentrate.

The results from the locked cycle tests were plotted on the same graphs presented earlier in Figure 13-2, showing copper recovery as a function of acid soluble copper, cyanide soluble copper, and total copper in the feed.

Figure 13-7 Copper recovery from variability samples and recent locked cycle tests vs copper speciation



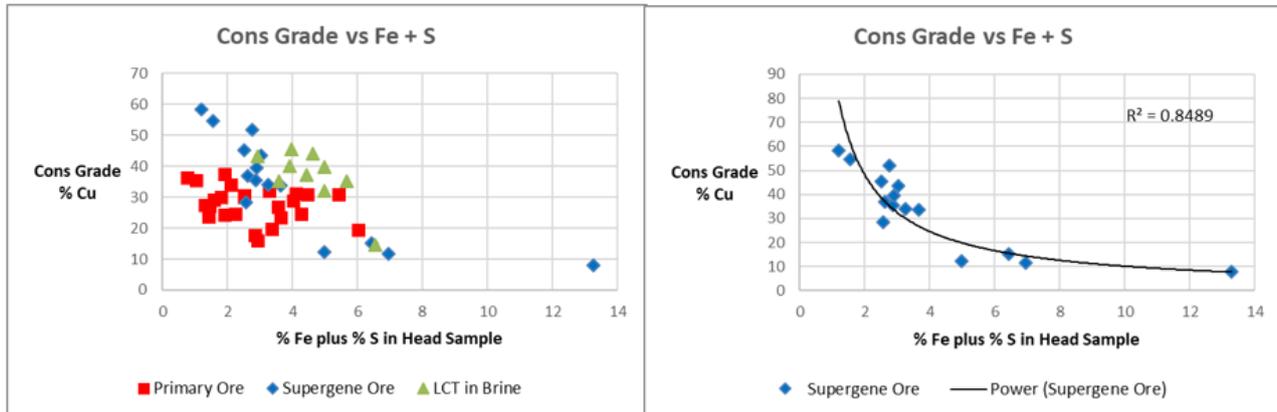
The graphs in Figure 13-7 illustrate the lower recoveries from the recent ten metallurgical samples compared with the variability samples tested in 2011. The variability sample results were obtained from batch tri-cleaner tests performed in tap water, whilst the results from the ten samples tested in 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.

The effect of pyrite content in the sample on concentrate grade is illustrated in Figure 13-8 (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 13-8 Copper concentrate grade variability samples and recent locked cycle tests vs pyrite content



14.3.4 Concentrate analyses

Multi element scans of concentrates produced from the locked cycle testwork on supergene and primary composite samples tested in 2010 to 2012 and from the 2020 testwork at ALS on the ten metallurgical samples produced the results listed in Table 13-13. (Only the elements of interest have been included in this table).

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 as a result 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.

Table 13-13 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
Ag	ppm	11.2	17.8	22.0	20.1	14.6	15.3	3.1	16.7	17.6	14.1	5.8	25.7
As	ppm	198	17	51	41	57	172	12	13	18	74	51	19
Bi	ppm	5	5	3.1	2.8	1.7	5.8	1.5	1.1	4.6	1.7	1.2	5.2
Cd	ppm	6	10	1.2	1.7	0.8	3.8	0.4	0.5	1.1	0.6	2	3.6
Cr	ppm	311	640	240	280	100	170	140	190	50	70	80	30
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
Se	ppm			250	280	200	210	100	250	340	200	190	330
Sr	ppm	20	32	45	65	23	25	18	97	13	60	14	4
Th	ppm			4	3	2	4	3	5	2	2	2	<2
U	ppm			1	1	<1	1	<1	1	1	<1	1	<1
Zn	ppm	300	400	90	30	30	130	20	20	90	40	50	180

14.3.5 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 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 primary grind was 80% passing 180 µm, and the regrind size for the copper concentrate 80% passing 24 µm.

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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 are listed in Table 13-14.

Table 13-14 Thickener size recommendations

Process Stream	Concentrate	Cln Scav Tail	Rougher Cons	Rougher Tails
Feed Density, % solids	15-25	25-35	25-35	25-35
Diluted Feedwell % solids	15	14	13	20
Slurry pH	11	11	7	7
Floc Dosage, g/t	10	20	80	30
Underflow density, % solids	68.8	48.2	47.6	67.2
Overflow clarity, mg/L	<150	<150	415	<150
Solids loading, t/m ² /h	0.2	0.8	0.5	1.4
Feed rate, tph	200	1350	1500	6750
Required Thickener Area, m ²	1000	1688	3000	4821
Single Thickener Diam, m	36	47	62	79

Outotec noted that a solids loading for a concentrate thickener of over 0.25 t/m²/h was not recommended. The rougher concentrate thickener area requirements would therefore double to 6,000 m², giving a single thickener diameter of 87 m.

In practice, multiple thickeners would be installed instead of single large units. Three x 50 m diameter units would be provided for both the rougher tailings and the rougher concentrate duties.

Filtration tests on the final concentrate 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 m³/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/m²/h; this is 2.35 m²/(t/h), and is considerably lower than the average of 1.14 m²/(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.

14.3.6 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 on the ten composite samples tested in 2020 to identify if any gold was recoverably by means into a gravity concentrate that could then be treated to produce doré.

The testwork procedure was the standard Knelson and panning test using a 100g cone. The tests were performed on each individual metallurgical composite and thus 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 13-15.

Table 13-15 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 2.2g/t Au. The best recovery of 20% Au from sample MET001 still resulted in a concentrate grade of only 2.9g/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 26g/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 the centrifugal concentrators currently included in the circuit design should be re-evaluated.

14.3.7 Geochemical characteristics of tailings

The tailings produced by the proposed processing plant would be typical of copper flotation tailings. The milling process would result in a particle size distribution with a top size of 150 µm. The ore would be mostly competent and hence there would not be a fine clay-like component. The tailings should be relatively free draining and following settlement, should consolidate to a dry density of around 1.4 t/m³.

Geochemical analyses of testwork tailings was undertaken in 2011 on two samples identified as M1 7758-62 and M1 7763-67. These samples contained low levels of residual S and the ABA (acid base accounting) testwork results placed the samples in the area of uncertainty with respect to the neutralization potential (NP) and the acid generating potential (AP).

This work was repeated in 2020 on each metallurgical composite, giving the results in Table 13-16.

Table 13-16 ABA testwork results, 2020

Sample	Paste pH	Sulphur %		NP	AP	NNP	NP/AP
		Total	Sulphide	t CaCO ₃ /kt	t CaCO ₃ /kt	t CaCO 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

All NNP (net neutralizing potential) results were less than minus 20, and all NP/AP results were less than 1, indicating that all the samples are potentially acid generating.

SPLP tests were not performed by ALS on these samples as part of the specified testwork campaign. They have since been requested and results are expected later in 2020.

SPLP (Synthetic Precipitation Leach) tests were performed in the initial testwork campaigns and repeated in 2020.

The two samples tested in 2011 contained high levels of Cu, Mo, Ni and Se and were considered as anomalous elements as their concentrations in the samples were higher than the background concentrations.

The leaching tests on these two samples and on the ten composites tested in 2020 indicated no significant concentration of the elements from the leaching (SPLP) extract were found. The final pH of the leachate varied between 8.36 and 6.21.

The conclusions drawn from this geochemical testwork is that:

- acid generation or significant concentration of metals from the tailings samples is not evident
- however, there is uncertainty that warrants further testwork and the need for monitoring during the operations phase

14.4 Metallurgical recovery estimates

The variability testwork conducted in 2012 included mineralogical information on sulphide sulphur content, iron and sequential copper data (i.e. acid soluble copper, cyanide soluble copper and residual copper – see Figure 13-1), together with recovery and concentrate grade data.

During the course of the Mineral Resource modelling (Item 14), multi variate analysis (neural network analysis) was completed on the variability data highlighting distinct groupings related to recovery and concentrate grades.

These groupings 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.

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 13-17, 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.

Table 13-17 Metallurgical domains, 2020 update

Ore Type	Total Copper Grade % Cu	Iron Grade Ranges % Fe	Recovery Cu %	Cons Grade % Cu
Oxidised Supergene, classified as a Mixed ore from domain 203.				
CC>>Cpy with > 30% acid soluble copper				
low Cu, low pyrite	<0.4	<1.5	65	15
low Cu, medium pyrite	<0.4	1.5 to 2.0	65	15
low Cu, high pyrite	<0.4	>2.4	65	15
medium Cu, low pyrite	>0.4	<1.5	74	20
medium Cu, medium pyrite	>0.4	1.5 to 2.0	74	20
medium Cu, high pyrite	>0.4	>2.4	70	20
Supergene Mixed, classified as Secondary ore from domains 304, 305, & 310				
CC>Cpy with 10 to 30% acid soluble copper				
low Cu, low pyrite	<0.4	<1.5	83	25
low Cu, medium pyrite	<0.4	1.5 to 2.0	83	25
low Cu, high pyrite	<0.4	>2.4	80	22
medium Cu, low pyrite	0.4 to 1.0	<1.5	88	34
medium Cu, medium pyrite	0.4 to 1.0	1.5 to 2.0	86	34
medium Cu, high pyrite	0.4 to 1.0	>2.4	84	30
high Cu, low pyrite	>1.0	<1.5	88	35
high Cu, medium pyrite	>1.0	1.5 to 2.0	88	35
high Cu, high pyrite	>1.0	>2.4	86	32
Supergene Transition, classified as Supergene mixed from domains 307, 308 & 309				
Cpy>CC with < 10% acid sol Cu (but actually most samples from this zone have CC>Cpy)				
low Cu, low pyrite	<0.4	<1.5	85	25
low Cu, medium pyrite	<0.4	1.5 to 2.0	85	25
low Cu, high pyrite	<0.4	>2.4	82	22
medium Cu, low pyrite	>0.4	<1.5	88	30
medium Cu, medium pyrite	>0.4	1.5 to 2.0	88	30
medium Cu, high pyrite	>0.4	>2.4	86	26
Hypogene, classed as Primary ore from domain 306				
Cpy>>CC with < 10% acid sol Cu				
low Cu, low pyrite	<0.4	<1.5	86	25
low Cu, medium pyrite	<0.4	1.5 to 2.0	86	25
low Cu, high pyrite	<0.4	>2.4	84	22
medium Cu, low pyrite	>0.4	<1.5	90	30
medium Cu, medium pyrite	>0.4	1.5 to 2.0	90	30
medium Cu, high pyrite	>0.4	>2.4	86	26

14.4.1 Comments on the estimates

Taking into account the results from the original Lumina testwork, as well as the recent locked cycle tests in brine, several trends are evident from the groupings in Table 13-17:

- low recovery and concentrate grades correlate with high soluble copper and high pyrite content

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- high pyrite content adversely 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 for some of the ore domains listed in Table 13-17, there is insufficient metallurgical testwork to fully support the recoveries and concentrate grades shown. The recoveries and grades for these domains have been interpolated from data which has a higher level of confidence.

From the testwork results and mine production schedules the following average life of mine recoveries are anticipated:

- copper recovery of 85.0% to a concentrate grade of 25.3 %Cu
- molybdenum recoveries of 40% to a moly concentrate grade of 47% to 50% Mo
- gold recoveries to the copper concentrate of 60%, with a grade of approximately 4.5 g/t

14.5 Representivity of sampling and testing

Significant testwork has been undertaken on numerous samples from Taca Taca, and this work has defined optimum conditions for flotation (grind, reagent suite, pH, slurry density, etc.). The majority of the early work was conducted in Lima tap water, and little work was undertaken using water from site.

Ten metallurgical samples were produced from four bore holes drilled in the area of the starter pit in 2019. These samples are representative of the first five years of operations. Locked cycle testwork was performed on these samples using brine from the Salar de Arizaro (adjacent to the project) for the rougher flotation and a combination of brine and brackish water for cleaner flotation.

Data from the latest testwork has been used in the process design designs and the recovery and concentrate grade information forms the basis of the production schedules and projected metallurgical recovery.

14.6 Comment on the adequacy of the original testwork

In the Company's opinion, the testwork has been performed to a high standard and in the majority of cases, the sample source (drill hole number and depth) are defined. The Company's review of the testwork adequacy is as follows:

1. Significant comminution testwork was undertaken on a range of samples from varying depths and orebody locations. Sufficient data was generated for process design work, including SAG milling design reports produced by specialist consultants.
2. Optimum flotation conditions (grind size, reagent additions, pH, slurry densities etc.) were adequately defined during the various testwork campaigns in both tap water and in brine sourced from the site.
3. Flotation variability work was performed on 40 different samples from a number of different drill holes and from various depths. Fifteen samples were classed as supergene with head grades ranging from 2.11% Cu to 0.51% Cu, and 25 were classed as primary with head grades ranging from 0.31% Cu to 0.89% Cu (average 0.49% Cu). This testwork was performed as batch cleaner tests in tap water.
4. Copper/molybdenum separation testwork was performed on bulk samples derived from locked cycle testwork on 100 kg composite samples of each ore type. This testwork was also carried out using tap water.
5. The early testwork program included four locked cycle tests that were performed on samples said to represent the first ten years of the Project life; i.e. two composite samples of each ore type, where one

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represented the likely ore feed within the first five years, and the other represented the likely ore feed from years six to ten. This testwork was carried out in Lima tap water.

6. The recent locked cycle testwork was undertaken on ten composite samples representing the first five years of operations. This work was performed in brine and brackish water and replicated as near as possible the flotation conditions and water quality to be expected on site.
7. The results from locked cycle testwork on the ten samples tested in 2019-2020 form the basis of the process design and the recovery estimates.
8. Sedimentation testwork to define thickener sizing was performed on samples from within the starter pit area, but was undertaken initially using tap water. This work was repeated in the 2019-2020 testwork campaign using brine.
9. Some additional testwork is recommended, but this is for optimisation purposes and is not critical to development of the project.

ITEM 15 MINERAL RESOURCE ESTIMATE

15.1 Introduction

Mineral Resource estimates were generated for copper (Cu), gold (Au), molybdenum (Mo), silver (Ag), iron (Fe) and sulphur (S). Grades were interpolated into a three-dimensional (3D) geological block model using ordinary kriging, and parent block estimates were post-processed using localised uniform conditioning (LUC).

Estimates were completed in December 2019 by David Gray (QP), FQM Group Mine and Resource Geologist, using commercially available software (Datamine Studio RM (v 1.6.75.0), Snowden Supervisor (v 8), and Viscovery SOMine 7). The Project limits and coordinates were based in the WGS84 UTM Zone 19s grid system.

Estimates used drill hole sample assay results exported from a secure database and a geological model interpretation that relates to the spatial distribution of copper, gold, and molybdenum mineralisation. Interpolation parameters were based upon the geology, styles of mineralisation, drill hole spacing, and geostatistical analysis of the data.

Mineral Resources were classified according to geological continuity, drill hole grid spacing, grade continuity, confidence in the grade estimate, and the reasonable prospects for eventual economic extraction. Reporting was guided by the Australian JORC Code (JORC, 2012) and the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy, and Petroleum (the CIM Guidelines, 2014). The Mineral Resource classification was guided by a life-of-mine pit shell and was reported using a 0.13% copper equivalent cut-off grade.

15.2 Available data

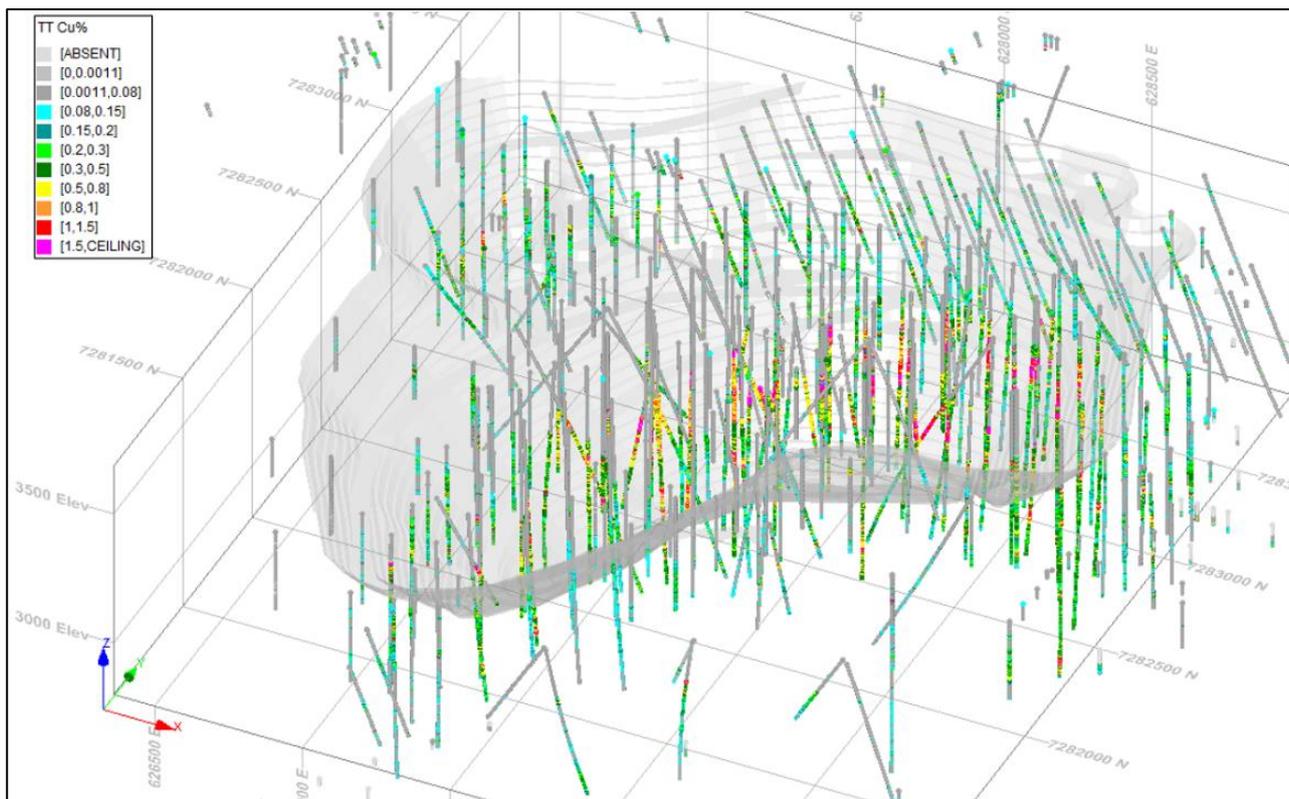
The upper limits of the 3D block model were defined by a high-resolution (0.5 m) topographic surface. Drillhole samples used in the estimate were drilled by BHP, Corriente, Rio Tinto, Lumina, and FQM (Table 11-3). Most holes, and those with greatest influence on the estimate, were drilled by Lumina between 2010 and 2012. In total, 158,643 m drilled in 395 holes provided the 75,803 samples used in the estimate (Figure 15-1).

Drillhole data includes collar coordinates, downhole surveys, assays, logs of geology, weathering, minerals, alteration and structures. For core drilled since 2010, core recovery, RQD, point load test, and density measurements were recorded, and magnetic susceptibility and SWIR spectrometer readings were taken. Files data were subject to routine validation checks with no data errors or inconsistencies identified.

All samples were assayed for Cu, Au, and Mo with more than 90% of samples having ICP multi-element analysis. Sequential leach copper assays were available for 10% of samples but these values were not estimated owing to their limited spatial distribution and variable degrees of sample oxidation which is not representative of in-situ mineralisation. Correction factors were applied to subsets of the sequential copper data in order to provide an indication of dominant copper mineral species within each geological 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.

Figure 15-1 Drillholes used in Mineral Resource estimate coloured by total %Cu (shown relative to the ultimate pit design)



15.3 Drillhole file

Collar, downhole survey, sample assay and logged data was validated and clipped to limit holes to the immediate vicinity of the deposit. Data files were combined to generate a 3D drillhole trace using Datamine's standard de-surveying process. De-surveyed drillhole data was used for geological modelling, visual, statistical and spatial analysis, and the estimation of mineralised domains. A separate drillhole file was used for estimating density values, owing to their comparatively narrow sample interval.

During drillhole file generation, the following adjustments were made to the data:

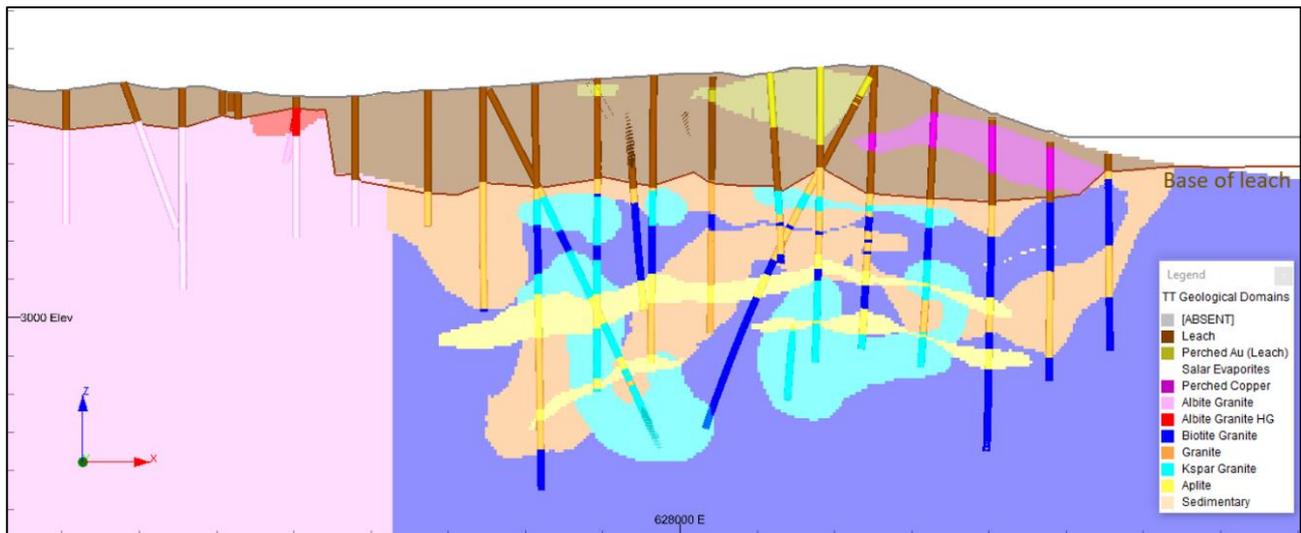
- assay data with negative values were reset to values at half the lower detection limit for that element
- different logging codes used by previous owners to represent the same lithology were grouped into a merged rock-type field, to facilitate consistency in 3D modelling
- sequential copper leach values were normalised to their associated total copper assay values to ensure the sum of respective proportions was aligned.

As 98.9% of samples had a 2 m sample length, drillhole sample assay data was not composited.

15.4 Geological model and domains

Eleven geologically distinct domains were identified from a combination of rock-type, weathering, alteration, and mineralisation characteristics (Figure 15-2). The domains were defined using Neural Network Analysis (NNA) of multi-element assays, the evaluation of sequential leach copper values, and correlations with drill core logging.

Figure 15-2 Section 7282775N, looking north, showing geological domains coded into drillhole samples and block model



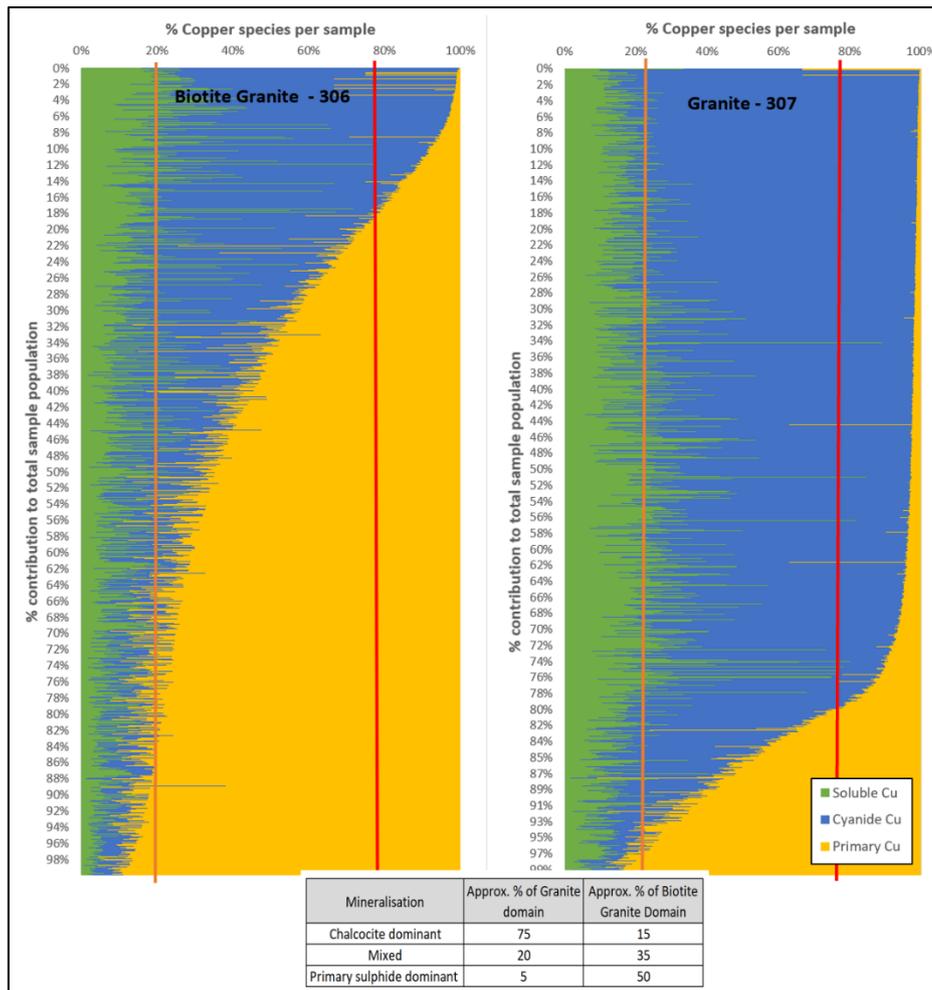
NNA identified major intrusive bodies and alteration assemblages from sample multi-element data which was supported by an understanding of major and trace element geochemistry. Where phyllic alteration was less intense, secondary biotite and potassium feldspar were preserved, making these areas visually, geochemically, and spatially distinct.

Analysis was conducted by the QP in collaboration with site geologists using Viscovery SOMine software. Results were validated against logged data and showed excellent correlation. Drillhole samples were coded with the resulting domains and were used to digitise sectional string envelopes that were linked with wireframe volumes and surfaces. Domains were given colloquial rock-type names for ease of operational use (Table 15-2).

The 150 m to 300 m thick leached cap and an underlying, narrow, discontinuous partially leached zone were modelled as sub-horizontal surfaces from visual relogging and assay data. Table 15-1 provides the weathering field codes used. Owing to their thin and erratic nature, the partially leached lenses were included in the fresh rock estimation domains. The leached-mineralised contact is displaced across the north-south trending TK2 fault. This fault was modelled in 3D using geophysical, mapping, logging, and assay datasets.

Analysis of sequential leach copper results provided an indication of dominant copper mineral species per domain. Interpretations were verified with visual inspections of drill core. Though alteration and mineralisation show an association, the relationship is not explicit and copper mineralisation within each domain is mixed to varying degrees. Figure 15-3 shows an example comparing sequential copper results from the 'granite' and 'biotite granite' domains. The pervasively phyllic altered granite domain (307) has dominant supergene chalcocite, whereas the biotite-granite domain (306) having less intense phyllic alteration and preserved coarse secondary biotite shows a preference to primary sulphide copper minerals.

Figure 15-3 Graphical representation of relative percent contribution of copper species per sample. Domain 306 and domain 307 are presented for comparison



Estimation domains were aligned with geological domains and were spatially defined by wireframe surfaces or volumes. The estimation domain field was coded to include weathering and rock-type code values (Table 15-3). Sample assay data was coded according to the wireframes per domain.

Though mafic dolerite dykes were defined from NNA, these were not modelled owing to limited sample numbers and insufficient data for defining dyke orientation and continuity. No estimation was conducted for the salar evaporite domain given its distance from the main deposit area and poor data support. Though an estimation was completed for the meta-sediment domain (310) to the northwest of the porphyry, this is not spatially relevant to the reported Mineral Resource.

Table 15-1 Weathering 'WEATH' field numerical codes and their associated descriptions

WEATH field number	WEATH field description
100	Leached
200	Partial leach
300	Fresh

Table 15-2 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

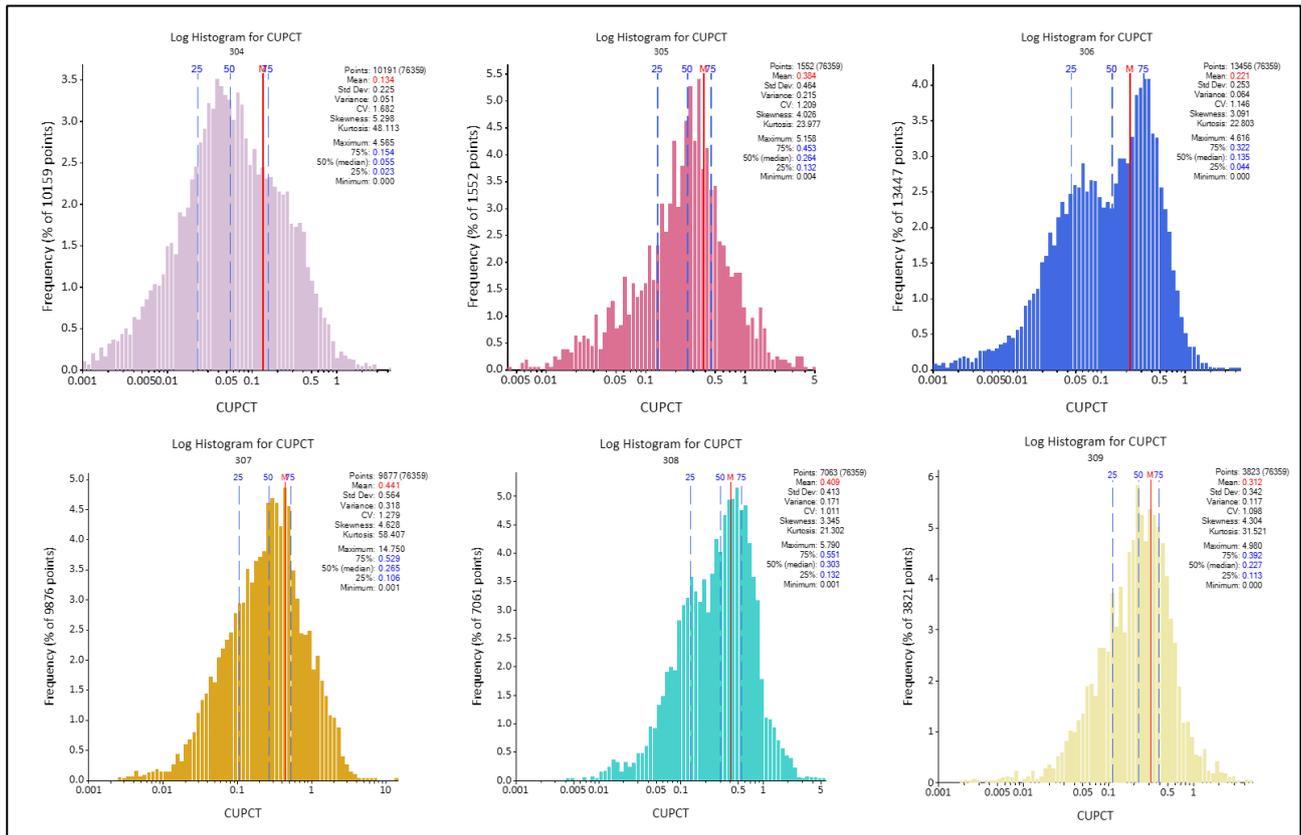
Table 15-3 Estimation domain 'DOMAIN' field codes and their associated descriptions. Domain field values created by adding WEATH and ROCK field values

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 southeast and south	Chalcocite>>soluble copper
Fresh Metasediments	310	Wireframe Surface	Overlaying the porphyry to the northwest	Soluble copper, chalcocite, Au-Cu veins

15.5 Data analysis

Univariate statistical analysis of sample data per domain was completed using Snowden Supervisor software. Data distributions were investigated for mixed populations and excessive variability by using histograms, log probability plots, and descriptive statistics. Statistical analysis (Figure 15-4) highlights that fresh mineralised domains had reasonable normal distributions with evidence for some population mixing.

Figure 15-4 Histogram distributions for the main fresh mineralised domains



Prior to estimation, histograms and coefficients of variation of sample grades were investigated for high-grade samples within the sample population. Though highest-grade values are real, they are often not representative of local grade distributions. Top-cuts were used to restrict the influence of these samples as per Table 15-4. Top-cuts applied to Cu and Mo values affected approximately 0.01% of samples, reduced the coefficient of variation and had a marginal effect on mean grade values. The top-cut applied to Au affected 0.04% of samples and reduced the coefficient of variation, lowering grade variability and improving grade estimates.

Table 15-4 Top-cut values applied to samples per metal, prior to estimation

Metal	Top-cut applied
Cu	6%
Au	2.5g/t
Mo	3000ppm

Univariate statistics suggest sample values per domain have reasonable normal distributions with limited domain mixing. The respective domains and top cuts have supported well-defined variogram models and are suitable for using ordinary kriging estimation.

15.6 Boundary analysis

Contact profiles were generated to evaluate grade changes across domain boundaries. These profiles graphically display the average grades at increasing distance from the contact boundary. Apart from the leach zone, all domains show gradual grade changes across boundaries indicating a limited need for samples from adjacent domains to be isolated during interpolation. Hard boundaries were used according to geological controls.

The following summarises the hard boundary conditions applied:

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- A hard boundary was used between the leached domain (101) and all other domains as per boundary analysis and in line with geological controls. Example contact profiles of the leached domain with the granite (307) and k-feldspar granite (308) are provided in Figure 15-5.
- Hard boundaries were used for domains 307 (granite) and 308 (K-feldspar granite) with all other domains. This was to limit mixing between domains with different copper mineralogy and grade profiles.
- A hard boundary was used for domain 309 with all other domains. Domain 309 represents aplite sills, an intrusive in sharp contact with surrounding igneous rocks. As such, mixing between this domain and others was limited to preserve domain identity.
- A hard boundary was used for domain 310 (meta-sediments) since it is associated with different geological and mineralisation controls to the other domains.

Figure 15-5 Contact profiles between leached domain 101 and domains 307 (left) and 308 (right)

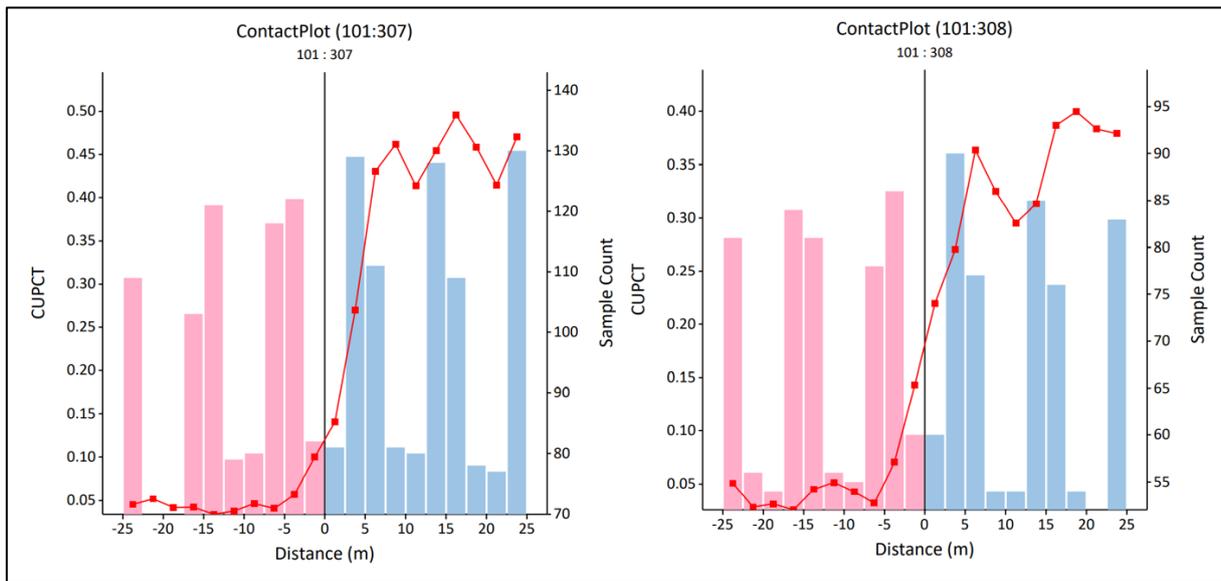
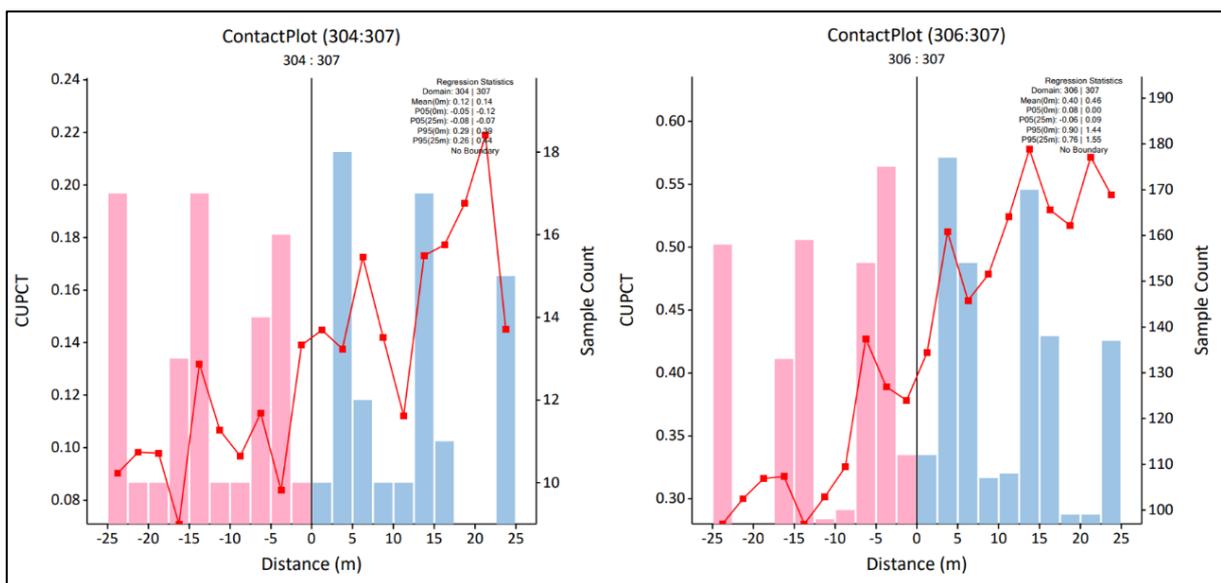


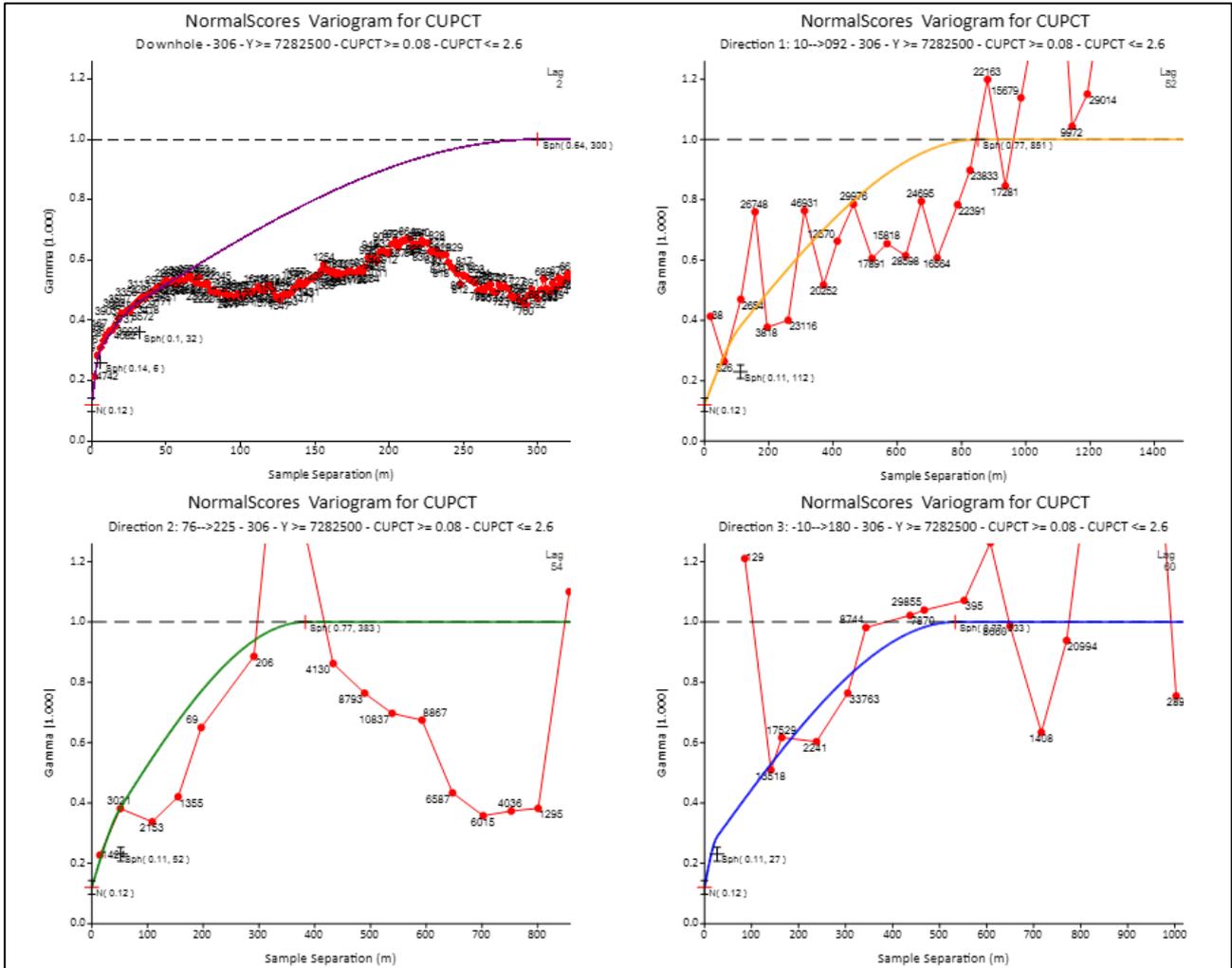
Figure 15-6 Example contact profiles showing soft boundary conditions between domains 304 and 307 (left) and domains 306 and 307 (right)



Soft boundaries were employed for domains 304 (albite granite), 305 (albite granite – high grade), and 306 (biotite granite). For domain 304 and 306, one sample above and three below the boundary were used

during estimation. For domain 305 one sample either side was used. Examples of contact profiles for domains 304 and 306 with domain 307 are provided in Figure 15-6.

Figure 15-7 Downhole, strike, dip, and plunge variograms for domain 306



15.7 Spatial analysis

3D continuity of domain sample grades was modelled using spatial analysis and variography. Variograms were generated from samples using Snowden Supervisor software. The following method was applied:

- Principal axes of anisotropy were determined using variogram fans based on normal scores variograms.
- Directional normal scores variograms were calculated for each of the principal axes of anisotropy.
- Downhole normal scores variograms were modelled for each domain to determine the normal scores nugget effect.
- Variogram models were determined for each of the principal axes of anisotropy using the nugget effect from the downhole variogram.
- The variogram parameters were standardised to a sill of one.
- The variogram models were back-transformed to the original distribution using a Gaussian anamorphosis.
- Variograms were standardised to the population variance per domain to facilitate post-processing of the grade panel estimates to SMU estimates.
- Variogram models were used to guide search parameters and complete ordinary kriging estimation.

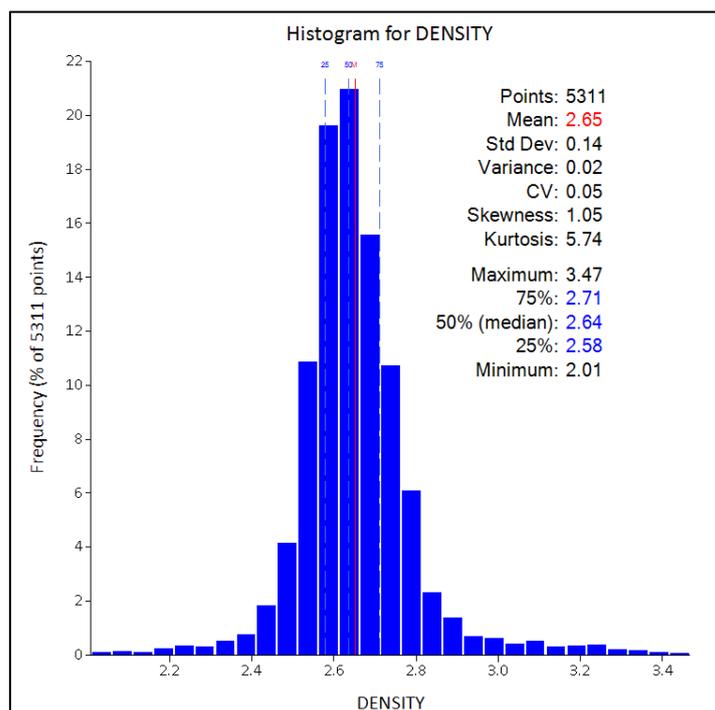
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Nugget values were low and clearly defined. Similarly, ranges of continuity were easily defined and had clear anisotropy and orientations. Variogram models for Cu, Au, and Mo have similar orientations. A summary of variogram parameters used in the estimation is detailed in Table 15-5 and an example variogram model is provided in Figure 15-7.

Table 15-5 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

Figure 15-8 Histogram showing distribution of density values across the deposit



15.8 Block model construction

A kriging neighbourhood analysis (KNA) was undertaken to determine the optimal block size, sample selection ellipse dimensions, and the minimum and maximum number of samples to be used during grade estimation. KNA was completed in Snowden Supervisor and used the modelled variograms and a series of estimates detailing the kriging efficiency and slope of regression values.

A parent block size of 60 mE by 60 mN by 15 mRL (bench height) was selected as having optimal kriging efficiency and regression slope values. Block sizes also consider drill grid spacing, smallest mining unit (SMU) dimensions, and the need to accurately reflect the volumes below the topographic surface. Selected search ellipse dimensions were aligned with variogram ranges (Table 15-6). Search parameters are outlined in Table 15-7.

An empty 3D block model was defined in Datamine as per the parent block size. The block model was coded using topography, weathering, and rock-type wireframe surfaces and volumes. For the aplite sills (domain 309), the block model was populated with dynamic anisotropy vectors calculated from the geometry of the associated aplite wireframe volume.

The LUC block size was set at sub-cell dimension of 7.5 mE by 7.5 mN by 7.5 mRL. This was to provide adequate volume filling of respective geology domains and support a 7.5 mE by 7.5 mN by 15 mRL SMU. The selected SMU dimension represents the grade and tonnage distribution in each 60 m by 60 m by 15 m parent cell at the scale of mining. Block model origin, extents, and sub-cell dimensions are presented in Table 15-8.

Table 15-6 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 15-7 Summarised search parameters per domain estimates (for all grades)

DOMAIN	Max. # samples per hole	Search Pass	Min. # of samples	Max. # of samples
101	6	First	8	24
102	6			
203	6			
304	4			
305	4			
306	4			
307	4	Second (1.5*initial search ellipse)	8	24
308	4			
309	4			
310	4			

Table 15-8 Block model settings used in the Mineral Resource estimate

Model Setting		Value
Origin	X	626,100mE
	Y	7,280,500mN
	Z	2,400mRL
Maximum	Northing	629,640mE
	Easting	7,284,880mN
	Elevation	3,810mRL
Parent cell size	X	60m
	Y	60m
	Z	15m
Minimum cell size	X	7.5m
	Y	7.5m
	Z	7.5m

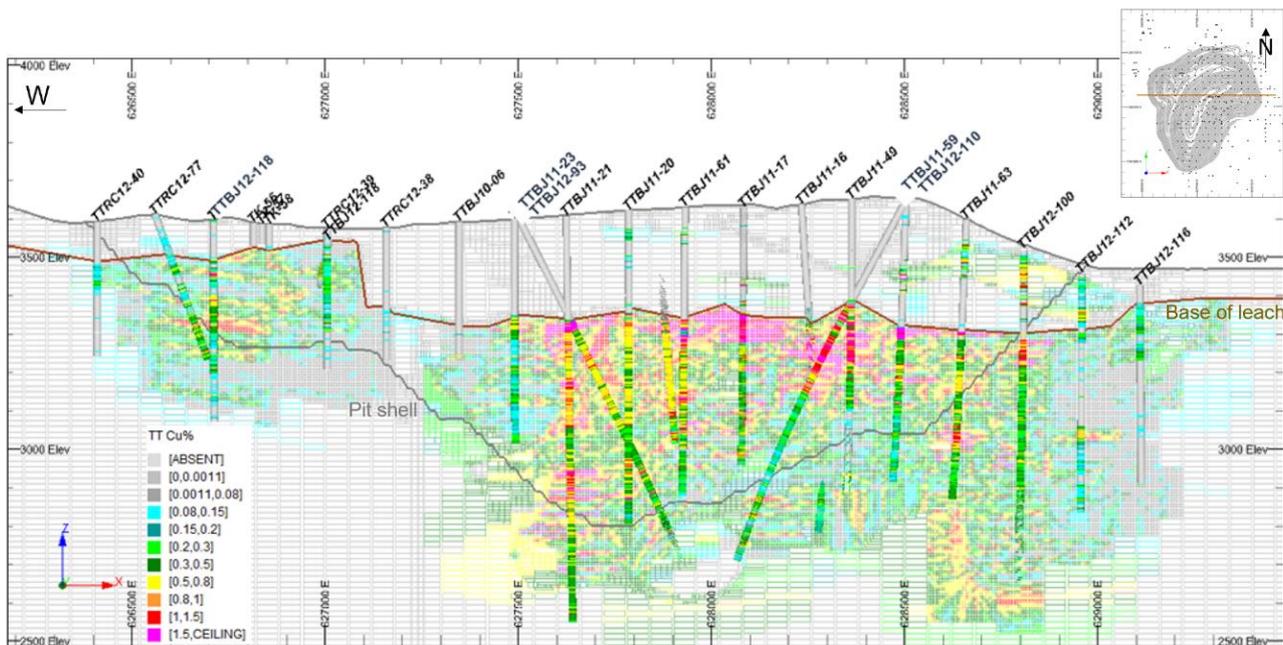
15.9 Density estimation

A total of 5,363 in-situ dry density measurements were collected from diamond core drilled between 2010 and 2012 (Figure 15-8). Values were obtained using the traditional Archimedes method of weighing samples in air and then in water. Samples show good coverage across the deposit (Figure 11-5) and had sufficient numbers of samples per domain to minimise variability in results introduced from moisture or pore spaces.

Density data was analysed statistically and outlier values removed. Top and bottom cuts excluded values less than 2.0 t/m³ and greater than 3.5 t/m³ as per typical granite density values. No density measurements were taken from the evaporites, thus a nominal value of 1.5 t/m³ was applied.

Weathering has the strongest influence on density and blocks were estimated per weathering domain (leached and fresh). Mean density values per weathering horizon are provided in Table 15-9. Blocks not estimated for density, due to sparse sample support, were assigned a mean density value of 2.65 t/m³.

Figure 15-9 Vertical section 7282775N, looking north. Compares drillhole sample grades and block model grades: %Cu (top); g/t Au (middle); ppm Mo (bottom)



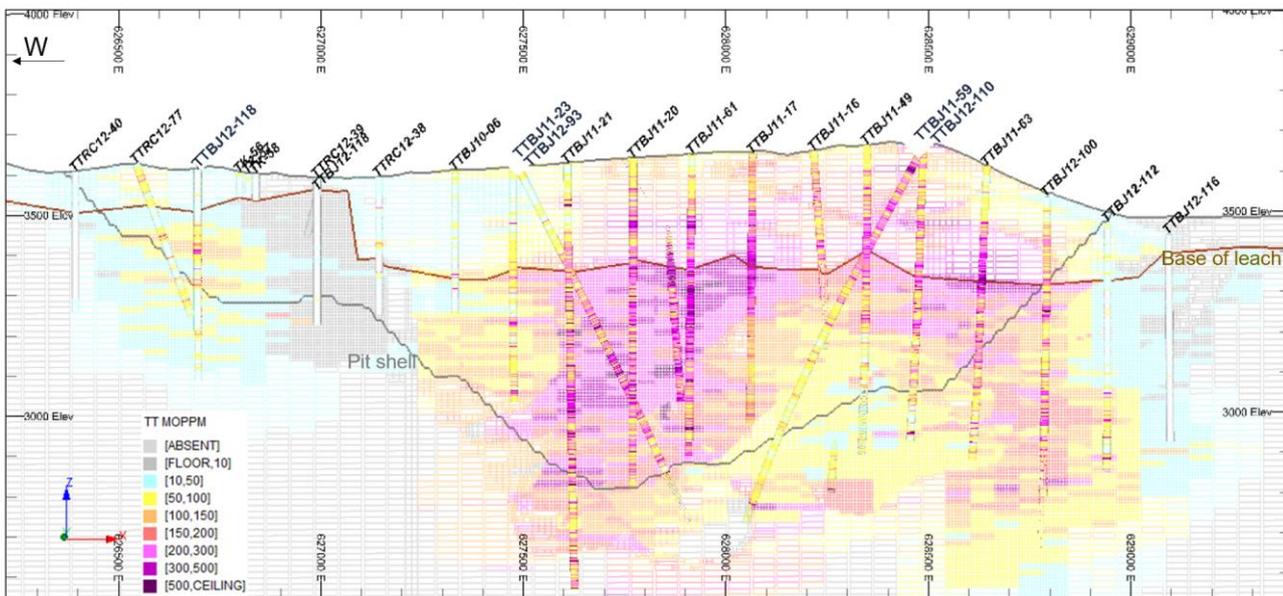
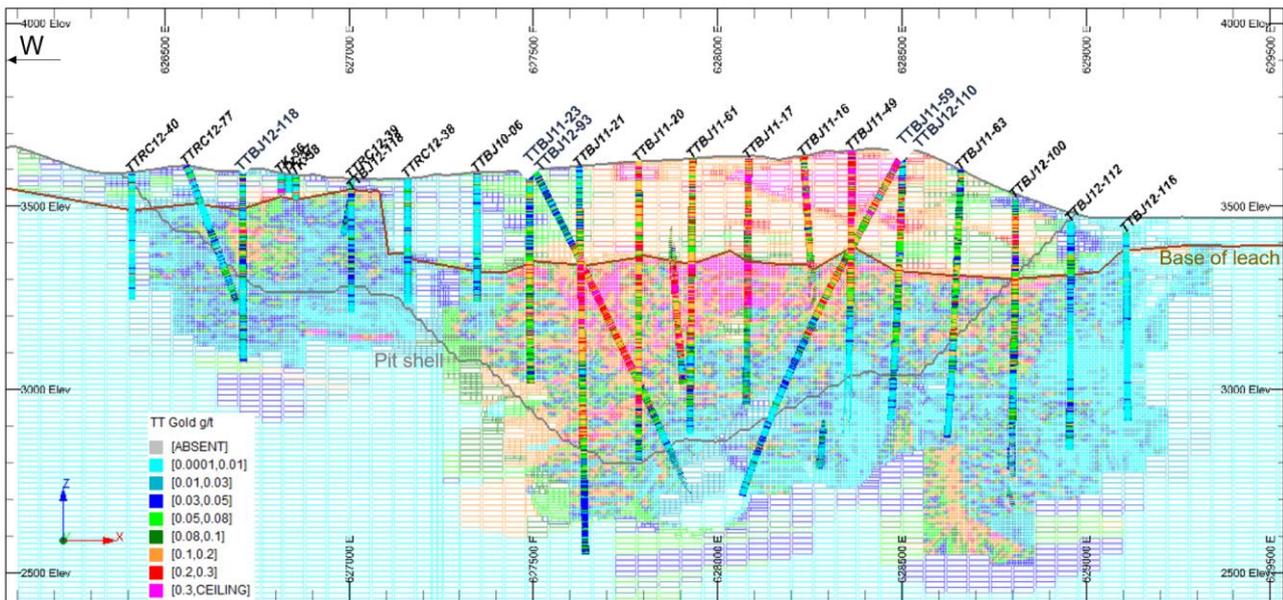


Table 15-9 Mean density value per weathering domain

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

15.10 Grade estimation

All estimates, including density, used ordinary kriging (OK) into the parent block dimension. OK was considered an appropriate estimation technique for mineralised domains owing to the near-normal distribution of grade values and limited domain grade population mixing. Kriging estimation parameters were based on variography, KNA, geological continuity, and drill grid spacing.

Estimates were populated into the coded block model per domain, using a sample assay dataset with top-cuts applied. Most blocks were estimated within the first search ellipse. Blocks around the periphery of the

deposit, in areas remote of regular grids of closely spaced samples, required a second search ellipse, 1.5 times the first. Parent block estimates used a discretization of 8 (X points) by 8 (Y points) by 4 (Z points) to better represent blocks dimensions. Blocks with absent grades were assigned waste grade (trace) values and were all located in peripheral non-mineralised zones.

To provide grade and tonnage estimates at the scale of mining, parent blocks estimates from the first search pass, and within fresh material, were post-processed using localised uniform conditioning (LUC). LUC estimates for copper were determined per parent block's sub-cells and the estimated panel grade. Uniform conditioning provides the proportions of parent block sub-cells above a range of cut-off grades. LUC determines a grade per sub-cell within the parent block, while maintaining metal content of the parent. Where applicable, LUC estimated grades replace original parent estimate grades.

The 7.5 m by 7.5 m by 7.5 m dimensioned sub-cells were re-blocked into the SMU block dimensions of 7.5 m by 7.5 m by 15 m. Although LUC improves representation of grade and tonnages expected during mining, the current drill spacing does not support spatially accurate grade estimates per SMU block. Per domain, however, post-processed LUC grades validate well with the parent block estimates with no metal lost or created.

15.11 Model validation

A series of validation steps were completed to ensure block grade estimates represent the prevailing geology and input sample data. These included:

- a visual comparison of the sample and block grades in 2D cross sections
- northing, easting, and vertical moving window grade trend (swath) plot slice validations
- a comparison of respective domain mean sample grades with mean estimated grades of the block model

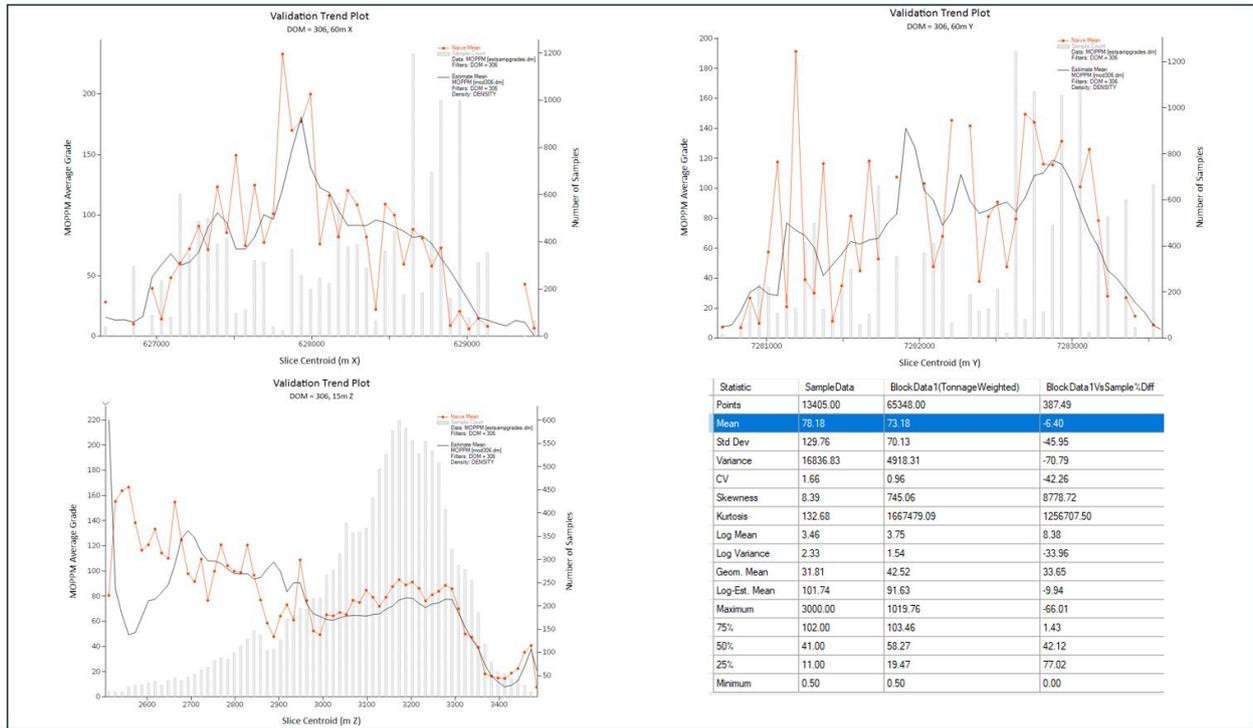
Visual validation suggests the grade tenor of the input data is represented in the block model estimates (Figure 15-9). Swath plot validations demonstrate estimates compare with input data, particularly where sufficient data informed block estimates. An example is provided in Figure 15-10, showing swath plots for domain 306 (granite). Input sample data was compared to OK and LUC estimated grades for Cu and Au, and to OK estimated grades for Mo.

Validation steps confirmed that block model estimates reflect the input data and can be considered a reliable representation of prevailing mineralisation and sample values.

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Figure 15-10 Swath plots for domain 306 at 60 m northing, 60 m easting, and 15 m elevation increments. Input data was compared to estimated OK and LUC grades for Cu (top), and Au (middle), and to OK estimated grades for Mo (bottom)





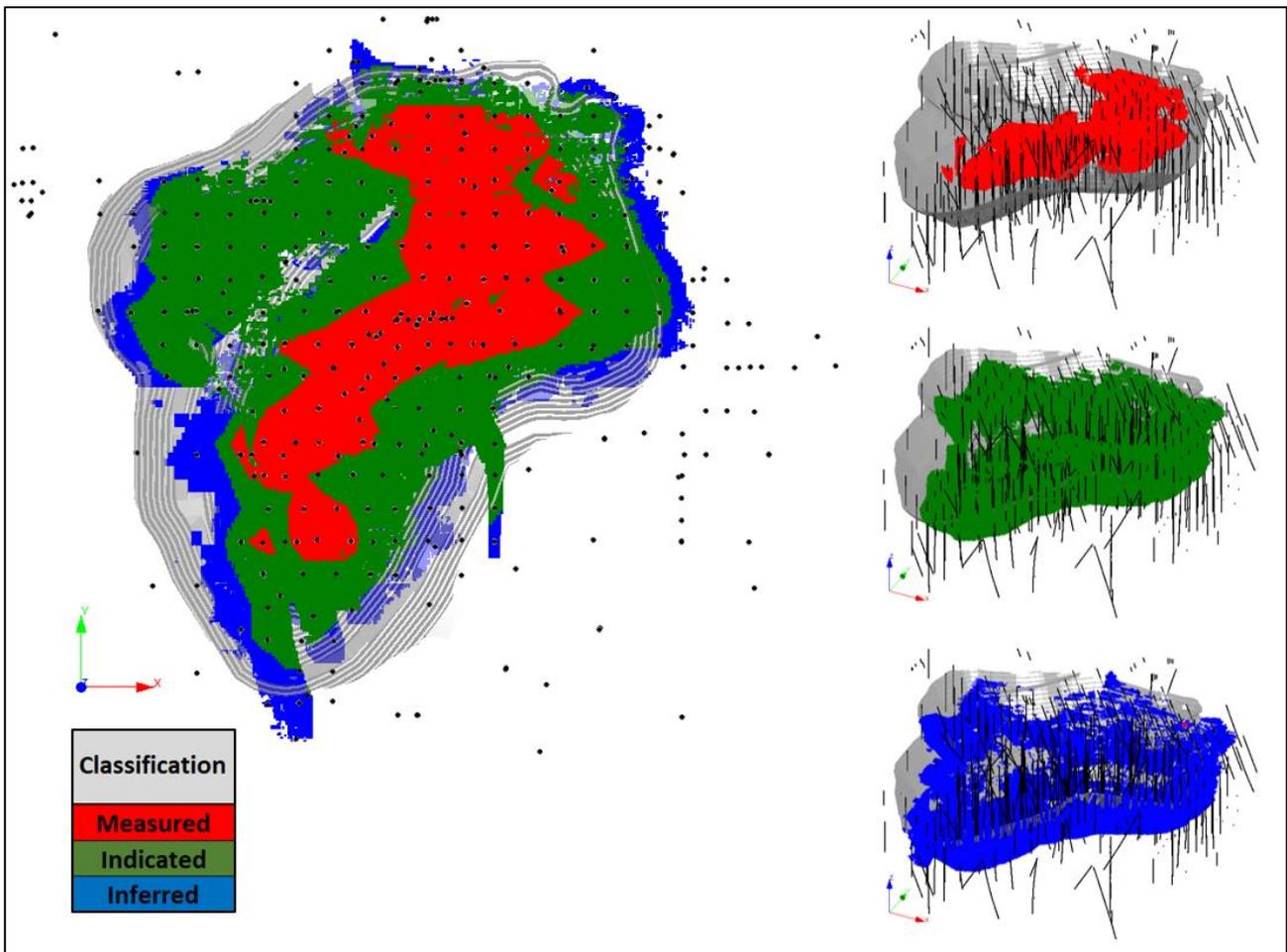
15.12 Mineral Resource classification

The Mineral Resource estimate was classified as Measured, Indicated, and Inferred in accordance with the Standards on Mineral Resources and Reserves of the Canadian Institute of Mining, Metallurgy, and Petroleum (the CIM Guidelines, 2014). Classification was based upon: verification of concession title; review of drilling, sampling, assaying, and geology; drill grid spacing; assessment of the reliability of the geological model; appropriate in-situ dry bulk density for estimation of tonnage; OK variance statistics (kriging efficiency and regression slope values) and appropriate validation of samples with estimates block grade; and the reasonable prospects for economic extraction.

The Mineral Resource classification was guided by a life of mine pit design (Figure 15-11). All mineralisation more than 100 m outside the pit shell was reported as unclassified owing to low expectation for eventual economic extraction. The leached horizon, salar evaporites, and meta-sediments were coded as unclassified by default. Fresh mineralisation within the unclassified volume external to the pit shell is under assessment for future exploration targeting.

Measured, indicated, and inferred volumes were delineated by wireframes.

Figure 15-11 Measured, Indicated, and Inferred volumes at >0.13%Cu equivalent cut-off, shown relative to drill holes and the ultimate pit design limits



15.13 Copper equivalent grades

Since mineralisation consists of several metals of economic values, 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 grades:

$$\% \text{ Cu equivalent} = \% \text{ Cu} + (\text{Au revenue} / \text{Cu revenue}) / \% \text{ Cu} + (\text{Mo revenue} / \text{Cu revenue}) / \% \text{ Cu}$$

Where Cu revenue is based on 85% recovery and \$3.00/lb Cu price, Au revenue is based on 60% recovery and \$1,200/oz Au price, and Mo revenue is based on 40% recovery and \$12.00/lb Mo price.

The December 2019 Mineral Resource statement was reported at a 0.13% copper equivalent cut-off grade.

15.14 Mineral Resource statement

The December 2019 Mineral Resource estimate statement is presented in Table 15-10. It is reported using a 0.13% copper equivalent cut-off and the classification was guided by a life of mine pit shell.

The perched gold domain (102) within the upper portions of the leached horizon was estimated to contain more than 100 million tonnes with trace copper mineralisation and gold grades at approximately 0.6 g/t. This domain is unique relative to others in that it is depleted in copper and will be mined as part of the waste pre-strip and stockpiled separately. As such, the reported Mineral Resource excludes this perched gold mineralisation.

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The Mineral Resource estimate is inclusive of the Mineral Reserve estimate listed in Item 15.

Table 15-10 Taca Taca December 2019 Mineral Resource statement using a 0.13% copper equivalent cut-off grade

Classification	Volume (Mbcm)	Tonnes (Mt)	Density (t/m ³)	Cu grade (%)	Mo grade (%)	Au grade (g/t)	Cu metal (kt)	Mo metal (kt)	Au metal (koz)	Recovery (%)	Concentrate (%)
Measured	157.7	421.5	2.67	0.60	0.016	0.14	2,542.8	67.02	1,852.6	85.2	25.9
Indicated	671.6	1,781.8	2.65	0.39	0.011	0.07	6,908.0	197.52	4,199.5	83.9	24.9
Measured & Indicated	829.3	2,203.3	2.66	0.43	0.012	0.09	9,450.7	264.54	6,052.1	84.2	25.1
Inferred	269.4	716.9	2.66	0.31	0.009	0.05	2,206.0	65.15	1,182.7	84.3	24.6

Table 15-11 Grade and tonnage data for Measured and Indicated material at a range of copper equivalent cut-off grades

Classification	Cut-off grade Cu _{eq} (%)	Tonnes (Mt)	Cu grade (%)	Mo grade (%)	Au grade (g/t)
Measured	0.00	503.0	0.52	0.014	0.12
Indicated	0.00	2,355.0	0.31	0.010	0.06
Measured & Indicated	0.00	2,858.0	0.34	0.010	0.07
Measured	0.05	485.0	0.53	0.014	0.12
Indicated	0.05	2,210.0	0.33	0.010	0.06
Measured & Indicated	0.05	2,694.0	0.36	0.011	0.07
Measured	0.10	444.0	0.58	0.015	0.13
Indicated	0.10	1,961.0	0.36	0.011	0.07
Measured & Indicated	0.10	2,405.0	0.40	0.012	0.08
Measured	0.15	410.0	0.62	0.016	0.14
Indicated	0.15	1,705.0	0.40	0.011	0.08
Measured & Indicated	0.15	2,114.0	0.44	0.012	0.09
Measured	0.20	380.0	0.65	0.017	0.15
Indicated	0.20	1,467.0	0.44	0.012	0.08
Measured & Indicated	0.20	1,847.0	0.49	0.013	0.09
Measured	0.25	353.0	0.69	0.018	0.15
Indicated	0.25	1,248.0	0.49	0.012	0.09
Measured & Indicated	0.25	1,602.0	0.53	0.014	0.10
Measured	0.30	327.0	0.73	0.018	0.16
Indicated	0.30	1,059.0	0.53	0.013	0.09
Measured & Indicated	0.30	1,386.0	0.58	0.014	0.11
Measured	0.35	301.0	0.77	0.019	0.17
Indicated	0.35	896.0	0.58	0.013	0.10
Measured & Indicated	0.35	1,197.0	0.63	0.015	0.12
Measured	0.40	277.0	0.81	0.019	0.17
Indicated	0.40	754.0	0.63	0.014	0.11
Measured & Indicated	0.40	1,031.0	0.68	0.015	0.13
Measured	0.45	253.0	0.85	0.020	0.18
Indicated	0.45	635.0	0.68	0.014	0.11
Measured & Indicated	0.45	889.0	0.73	0.016	0.13
Measured	0.50	231.0	0.90	0.020	0.19
Indicated	0.50	533.0	0.73	0.015	0.12
Measured & Indicated	0.50	764.0	0.78	0.016	0.14

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.

15.15 Comparisons with previous estimate

The December 2019 Mineral Resource estimate includes the following additions:

- a 3D geological model to guide and control grade estimates
- improved domain definition considering weathering, alteration, lithology, and dominant mineralisation style and allowing for an estimate more representative of prevailing mineralisation

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- estimated density values, in comparison to mean values assigned per domain in the previous model
- a LUC change of support method, reflecting a revised grade interpolation process
- additional samples from the inclusion of four recently drilled metallurgical holes

The 2019 Mineral Resource classification used wireframe volumes to delineate Measured, Indicated, and Inferred resources based on confidence in kriged estimate and geological continuity. The classification was also guided by an updated life of mine pit shell based on criteria for reasonable economic extraction.

The previous Mineral Resource estimate (Table 15-12) was detailed in the PEA report of May 2013 (Ausenco) and uses a 0.3% copper equivalent cut-off grade based on a copper price of \$2.00/lb, a gold price of \$800/oz and a molybdenum price of \$12.00/lb. The inventory was constrained by a notional pit shell and included supergene and hypogene mineralisation.

Table 15-12 Previous Mineral Resource statement as at May 2013 (Ausenco PEA report) and using a 0.3% copper equivalent cut-off grade

Classification	Volume (Mbcm)	Tonnes (Mt)	Density (t/m ³)	Cu (%)	Au (g/t)	Mo (%)
Total Measured						
Supergene	-	-		-	-	-
Primary Sulphide	-	-		-	-	-
Subtotal	-	-		-	-	-
Total Indicated						
Supergene	264.3	701.0	2.65	0.60	0.08	0.009
Primary Sulphide	547.8	1,453.0	2.65	0.37	0.08	0.015
Subtotal	816.3	2,165.0	2.65	0.44	0.08	0.013
Total Meas. plus Ind.						
Supergene	264.3	701.0	2.65	0.60	0.08	0.009
Primary Sulphide	551.6	1,463.0	2.65	0.37	0.08	0.015
Total	816.3	2,165.0	2.65	0.44	0.08	0.013
Total Inferred	347.5	921.0	2.65	0.37	0.05	0.012

There are marginal changes to the updated FQM Mineral Resource statement when compared with the previous May 2013 estimate. Changes are largely associated with improved cut-off grades from the associated Mineral Reserve conversion studies.

It is the opinion of the QP that the resulting changes to this Mineral Resource statement reflect the confidence in the underlying data and that the estimates are believed representative of the prevailing mineralisation. Detailed grade and geology knowledge has been gained from analysis of sequential copper data thereby enabling an improved understanding and definition of weathering profiles and the associated impact on style of copper mineralisation.

ITEM 16 MINERAL RESERVE ESTIMATE

16.1 Introduction

Detailed technical information provided under this item relates specifically to the Mineral Reserve estimate completed for this Technical Report and based on 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.

16.2 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 a number of metallurgical domains.

The ultimate (optimal) and lesser pit optimisation shells derived from the use of Whittle Four-X pit optimisation software then enabled the creation of practical and detailed open pit phase 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 cashflow modelling, is described in Item 16.

16.3 Open pit optimisation

An initial check optimisation by the Company was done on the Ausenco Mineral Resource model to emulate and confirm the pit shells adopted by them for mine planning. The Ausenco optimisation was done on the basis of a \$2.00/lb copper price, and the stated rationale for this was (Ausenco, 2013):

The \$2.00/lb Cu shell with Indicated and Inferred resources was selected as the basis for subsequent pit designs. A shell at higher Cu prices through \$2.75/lb could have been selected given recent market history, more than doubling the contained mineral resource tonnages – but at lower incremental head grades. This would require additional tailings storage capacity and would extend the Project life well beyond 30 years. It was felt that the \$2.00/lb Cu shell would capture the bulk of the Project’s potential value and would be more reasonable to use at this stage of evaluation.

The latest optimisation by the Company was done using a (reblocked) mine planning model derived from the FQM Mineral Resource estimate model described in Item 14. The optimal pit shell was selected on an undiscounted cashflow basis and with the objective of achieving a total plant feed tonnage inventory equal to or better than that modelled by Lumina in their cashflow models (Ausenco, 2013), and after constraining the model to avoid mining into the salar.

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Further to the comments made by geotechnical engineers Wyllie and Norrish (W&N, 2016), the new optimisation was constrained to prevent pit shells from daylighting into the salar, with the subsequent risk of water ingress (refer to Item 16.2.3).

The FQM optimisation was based on a long term \$3.00/lb copper price, and with a design pit shell selected after considering the undiscounted cashflow differences over a range of revenue factors. In this respect, the shell selection approach was similar to that adopted by Lumina in 2013.

16.3.1 Pit slope design criteria

Pit optimisation input included overall slope design angles as shown in Figure 17-3. The geotechnical engineering basis for these design angles is outlined in Item 16.2.3.

16.3.2 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.

16.3.3 Metal prices

The optimisation inputs for metal prices were as follows:

- copper = \$3.00/lb (\$6,614/t)
- molybdenum = \$12.00/lb (\$26,455/t)
- gold = \$1,200/oz

16.3.4 Metal recoveries

For optimisation purposes, each ore type within the mine planning model was assigned a specific copper recovery as per the values in Table 16-1 (i.e. grade dependant copper recovery equations are as yet unavailable). Molybdenum was assigned a constant recovery of 40% and gold was assigned a constant recovery of 60%. The recovery values in Table 16-1 reflect the information tabled in Item 13.4.

Table 16-1 Pit optimisation input, copper recovery parameters

Geological Process	Copper species	Relative proportion	% of Total Ore	Domain (DOM) numeric value	Material type (MATTYPE) Name	July 2020 Cu Recovery, %	July 2020 Cu Con Grade, % Cu
Supergene	Chalcocite, chalcopyrite, black Cu oxides, soluble copper	CC>CPY + Sol Cu	3%	203	MIXED	65	15
				203	MIXED	65	15
				203	MIXED	65	15
				203	MIXED	74	20
				203	MIXED	74	20
				203	MIXED	70	20
Supergene	Chalcocite, chalcopyrite	CC>CPY	19%	304, 305, 310	SECONDARY	83	25
				304, 305, 310	SECONDARY	83	25
				304, 305, 310	SECONDARY	80	22
				304, 305, 310	SECONDARY	88	34
				304, 305, 310	SECONDARY	86	34
				304, 305, 310	SECONDARY	84	30
	Chalcocite, Chalcopyrite, bornite			304, 305, 310	SECONDARY	88	35
				304, 305, 310	SECONDARY	88	35
				304, 305, 310	SECONDARY	86	32
Supergene	Chalcocite, chalcopyrite	CC>>CPY	59%	307,308, 309	SUPERGENE MIXED	85	25
				307,308, 309	SUPERGENE MIXED	85	25
				307,308, 309	SUPERGENE MIXED	82	22
	Chalcocite, chalcopyrite, bornite			307,308, 309	SUPERGENE MIXED	88	30
				307,308, 309	SUPERGENE MIXED	88	30
				307,308, 309	SUPERGENE MIXED	86	26
Hypogene	Chalcopyrite, chalcocite	CPY>CC	20%	306	PRIMARY	86	25
				306	PRIMARY	86	25
				306	PRIMARY	84	22
	Chalcopyrite, chalcocite, bornite			306	PRIMARY	90	30
				306	PRIMARY	90	30
				306	PRIMARY	86	26

The overall tonnes weighted recovery and concentrate grades used as input to the optimisation were as listed in Table 16-2.

Table 16-2 Pit optimisation input, average copper recovery parameters

	Cu rec (%)	Cu con (%)
Supergene	68.3	17.4
Supergene	81.4	24.6
Supergene	85.3	25.9
average	83.8	25.3
Hypogene	86.0	25.0

16.3.5 Mining costs

The mining costs referenced for pit optimisation were estimated from preliminary mine planning and cost modelling. These costs vary with pit depth and haulage distance, and hence the following algorithm was derived for input to the optimisation:

- Ore mining (\$/t) > 3,511 mRL = \$1.60/t
- Ore mining (\$/t) < 3,511 mRL = \$9.04 – (0.0021 x RL)
- Waste mining (\$/t) > 3,481 mRL = \$1.45/t
- Waste mining (\$/t) < 3,481 mRL = \$8.37 – (0.002 x RL)

Figure 16-1 shows a graph which depicts the variance trend and reflects the cost calculation algorithm. To note is that the varying ore mining cost algorithm incorporates the estimated stockpile rehandle cost.

Figure 16-1 Graph showing mining cost variance with pit depth



Item 22.5.2 describes the comprehensive estimation of the mining costs for input to the cashflow model. The unit costs, varying with mining depth for the cashflow model estimates, are typically 5% less for ore and 10% higher for waste than those that would be predicted from the algorithm pertaining to Figure 16-1.

16.3.6 Operating costs

Since the Project will be mill constrained, the operating costs are the sum of the fixed and variable costs (other than mining costs). These costs are summarised below:

- fixed costs (equivalent general and administration (G&A) costs in variable terms) = \$1.05/t (refer to Item 22.5)
- process fixed plus variable operating costs = \$4.82/t (this was subsequently re-estimated as \$4.69/t for the cashflow model; refer to Item 22.5)
- plus allowances for:
 - rail load-out maintenance = \$0.06/t
 - water supply tariff = \$0.02/t
- yielding a total operating cost (PROCAST in Whittle 4X terms) = \$5.95/t processed

16.3.7 Metal costs

The metal costs (i.e. transport and refining charges (TCRCs) and royalties) in Table 16-3 reflect the information outlined in Item 21.6. Note that the concentrate charges listed are inclusive of treatment and freight costs. The net return values (i.e. revenue less metal costs) are listed in Table 16-4.

In this instance, the included royalties are listed on a gross revenue basis. In the cashflow model, the royalties are modelled as net of the charges described in Item 4.6.

Table 16-3 Pit optimisation input, metal costs

Metal Costs	Units	Cu at \$3.00/lb	
		Primary	Non-primary
Copper concentrate charges:			
Cu payable	%	96.2%	96.2%
Cu conc. grade	%	25.0%	25.3%
Concentrate rail transport	\$/dmt	\$60.00	\$60.00
Port charges	\$/dmt	\$13.50	\$13.50
Sea freight charges	\$/dmt	\$37.00	\$37.00
Cu treatment	\$/dmt	\$90.00	\$90.00
Cu refining (on payable)	\$/lb	\$0.09	\$0.09
Copper metal cost	\$/lb	\$0.468	\$0.463
Molybdenum concentrate charges:			
Mo payable	%	86.0%	86.0%
Mo con grade	%	47.0%	47.0%
Concentrate rail transport	\$/dmt	\$48.00	\$48.00
Port charges	\$/dmt	\$13.50	\$13.50
Sea freight charges	\$/dmt	\$37.00	\$37.00
Mo treatment	\$/dmt	\$68.19	\$68.19
Mo refining (on payable)	\$/lb	\$0.00	\$0.00
Molybdenum metal cost	\$/lb	\$0.187	\$0.187
Au in concentrate charges			
Au payable	%	90.0%	90.0%
Au refining (on payable)	\$/oz	\$5.10	510.0%
Gold metal cost	\$/oz	\$4.59	\$4.59
Royalties:			
All metals	%	4.5%	4.5%
Metal Costs:			
Cu Metal Cost	\$/lb	\$0.60	\$0.60
Mo Metal Cost	\$/lb	\$0.73	\$0.73
Au Metal Cost	\$/oz	\$58.59	\$58.59

Table 16-4 Pit optimisation input, net return

Net Return	Units	Cu at \$3.00/lb	
		Primary	Non-primary
Processing Parameters:			
Cu recovery	%	86%	84%
Mo recovery (thru Mo con)	%	40%	40%
Au recovery	%	60%	60%
Price less Metal Costs:			
Cu Metal Price	\$/lb	\$3.00	\$3.00
Cu Metal Cost	\$/lb	\$0.60	\$0.60
Cu Net Return	\$/lb	\$2.40	\$2.40
Cu Net Return (recovered)	\$/lb	\$2.06	\$2.01
Mo Metal Price	\$/lb	\$12.00	\$12.00
Mo Metal Cost	\$/lb	\$0.73	\$0.73
Mo Net Return	\$/lb	\$11.27	\$11.27
Cu _{eq} Net Return (recovered)	\$/lb	\$0.06	\$0.06
Au Metal Price	\$/oz	\$1,200	\$1,200
Au Metal Cost	\$/oz	\$58.59	\$58.59
Au Net Return	\$/oz	\$1,141	\$1,141
Cu _{eq} Net Return (recovered)	\$/lb	\$0.14	\$0.14
Total Net Return (recovered)	\$/lb	\$2.26	\$2.21
Total Net Return (recovered)	\$/10kg	\$49.75	\$48.81

16.3.8 Marginal cut-off grades

Whittle uses the following simplified formula to calculate the marginal cut-off grade as listed in Table 16-5.

$$\text{Marginal COG} = (\text{PROCOST} \times \text{MINDIL}) / (\text{NR})$$

where PROCOST is the total operating cost

MINDIL is the mining dilution factor

NR is the net return

Table 16-5 Overall average marginal cut-off grade

Marginal cut off grade	Units	Cu at \$3.00/lb	
		Primary	Non-primary
PROCOST	\$/t ore	\$5.95	\$5.95
MINDIL		1.05	1.05
TOTAL NET RETURN (recovered)	\$/10kg	\$49.75	\$48.81
C/O Equivalent grade	%Cu	0.13	0.13

Had the subsequently revised process operating cost of \$4.69/t (yielding a PROCOST of \$5.82/t, Item 21.5) been used in the optimisation process, the impact to the marginal cut-off grade would have been negligible.

16.3.9 Optimisation results

The optimisation results are listed in Table 16-6 and are shown graphically in Figure 16-2.

For all of the input parameters listed above, the optimal pit shell at \$3.00/lb Cu (the revenue factor 1.00 pit) is shell no. 15. This yields a total plant feed of 1,943 Mt at an average grade of 0.43 %Cu; the waste totals 3,210.6 Mt and the overall strip ratio is 1.7 : 1.

Table 16-7 shows a highlighted lesser pit shell no. 9, which corresponds to a revenue factor of 0.70. This factor results in an adjustment of the total revenue for this particular shell equating to the following metal prices:

- copper = \$2.10/lb (\$4,630/t)
- molybdenum = \$8.40/lb (\$18,519/t)
- gold = \$850/oz

These prices would have the impact of reducing the plant feed tonnage and increasing the average feed grades, such that the average marginal cut-off grade would increase from 0.13 %Cu_{eq} to 0.20 %Cu_{eq}.

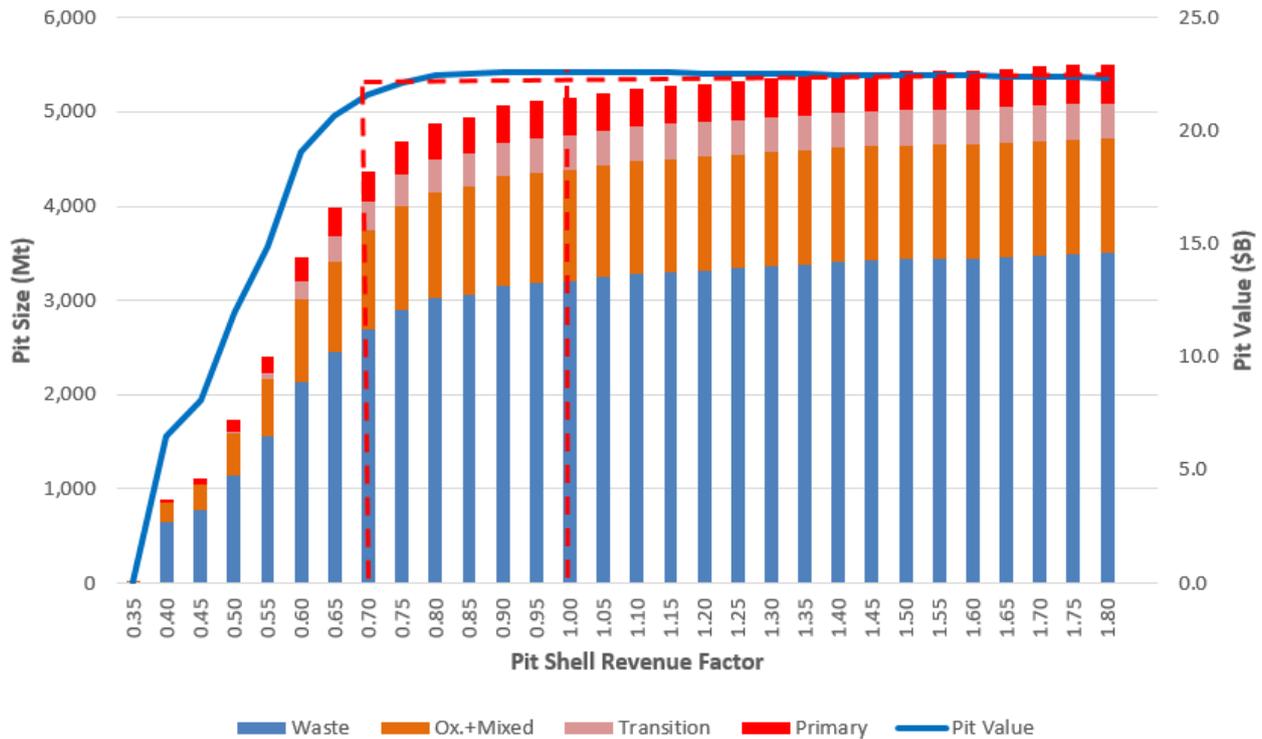
The difference in plant feed between the two shells is 263 Mt at an average grade of 0.31% Cu (a 13.5% tonnage reduction, for a 10% reduction in copper metal). The waste tonnage reduction is about 16%.

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Table 16-6 Summary of optimisation results

Pit Shell	Rev. Factor at \$3.00/lb Cu	Pit Size (Mt)	Waste (Mt)	Strip ratio waste/ore (t:t)	Total Plant Feed (Mt)	Plant Feed		Plant Feed			Plant Feed Grade (diluted)			Recovery			Recovered Metal		
						Measured (Mt)	Indicated (Mt)	Supergene		Hypogene Primary (Mt)	Cu (%)	Au (g/t)	Mo (ppm)	Cu (%)	Au (%)	Mo (%)	Cu (Mt)	Au (koz)	Mo (Mt)
								Ox.+Mixed (Mt)	Trans. (Mt)										
1	0.30	0.0	0.0	0.6	0.0	0.0	0.0	0.0	0.0	0.0	0.96	0.59	128.64	74.0%	60.0%	40.0%	0.00	0.3	0.0
2	0.35	0.8	0.5	1.4	0.3	0.0	0.3	0.0	0.0	0.0	0.82	0.57	168.37	74.0%	60.0%	40.0%	0.0	3.7	0.0
3	0.40	892.7	653.4	2.7	239.4	147.4	92.0	207.1	0.0	32.3	0.75	0.15	159.99	85.5%	60.1%	40.0%	1.5	693.8	15.3
4	0.45	1,106.9	782.5	2.4	324.4	192.5	131.8	265.6	0.4	58.4	0.70	0.15	158.79	85.6%	59.9%	40.0%	1.9	911.7	20.6
5	0.50	1,735.4	1,147.1	2.0	588.3	313.1	275.2	449.3	10.7	128.2	0.59	0.13	151.45	85.8%	59.9%	40.0%	3.0	1,439.4	35.6
6	0.55	2,401.8	1,555.4	1.8	846.4	377.3	469.1	613.7	56.4	176.2	0.54	0.12	145.96	85.8%	59.9%	40.0%	3.9	1,923.0	49.4
7	0.60	3,454.8	2,139.8	1.6	1,315.0	402.3	912.7	872.5	183.6	258.9	0.48	0.10	133.09	85.6%	60.2%	40.0%	5.4	2,570.8	70.0
8	0.65	3,980.5	2,453.1	1.6	1,527.4	408.2	1,119.2	964.1	270.3	293.0	0.46	0.10	125.84	85.6%	59.8%	40.0%	6.0	2,821.0	76.9
9	0.70	4,372.8	2,692.7	1.6	1,680.1	408.7	1,271.4	1,048.7	303.7	327.7	0.45	0.09	122.70	85.5%	60.1%	40.0%	6.4	2,985.7	82.5
10	0.75	4,691.2	2,898.6	1.6	1,792.6	408.7	1,383.9	1,099.5	335.9	357.2	0.44	0.09	120.17	85.6%	59.8%	40.0%	6.7	3,102.3	86.2
11	0.80	4,879.0	3,019.5	1.6	1,859.5	408.7	1,450.8	1,133.1	350.4	376.0	0.44	0.09	119.26	85.4%	60.2%	40.0%	6.9	3,165.9	88.7
12	0.85	4,951.2	3,065.7	1.6	1,885.5	408.7	1,476.8	1,146.3	355.7	383.5	0.43	0.09	118.85	85.6%	59.8%	40.0%	7.0	3,187.5	89.6
13	0.90	5,068.2	3,148.9	1.6	1,919.3	408.7	1,510.6	1,165.8	360.9	392.6	0.43	0.09	118.50	85.5%	59.9%	40.0%	7.1	3,217.4	91.0
14	0.95	5,115.0	3,180.3	1.6	1,934.6	408.7	1,526.0	1,175.2	362.9	396.6	0.43	0.09	118.42	85.4%	59.7%	40.0%	7.1	3,229.3	91.6
15	1.00	5,153.4	3,210.6	1.7	1,942.8	408.7	1,534.1	1,179.2	365.6	398.0	0.43	0.09	118.28	85.5%	60.3%	40.0%	7.1	3,237.7	91.9
16	1.05	5,200.4	3,245.3	1.7	1,955.1	408.7	1,546.4	1,186.1	367.4	401.6	0.43	0.09	118.22	85.5%	60.1%	40.0%	7.2	3,248.2	92.5
17	1.10	5,251.1	3,283.4	1.7	1,967.7	408.7	1,559.0	1,195.4	368.4	403.8	0.43	0.09	118.16	85.4%	59.9%	40.0%	7.2	3,256.6	93.0
18	1.15	5,277.3	3,304.0	1.7	1,973.3	408.7	1,564.6	1,198.6	369.7	404.9	0.43	0.09	118.08	85.5%	59.8%	40.0%	7.2	3,260.9	93.2
19	1.20	5,299.8	3,322.3	1.7	1,977.5	408.7	1,568.8	1,200.3	370.5	406.6	0.43	0.09	117.99	85.5%	59.7%	40.0%	7.2	3,264.3	93.3
20	1.25	5,321.6	3,339.7	1.7	1,981.9	408.7	1,573.2	1,202.9	371.6	407.4	0.43	0.09	117.94	85.4%	60.3%	40.0%	7.2	3,267.2	93.5
21	1.30	5,356.4	3,368.7	1.7	1,987.7	408.7	1,579.0	1,206.2	372.5	409.0	0.43	0.09	117.92	85.5%	60.2%	40.0%	7.2	3,272.2	93.8
22	1.35	5,374.6	3,384.5	1.7	1,990.1	408.7	1,581.5	1,207.7	372.9	409.5	0.43	0.09	117.91	85.5%	60.2%	40.0%	7.2	3,274.1	93.9
23	1.40	5,406.7	3,412.4	1.7	1,994.3	408.7	1,585.6	1,209.4	373.1	411.8	0.43	0.09	117.91	85.4%	60.1%	40.0%	7.2	3,278.1	94.1
24	1.45	5,421.4	3,424.9	1.7	1,996.5	408.7	1,587.8	1,210.4	373.5	412.6	0.42	0.09	117.87	85.6%	60.1%	40.0%	7.2	3,279.5	94.1
25	1.50	5,432.8	3,434.6	1.7	1,998.1	408.7	1,589.5	1,210.8	373.9	413.5	0.42	0.09	117.82	85.5%	60.1%	40.0%	7.2	3,280.5	94.2
26	1.55	5,437.4	3,438.8	1.7	1,998.6	408.7	1,589.9	1,210.9	374.2	413.5	0.42	0.09	117.80	85.5%	60.1%	40.0%	7.2	3,280.9	94.2
27	1.60	5,442.9	3,443.8	1.7	1,999.2	408.7	1,590.5	1,211.0	374.6	413.6	0.42	0.09	117.78	85.5%	60.1%	40.0%	7.2	3,281.2	94.2
28	1.65	5,461.0	3,460.1	1.7	2,000.9	408.7	1,592.2	1,212.2	374.8	413.9	0.42	0.09	117.77	85.5%	60.0%	40.0%	7.3	3,282.2	94.3
29	1.70	5,484.4	3,481.4	1.7	2,003.1	408.7	1,594.4	1,212.7	375.0	415.3	0.42	0.09	117.79	85.5%	60.0%	40.0%	7.3	3,284.8	94.4
30	1.75	5,497.7	3,493.4	1.7	2,004.3	408.7	1,595.6	1,213.0	375.2	416.1	0.42	0.09	117.77	85.4%	60.0%	40.0%	7.3	3,286.0	94.4
31	1.80	5,506.4	3,501.2	1.7	2,005.1	408.7	1,596.5	1,213.7	375.3	416.1	0.42	0.09	117.77	85.4%	60.0%	40.0%	7.3	3,286.6	94.5

Figure 16-2 Graphical pit optimisation results



The undiscounted operating cashflow from the revenue factor 1.00 pit shell (no. 15) is \$22.6B, compared with \$21.6B for the revenue factor 0.70 pit shell (no. 9). This is a 4.5% difference overall and reflects that the costs associated with mining and processing the inventory beyond shell no.9 are marginally recouped by the additional revenue. This comparison is illustrated in Figure 16-2.

If the respective cashflows were to be discounted over 32 years, then the cashflow differences would likely be negligible.

16.4 Optimisation sensitivity analyses

Table 16-7 summarises the results of optimisation sensitivity analyses, assessing the impact of varying the input mining costs, processing costs and processing recovery. Adjustments to recovery would have the same magnitude of effect as adjustments to the insitu ore grades. The results in Table 16-7 indicate that, as expected, the undiscounted cashflow is more impacted by changes in process recovery (since this impacts directly on revenue) than to mining or processing cost changes.

Additional sensitivity analyses indicate that for such a relatively deep open pit, there appears to be a 10% reduction in undiscounted cashflow for only a modest four degree flattening of the overall slope angle. This highlights the importance of the geotechnical drilling campaign, analysis and pit design review, the recommendations for which are outlined in Item 26.

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Table 16-7 Summary of optimisation sensitivity analysis results

Sensitivity case	Pit Shell	Rev. Factor at \$/lb Cu	Pit Size (Mt)	Waste Mt (Mt)	Strip ratio waste/ore (t:t)	Total Mt Plant Feed (Mt)	Plant Feed		Plant Feed			Plant Feed Grade (diluted)			Recovery			Recovered Metal			Change in Pit Value (%)
							Measured (Mt)	Indicated (Mt)	Supergene		Hypogene Primary (Mt)	Cu (%)	Au (g/t)	Mo (ppm)	Cu (%)	Au (%)	Mo (%)	Cu (Mt)	Au (koz)	Mo (kt)	
									Ox+Mixed (Mt)	Trans. (Mt)											
Basecase: Mo 40% Au 60%	15	1.00	5,153.4	3,210.6	1.7	1,942.8	408.7	1,534.1	1,179.2	365.6	398.0	0.43	0.09	118.28	85.5%	60.3%	40.0%	7.1	3,237.7	91.9	
Basecase + mcost+5%	15	1.00	5,136.1	3,199.2	1.7	1,936.8	408.5	1,528.4	1,176.7	363.4	396.8	0.43	0.09	118.42	85.4%	59.7%	40.0%	7.1	3,232.5	91.7	98%
Basecase + mcost-5%	15	1.00	5,187.0	3,234.3	1.7	1,952.7	408.8	1,543.8	1,184.9	367.2	400.7	0.43	0.09	118.20	85.5%	60.1%	40.0%	7.1	3,245.7	92.3	102%
Basecase + pcost+5%	14	1.00	5,140.4	3,236.3	1.7	1,904.1	404.3	1,499.8	1,158.3	354.1	391.7	0.44	0.09	119.31	85.5%	60.3%	40.0%	7.1	3,211.7	90.9	97%
Basecase + pcost-5%	15	1.00	5,185.7	3,198.5	1.6	1,987.2	413.2	1,574.0	1,204.1	377.0	406.0	0.42	0.09	117.29	85.4%	60.2%	40.0%	7.2	3,267.2	93.2	103%
Basecase + recov+5%	15	1.00	5,191.7	3,202.7	1.6	1,989.0	413.2	1,575.8	1,205.3	377.4	406.3	0.42	0.09	117.27	89.8%	63.1%	42.0%	7.5	3,432.0	98.0	110%
Basecase + recov-5%	14	1.00	5,115.9	3,219.9	1.7	1,896.0	403.9	1,492.1	1,153.4	351.8	390.9	0.44	0.09	119.43	81.2%	56.7%	38.0%	6.7	3,043.9	86.0	90%

16.5 Open pit design

Detailed pit designs were produced from pit optimisation shells (based on the revenue factor 0.70 pit optimisation shell as the selected ultimate) to account for batters, berms and haul roads. These practical designs provided the definition required for the mining production schedule described in Item 16.3.

The specific pit design parameters are listed in Item 16.2.1.

Table 16-8 is a validation inventory report between the detailed ultimate pit design and the optimisation shell upon which it was based. The RF 0.70 shell tonnages and grades are (mining) undiluted/recovered for direct comparison with the pit design. The validation shows excellent correlation.

Table 16-8 Validation between selected pit shell and design pit

		TR Pit Shell RF 0.7	TR Pit Design	Design vs Shell
Mineralised Waste				
COG (eff. \$2.10/lb)	%Cu _{eq}	0.19		
Mined ore	Mt	300.6	319.3	106%
Mined grade	%Cu	0.16	0.15	
	ppm Mo	76.8		
	g/t Au	0.04		
In situ Cu metal	kt	466.0	493.8	106%
LG+HG				
COG (eff. \$2.10/lb)	%Cu _{eq}	0.62		
Mined ore	Mt	1,383.6	1,443.7	104%
Mined grade	%Cu	0.54	0.53	
	ppm Mo	140.3		
	g/t Au	0.11		
In situ Cu metal	kt	7,462.0	7,648.0	102%
Total Plant Feed				
COG (eff. \$2.10/lb)	%Cu _{eq}	0.55		
Mined ore	Mt	1,684.2	1,762.9	105%
Mined grade	%Cu	0.47	0.46	
	ppm Mo	129.0	127.6	
	g/t Au	0.10	0.09	
In situ Cu metal	kt	7,928.0	8,141.8	103%
Waste mined	Mt	2,692.7	2,771.8	103%
Total mined	Mt	4,376.9	4,534.8	104%

16.5.1 Staged and ultimate pits

Five phases and an ultimate pit design were produced, incorporating the detailed design parameters listed in Item 16.2. Table 16-9 lists the inventory within each pit phase design, whilst Figure 16-3 to Figure 16-8 show plan views of the phase and ultimate pit designs.

Table 16-9 Tabulation of the inventory within each pit phase and ultimate pit design

Parameter	Unit	Phase 1a	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	TOTAL
Ore mined								
Ore mined	Mt	34.0	109.0	79.9	167.2	506.5	866.4	1,762.9
Mined grade	%Cu	0.51	0.79	0.70	0.67	0.44	0.37	0.46
In situ Cu metal	kt	172.1	860.5	558.9	1,119.5	2,227.1	3,203.6	8,141.8
Waste	Mt	148.8	212.3	290.6	416.8	724.6	987.1	2,771.8
Total mined	Mt	182.7	321.3	370.5	584.0	1,231.1	1,853.5	4,534.8
Strip ratio		4.4	1.9	3.6	2.5	1.4	1.1	1.6

Figure 16-10 shows a perspective view of the design ultimate pit with maximum dimensions annotated.

Figure 16-3 Taca Taca phase 1a design pit

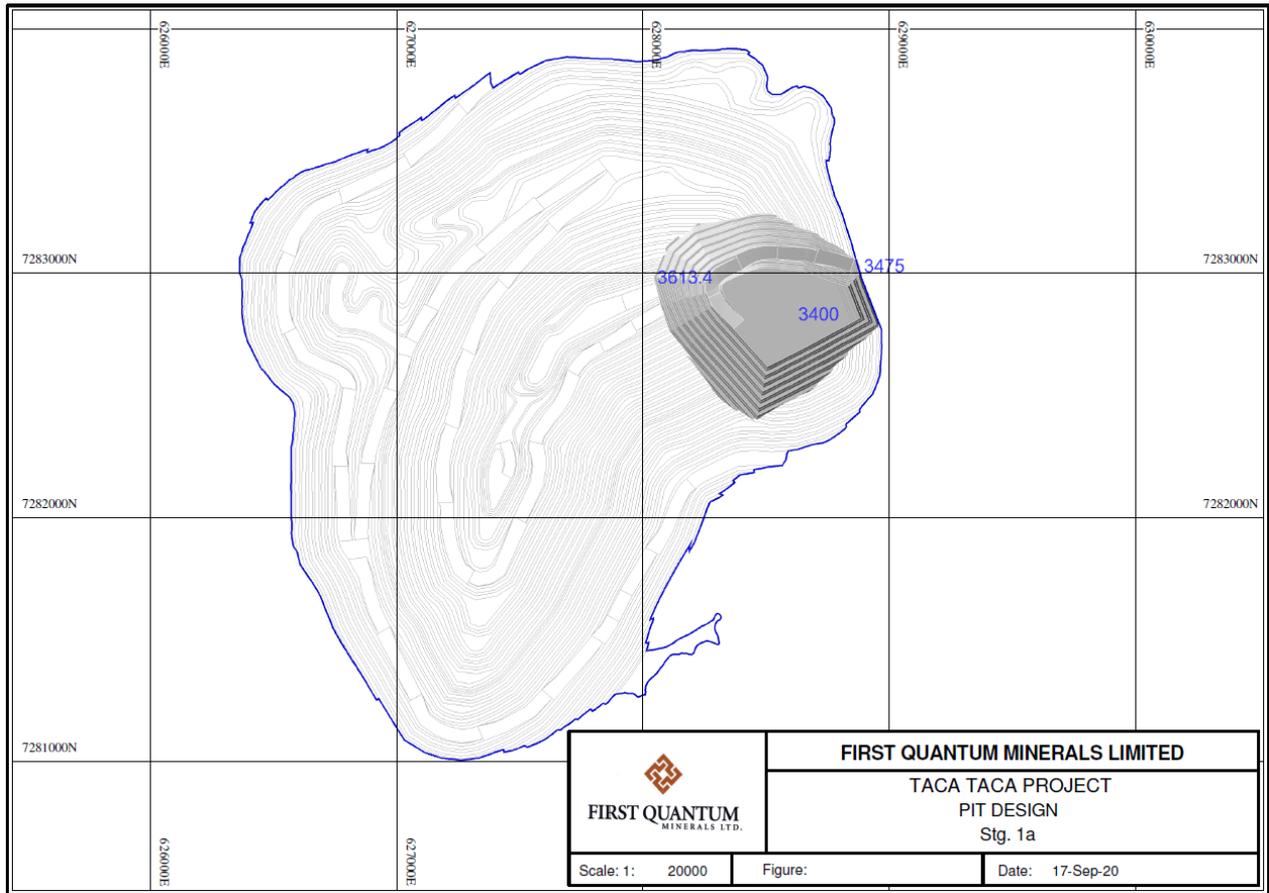


Figure 16-4 Taca Taca phase 1 design pit

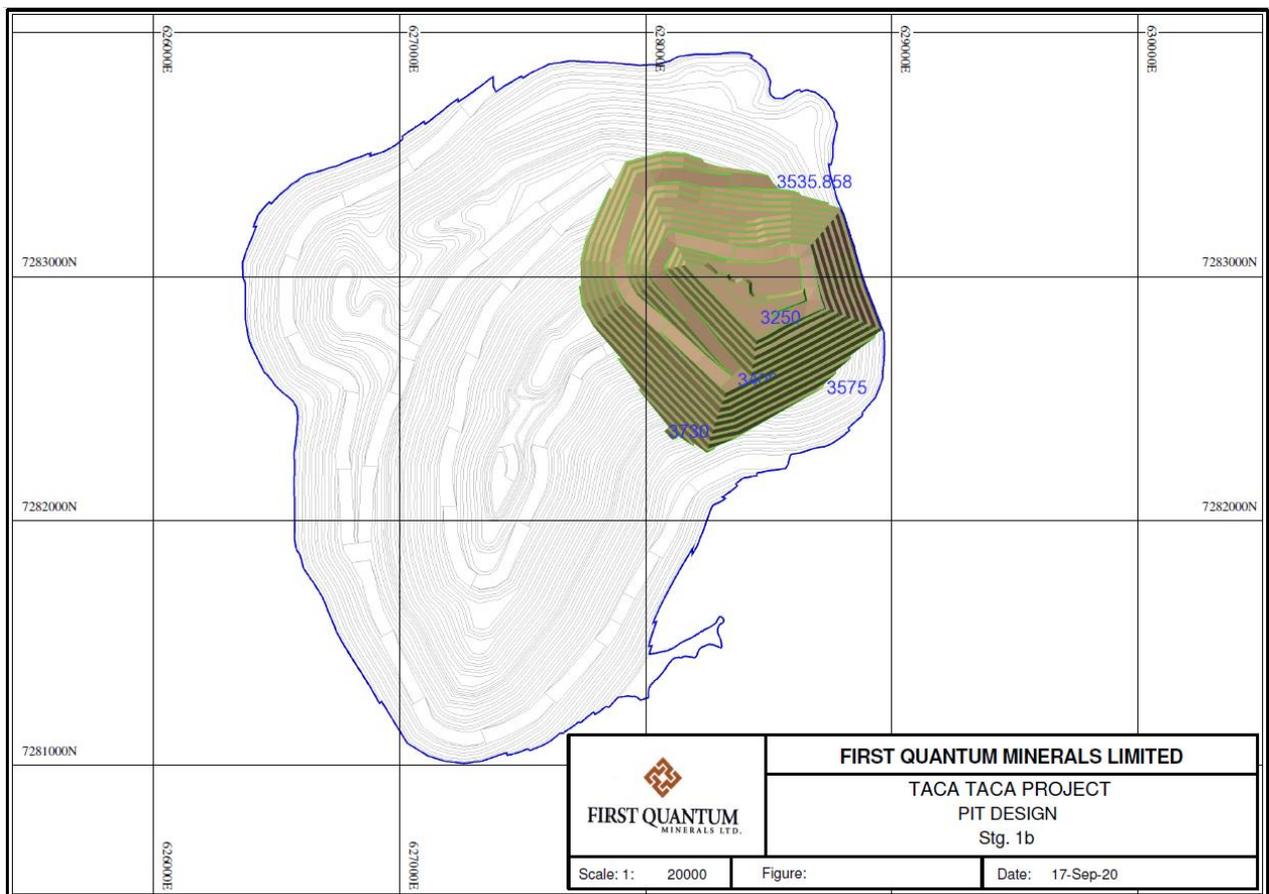


Figure 16-5 Taca Taca phase 2 design pit

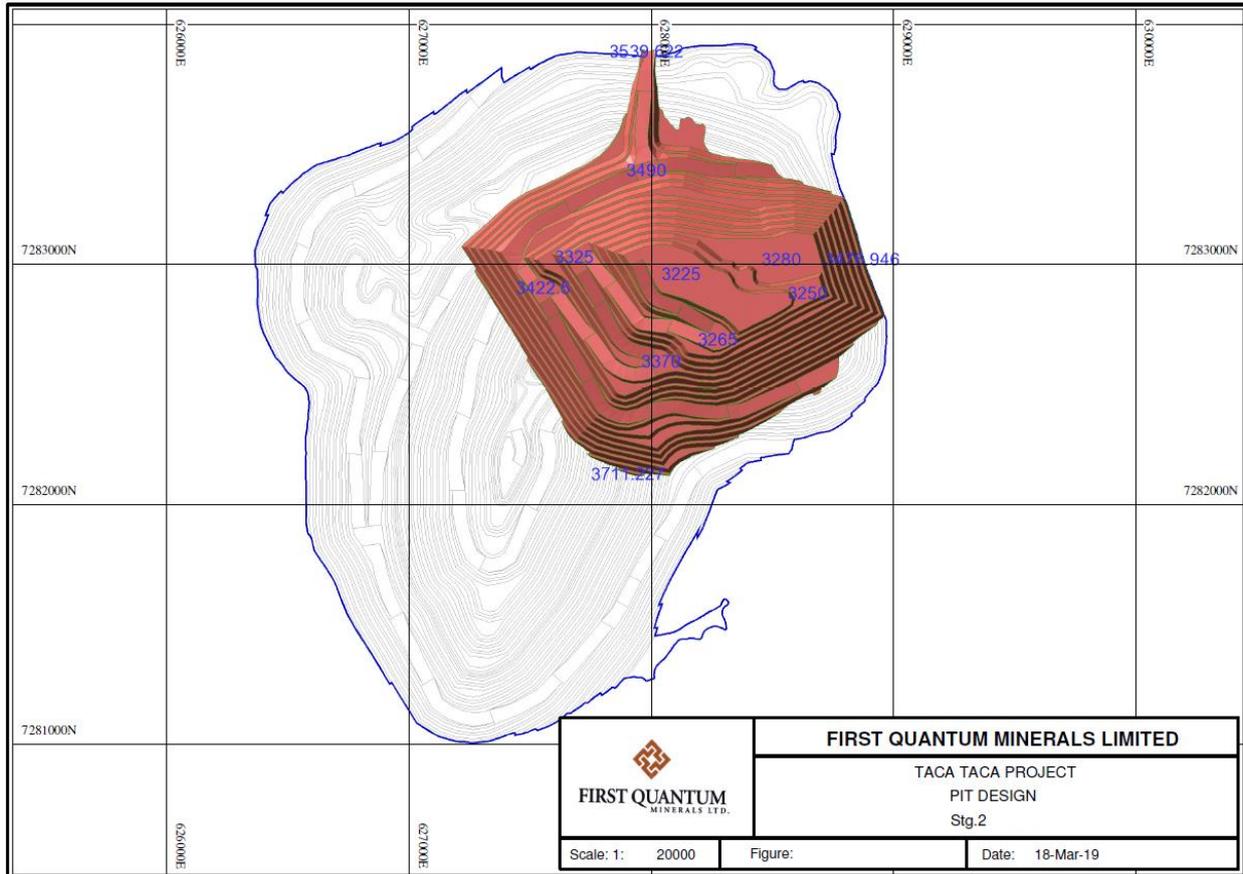


Figure 16-6 Taca Taca phase 3 design pit

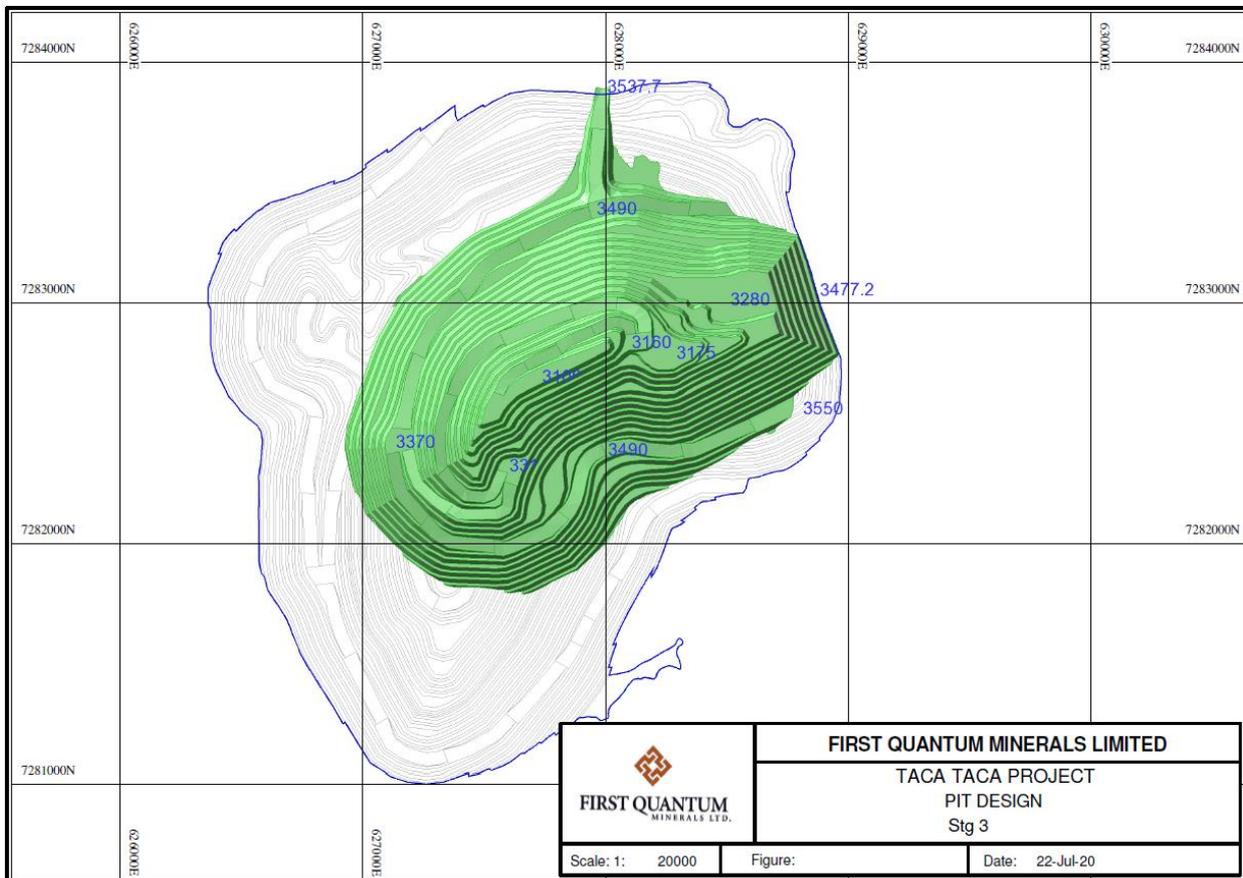


Figure 16-7 Taca Taca phase 4 design pit

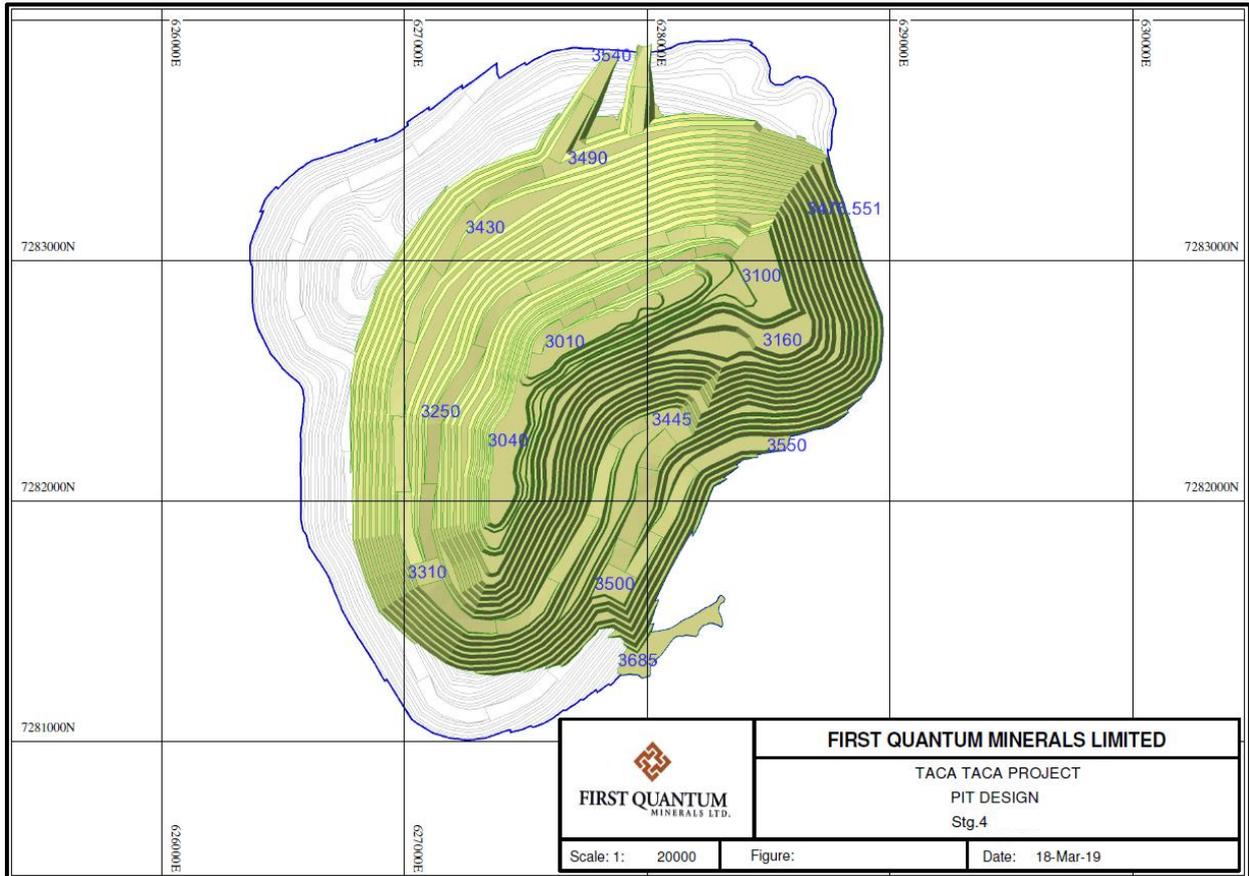


Figure 16-8 Taca Taca ultimate design pit



Figure 16-9 Taca Taca overlain phase and ultimate pit designs

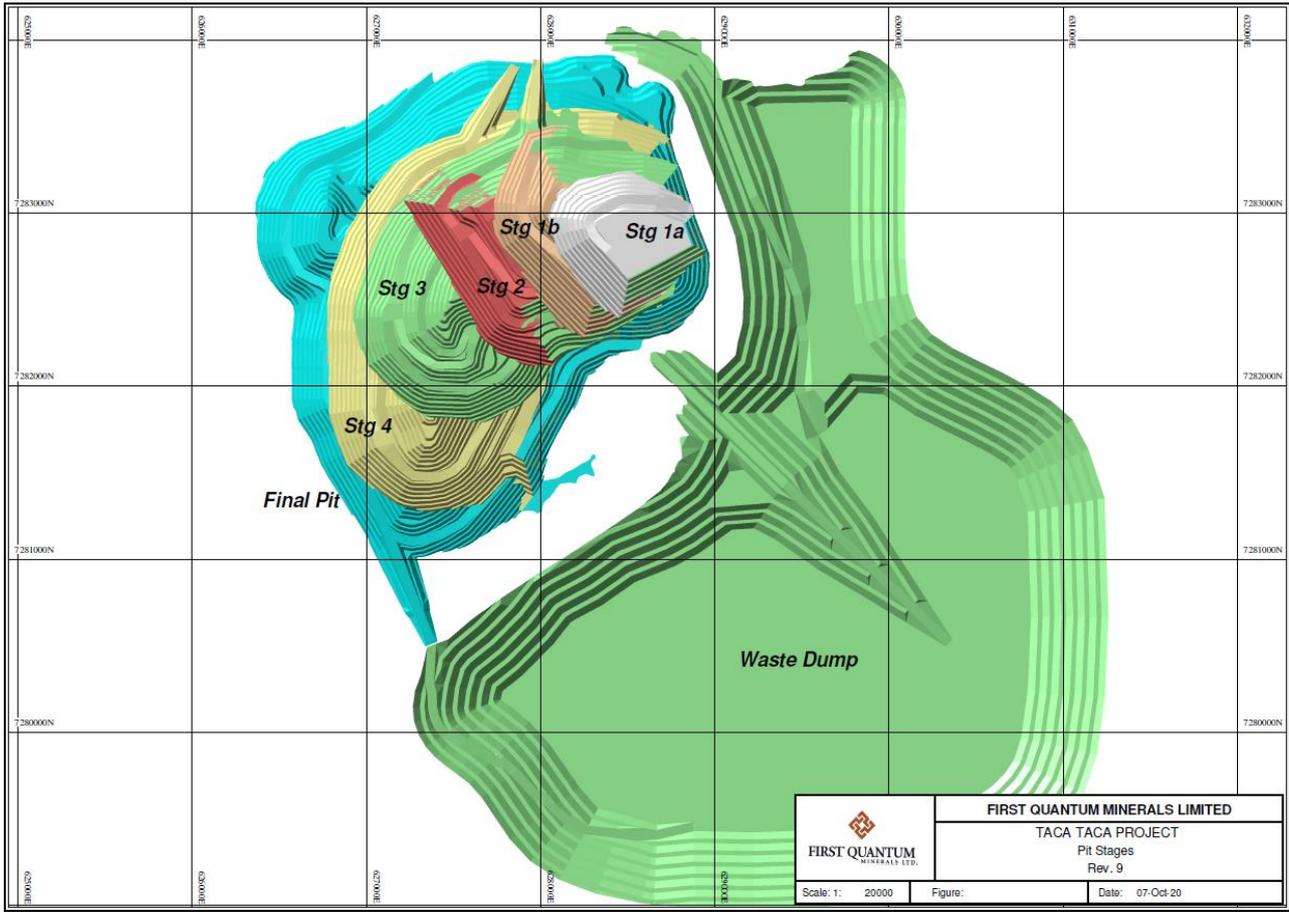
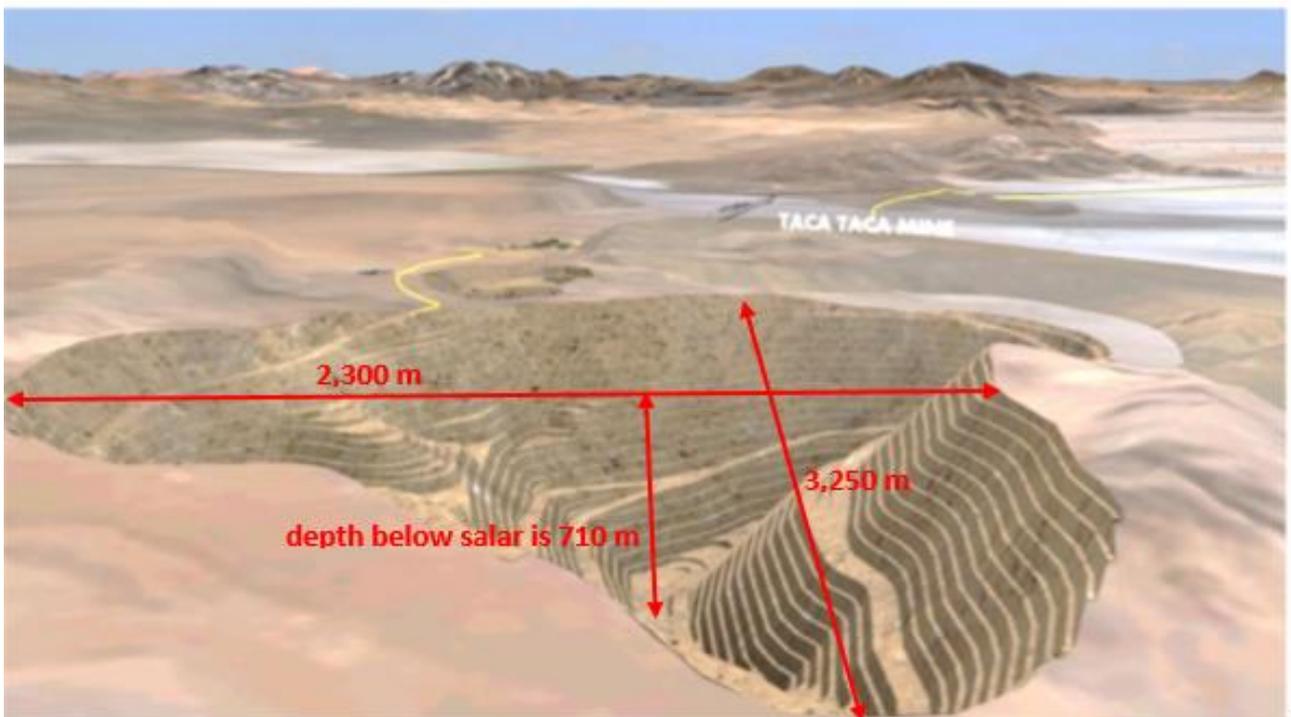


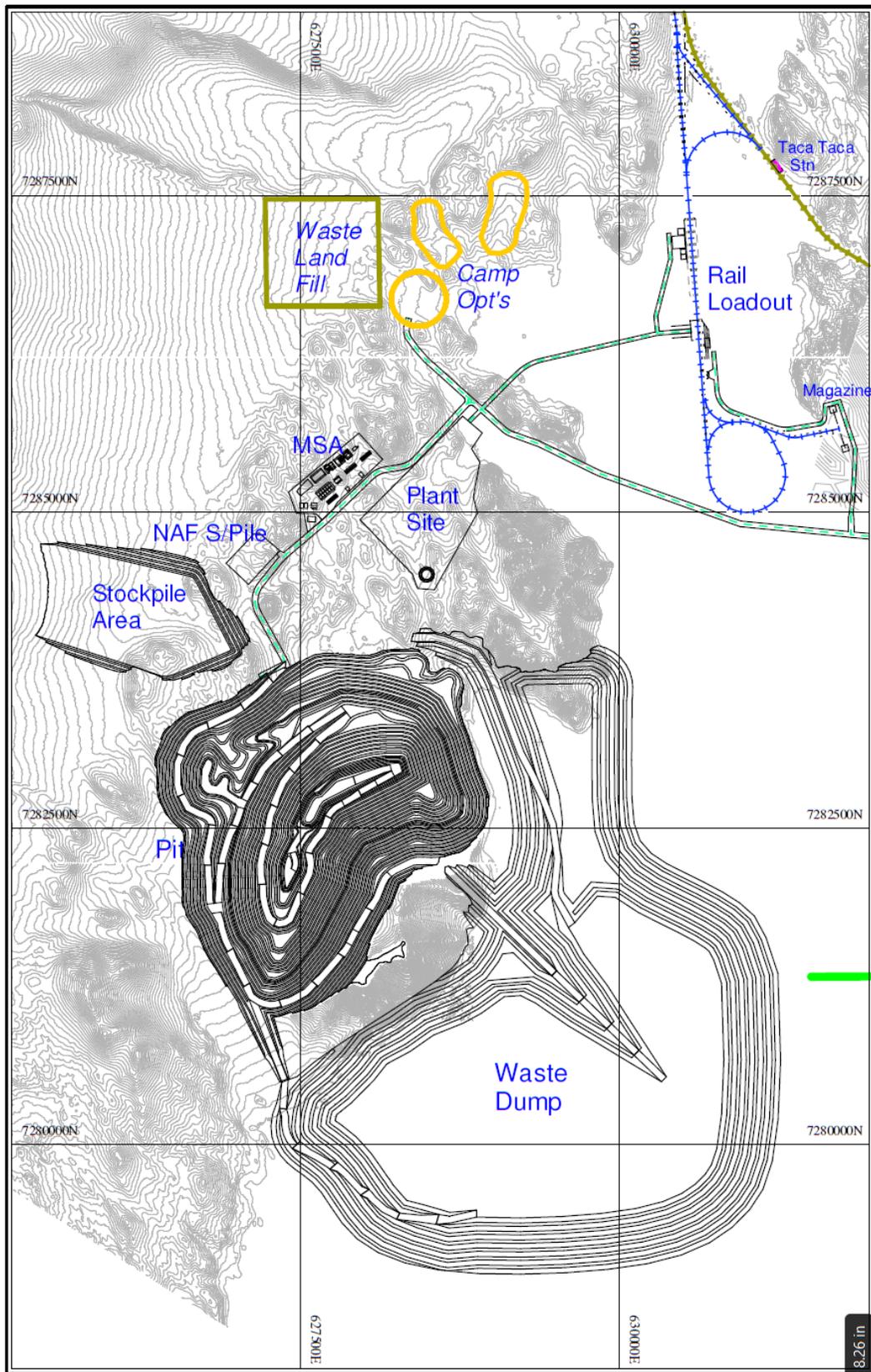
Figure 16-10 Perspective view of the Taca Taca ultimate pit



16.6 Waste dump design

The specific waste dump design parameters are provided in Item 16.2.6. Figure 16-11 shows the location and ultimate design for the waste dump, located on the Salar de Arizaro, immediately to the east of the pit area.

Figure 16-11 Waste dump and stockpile locations



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Waste rock generated from mining the pits would be stored within a single large dump located to the east of the pit area. Excluding auriferous material mined as waste, but including marginal ore, the total mined quantity is expected to be approximately 3,056 Mt and will require a dump landform with a capacity of about 1,533 Mm³ (assuming an insitu to placed bulking factor of 1.3). The ultimate height of the dump is over 200 m.

This capacity will be sufficient for storage of NAF and PAF waste (1,369 Mm³), separately from marginal ore (164 Mm³). The marginal ore could be stored at the northern end of the dump such that it could be readily reclaimed to the processing plant in the final years of the Project.

The overall final slope angle for the dump would be <25°. The dump would be constructed from 20 m high dump lifts with 36 m wide berms constructed at 20 m vertical intervals, generally progressing from west to east (i.e., progressing towards the salar). The design final slope angle for the dump should be sufficiently flat to contain, within the cross-sectional profile of the slope, any slumping which may arise due to compaction of the underlying salar surface.

Figure 16-11 also shows a small area set aside for stockpiling NAF waste, on the north western side of the pit.

16.7 Ore stockpile design

Figure 16-11 shows a surface stockpile site provided for active ore stockpiling/reclaim. The maximum size of these ore stockpiles is up to 13.8 Mt (In Years -1 and again in Year 7) or 6.9 Mm³. The design stockpile has a capacity of 42.5 Mm³, and can therefore also accommodate the proposed separate storage of approximately 55 Mt of auriferous material mined from the leach cap and not in the current Mineral Reserve.

The ore stockpile design parameters are the same as those provided in Item 16.2.6 for swelled waste rock.

16.8 Project site layout

Figure 16-12 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.

16.9 Mineral Reserve statement

The Mineral Reserve estimate provided in Table 15-10 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 16-10 accounts for mining dilution and recovery.

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 in the order of 0.13% Cu_{eq}, based on long term metal price projections of \$3.00/lb Cu, \$12.00/lb Mo and \$1,200/oz Au. An elevated cut-off grade of 0.20% Cu_{eq} was used to determine the plant feed inventory for the production schedule.

Figure 16-12 Layout of the proposed Project site

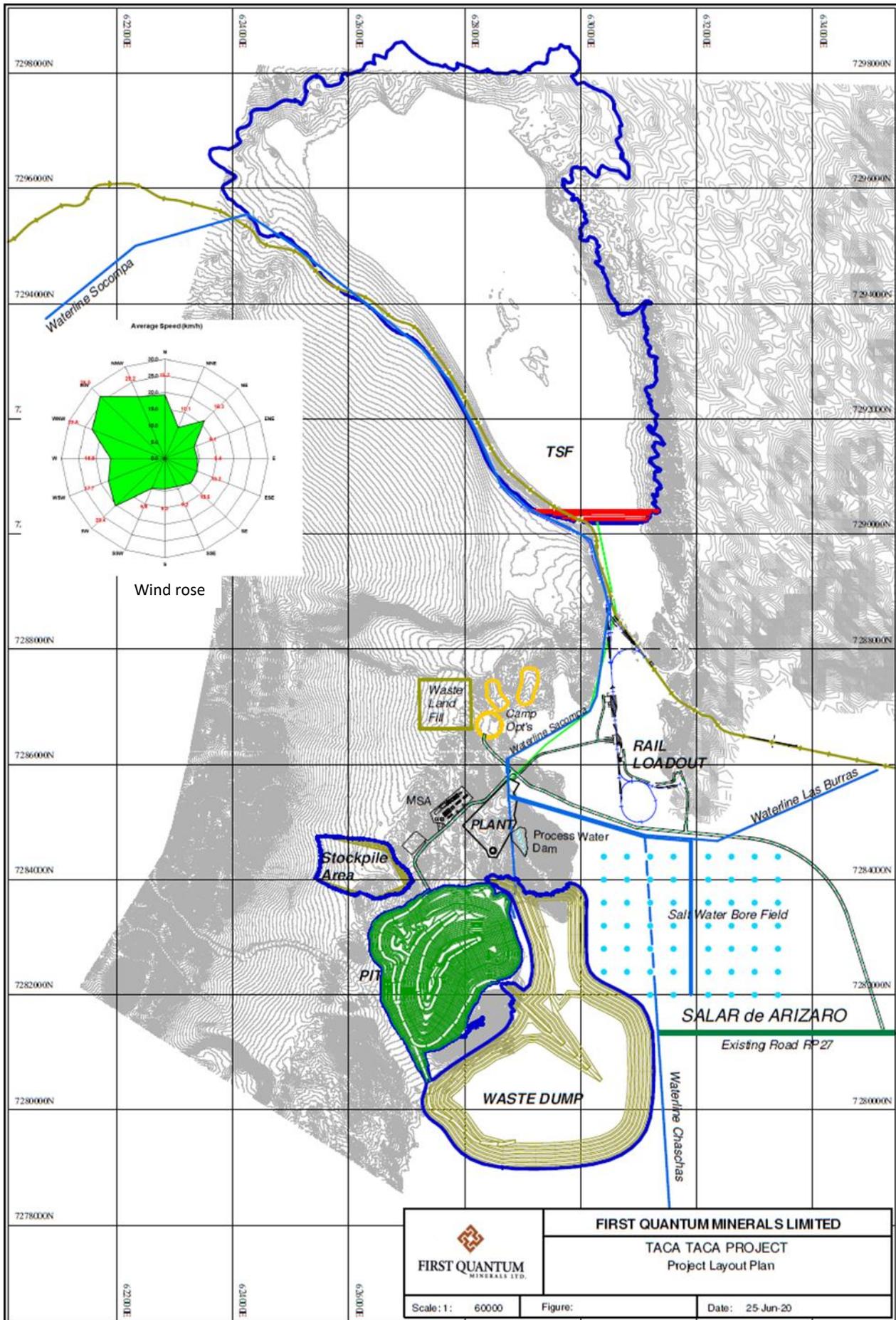


Table 16-10 Taca Taca Mineral Reserve estimate, at October 2020

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

ITEM 17 MINING METHODS

17.1 Mining method and description

The deposit 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.

17.1.1 Open pit mining equipment

The Taca Taca pit would be mined using conventional open pit methods involving blasthole drills, diesel hydraulic excavators, electric shovels and off-highway haul trucks.

17.1.2 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. With increasing mining depth, production drilling and blasting would take place in rock conditions requiring a range of drilling/charging patterns and powder factors.

At this time, there is inadequate geological definition to be able to develop comprehensive drill and blast designs, therefore indicative geological properties have been used to devise generic production and wall control blasting requirements (and cost estimates) based on the following criteria:

- rock properties:
 - density = 2.65 g/cm³
 - unconfined compressive strength = 150 MPa
 - Young's Modulus = 54 GPa
 - Poisson's ratio = 0.25
- 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 251 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 110%⁸
- a packaged explosive with a diameter of 30 mm and a density of 1.18 g/cm³
- a waste fragmentation target with an 80% passing size of 350 mm, which should be suitable for efficient excavation using large shovels and excavators with a bucket size greater than 30 m³

⁸ Relative weight strength compared to an equal mass of Ammonium Nitrate/Fuel Oil (ANFO) at 0.80 g/cm³ and an effective energy of 2.30 MJ/kg.

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- an ore fragmentation target with a 99.95% passing size of 800 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

As shown in Table 17-1, the resulting design parameters produce powder factors for the fresh rock of between 0.65 kg/m³ and 1.21 kg/m³, depending on the blasting application.

Table 17-1 Drilling and blasting parameters

INPUTS				
Rock Type		All	Waste	Ore
Blast Type		Trim	Production	Production
% passing	%	80	80	99.95
Passing size	mm	350	350	800
Hole diameter	mm	251	311	270
Bench height	m	15.0	15.0	15.0
Explosive density	g/cm ³	1.20	1.20	1.20
Weight strength	%	110	110	110
RESULTS				
Indicative Drill Type		PV271	PV351	PV271
Subdrill	m	0.0	3.0	2.5
Burden	m	7.0	9.0	6.3
Spacing	m	8.1	10.4	7.2
Burden x Spacing	m ²	57	94	45
Rock volume	m ³	851	1,404	680
Stemming length	m	5.5	6.4	5.5
Charge length	m	9.5	11.6	12.0
Charge weight	kg	564	1,057	824
Powder Factor	kg/m ³	0.66	0.75	1.21

It is anticipated that controlled blasting techniques would be required 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.

17.1.3 Loading and hauling

A fleet of Company owned and operated primary mining equipment would be used for loading and hauling of plant feed to a ROM pad crusher at the processing plant⁹, and for waste to external dumps, and ore to interim stockpiles. It is expected that on-highway trucks would be used to haul material to the TSF starter embankment as required.

The primary loading and hauling equipment is expected to be electric powered shovels with approximately 90 t bucket capacity, matched with 360 t capacity haul trucks. A large excavator (800 t mass) would supplement the shovel fleet in smaller working areas to maintain efficient production. Front end loaders would be used for clean-up and stockpile rehandle as required.

⁹ As the Project engineering phase proceeds, the implementation of in-pit crushing and conveying (IPCC) technology will be considered.

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A diesel powered excavator (350 t mass) and small haul trucks (85 t capacity) would be used to perform pioneering work, develop haul ramps for the shovels and other development activities as required.

Depending on excavation conditions and horizons, bench heights and mining lateral dimensions may vary in different areas of the open pit.

17.1.4 Trolley-assisted haulage

An improved haulage efficiency measure can be adopted, where applicable, using trolley assisted trucks for ore and waste haulage. Haulage cycle times can be significantly reduced for trucks receiving direct electric power to their wheel motors.

The geometry of the mineralisation and the consequent shape of the pit designs do not result in long straight haul ramps that are typically adopted for trolley assisted haulage, until approximately eight years into the Project life. Nevertheless, trolley assist concepts have been assumed (and accounted for in the mining cost estimates), and the widths of the ultimate pit haul ramps designed accordingly. 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 55 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 phase pit designs, even with curved haulage segments. This aspect is to be further evaluated as the Project engineering phase proceeds.

17.1.5 In-pit ore crushing and conveying

Also subject to further work during the engineering phase, there is the possibility that in-pit crushing and conveying (IPCC) of ore could be adopted. Several of the existing Company open pit mining projects are making use of (or are currently installing) IPCC of mined ore direct to stockpile(s) at the processing plant.

Preliminary mining studies for the Taca Taca Project indicate that IPCC could have limited applicability due to the geometry of the orebody (specifically the conical shape and significant depth) and the configuration of the cutback phases. Whilst production schedule and cashflow scenarios have been run with and without IPCC, the relative economics are not readily distinguishable at this stage of Project engineering.

For this Technical Report, the ultimate and phased pit designs plus the associated production schedule, assume no IPCC.

17.1.6 Waste dumping

Waste rock generated from mining the pit phases would be stored primarily within a single large dump located on the Salar de Arizaro, to the east of the pit area.

The waste dump would ultimately be operated in accordance with industry best practice and, where possible, would be formed to a long term stable angle as mining activities proceed. The overall final slope angle for the dumps would be <25°. Notwithstanding this, is the potential for compaction of the underlying salar surface leading to slumping, the impact and proposed management of which is described below.

Dump slope stability

In the selection of the preferred site for the waste dump, the following was taken into account:

- the dump is located on the Salar de Arizaro, well 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°; very flat and not atypical for a waste dump design

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- 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 16.2.3):
 - the ultimate dump height is in the order of 200 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 so as 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.

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.

Investigation into the acid generating potential of waste rock will be further considered as the Project engineering phase proceeds and may guide the detailed development and design of the waste rock dumps¹⁰ NAF material is approximately 94 Mbcm (9% of the total waste volumes) and this material would be mainly used for TSF embankment construction and for surfacing the waste dump.

The same generalisations as above also apply to the long term marginal ore stockpile. Subject to further investigation and if necessary, the top of the waste dump and the stockpile could each be contoured to prevent water from collecting on the surface and forming ponds. Any slope run-off would be intercepted by perimeter drains which (if appropriate) would direct water to suitably sized settling ponds where water pH could be monitored and treated.

¹⁰ Indications are that despite an ABA classification of PAF waste, the prevailing climatic conditions are such that there is an expectation of minimal run-off from the dump.

Mine water management

Further to the above, M&A (November 2018) outlined the basis for a water management plan to prevent, mitigate and control any potential impacts on surface and groundwater that the Project activities could generate. In particular, the plan addresses “contact” water and “non-contact”¹¹ water and the minimisation of any interaction between contact water and other natural fresh water bodies whether at surface or in groundwater.

In terms of surface water controls, a detailed plan is yet to be formalised, however, the major components for mine site water management could include (M&A, November 2018):

- diversion channels and collection ponds to intercept clean water inflows to the pit areas, and other key collection points
- collection channels for intercepting runoff from disturbed areas
- sediment ponds to settle out suspended solids
- pumping facilities for taking the collected water to other facilities, such as the processing plant

17.1.7 Ore stockpiling

Active stockpiling and reclaim of plant feed has been minimised as far as possible, in preference for maximising mine direct feed to the primary crushers. There is a stockpile site provided on surface for active stockpiling, and this site is also intended for the stockpiling of auriferous material sourced from the near-surface leach cap.

The extent of active stockpile movements over the life of mine is indicated in graphs provided in Item 16.3. Item 16.3 also lists the inventory of marginal ore that is mined onto a longer term stockpile for eventual reclaim in the final years of the Project. This stockpile will be accommodated in the northern end of the waste dump.

17.1.8 Pre-mining activities

As part of the construction phase, initial mining areas will have been stripped of thin topsoil 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 definition), geological mapping, construction of temporary access ramps, extension of services to drills and shovels, and digging of drainage ditches and water sumps.

17.1.9 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.

¹¹ Contact water is water that has been contaminated through interaction with mining and/or processing activities.

17.2 Mine planning parameters

17.2.1 General design parameters

The design of the open pit slopes has followed the geotechnical specification for the inter ramp slope parameters stated in Table 17-2, with batter, bench and haul ramp design parameters typically as follows:

- mining batter height of 15 metres
- bench heights = 30 metres
- batter angles to conform with the overall slope criteria
- haul ramp width (including ditch and safety berm) of up to 55 metres
- haul ramp gradient of 10%

Table 17-2 Pit slope design parameters (Ausenco, 2013)

Design Parameter	Slope Sector					
	1 N	2 NE	3 E Central	4 WE	5 SW	6 NW
Inter ramp angle (degrees)	43.5	48.5	48.0	50.0	48.5	48.6
Bench face angle (degrees)	55.0	62.0	61.0	64.0	62.0	62.0
Catch bench vertical interval (m)	30.0	30.0	30.0	30.0	30.0	30.0
Catch bench width (m)	10.6	10.6	10.4	10.5	10.6	10.6

Following a geotechnical update review in 2016, the North East (Sector 2) slope design recommendations were modified (steepened) (refer to Item 16.2.3).

17.2.2 Proximity to the Salar de Arizaro

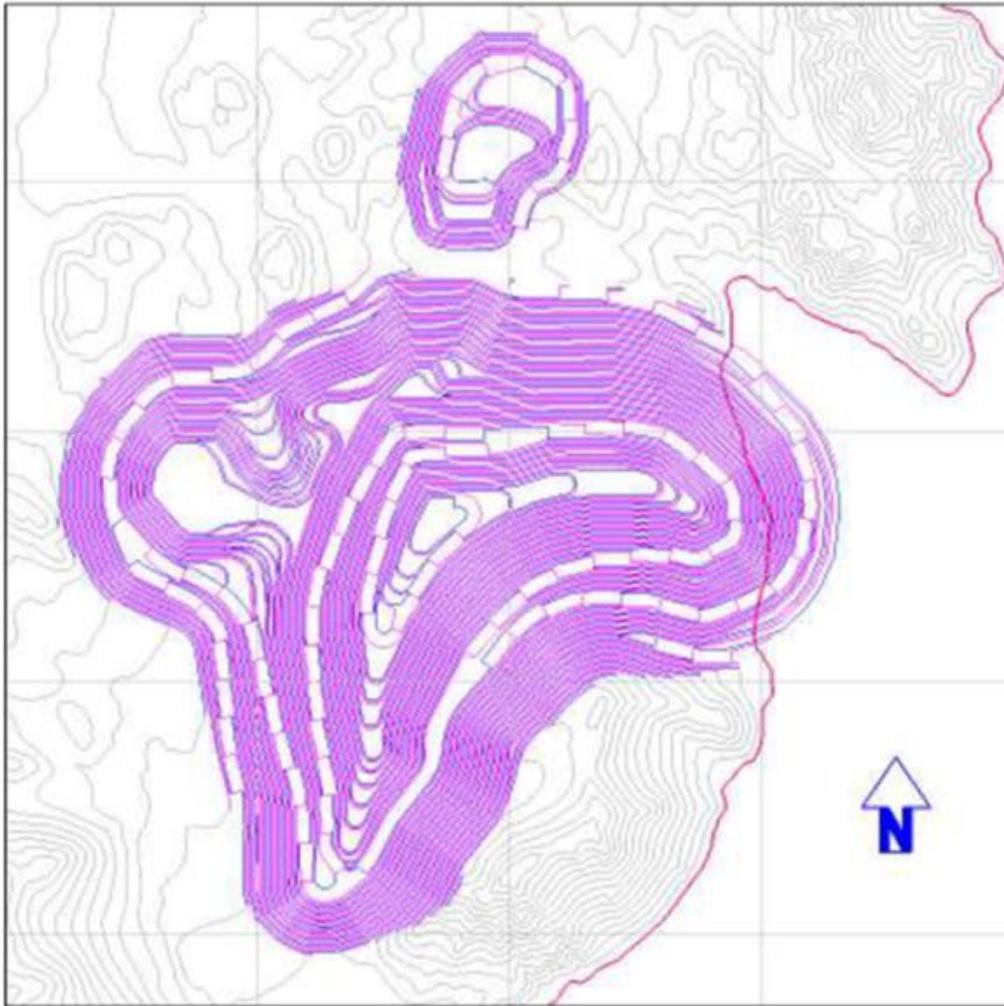
The previous Project owners described their process for optimising and designing the Taca Taca ultimate open pit (Ausenco, 2013). Figure 17-1 shows the ultimate pit design from the Lumina PEA report (Ausenco, 2013) and the lateral extent of the pit limits¹².

The first phase design of the Lumina proposed pit extended about 120 m onto the salar in Year 2, whilst the ultimate phase limits projected a further 200 m onto the salar by Year 18. These extensions onto the salar were caused to some extent by the incorporation of several haul ramp passes into the North East sector of the pit phases.

The PEA report described a deep excavated trench that would need to be mined into the salar to the east of the Lumina design pit, within which would be constructed a clay core bund. The purpose of this trench and bund would be to cut-off the inflow of saline water to the pit.

¹² Figure 16-1 shows a satellite pit immediately north of the Lumina design pit. This is referred to as the “Little Taca Pit”. This part of the overall Taca Taca resource is poorly defined by drilling coverage, and owing to subsequent optimisation by the Company, is not currently part of the Mineral Reserve inventory.

Figure 17-1 Ultimate pit shell for the PEA report (Ausenco, 2013)



The Company reviewed the PEA design and completed another geotechnical review, re-optimisation and a cashflow analysis for an alternative pit design. The geotechnical component of this work produced a revised (steeper) set of slope design parameters for the eastern wall to avoid encroachment onto the salar, with further detail on this work being provided in Item 16.2.3. The resource model was then re-optimised to avoid the pit shell encroaching onto the salar, thus avoiding the requirement for the deep trench and clay core bund to stem salar inflows. Information on the outcome of this work is provided in Item 15.3.

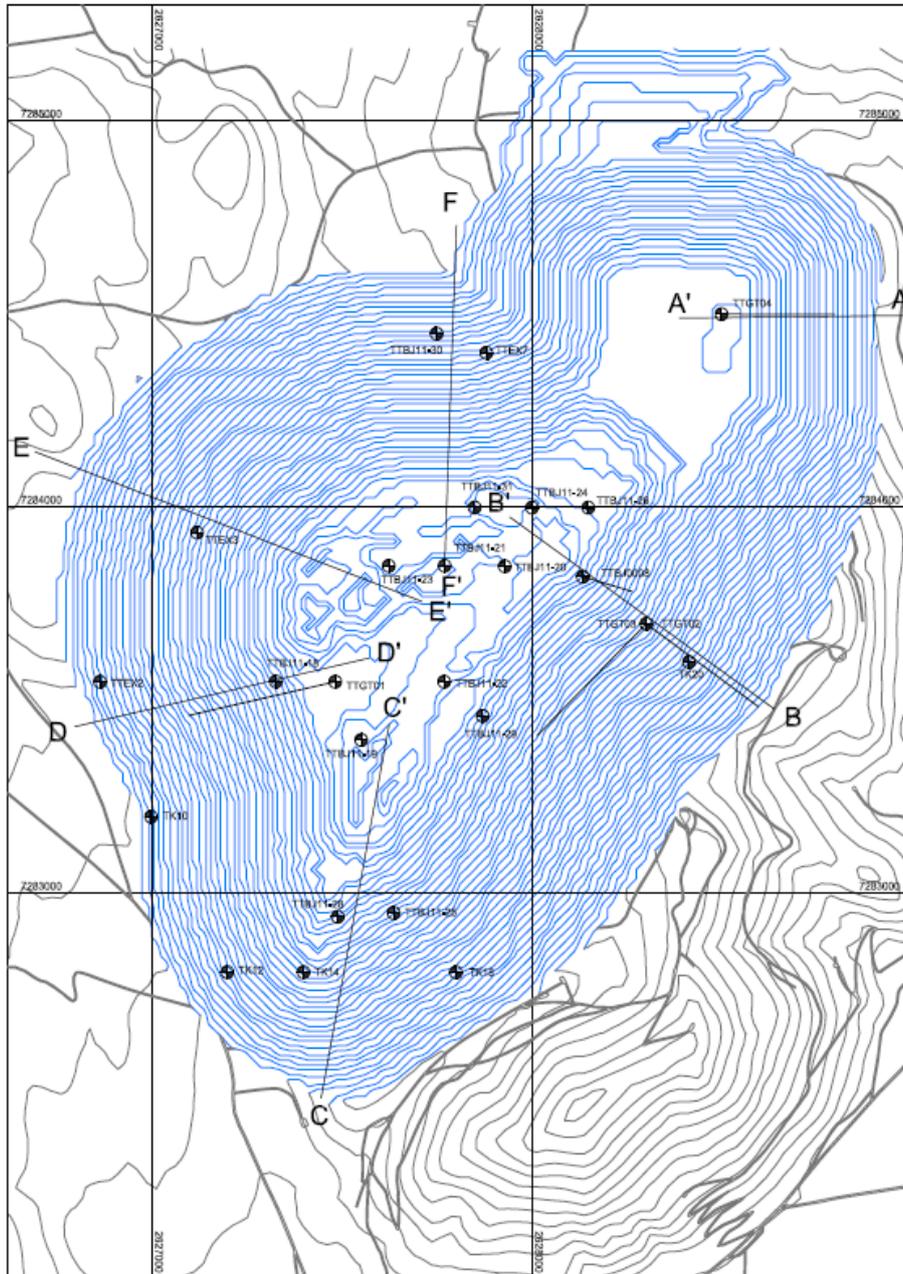
17.2.3 Geotechnical parameters

Mine geotechnical engineering

Geotechnical consultants Wyllie & Norrish (W&N) completed a geotechnical assessment and report for Taca Taca in January 2012 (W&N, 2012). This assessment involved:

- the relogging of 900 m of drill core recovered from six diamond holes drilled in early exploration programmes (five of which were vertical holes; refer to Figure 17-2)
- logging of 4,350 m of drill core recovered from eight diamond holes drilled by Lumina in 2011 (four of which were specifically for mine geotechnical purposes; two were vertical and six were inclined; refer to Figure 17-2)
- field point load testing, uniaxial compressive strength (UCS) and direct shear testing on intact (pre-split) Lumina core

Figure 17-2 Plan showing number of and location of drill holes used for mine geotechnical engineering



W&N completed limit equilibrium slope stability analyses to assess the combined impact of pit slope orientation, rock mass quality, and potential blast damage to pit walls, and also due to pore pressure influence. Presumably, due to the predominance of vertical drill holes and the paucity of oriented core measurements, the analyses did not consider the impact of structures such as faults, veins, joints, foliation etc. The leach cap and sulphide horizons were considered separately and the slope design recommendations were said to be conservative and suitable for prefeasibility stage planning (W&N, 2012). The recommendations were provided assuming that the leach cap (more permeable than the sulphide horizon) would be free-draining and that the underlying sulphide horizon would be depressurised by vertical perimeter bores and/or horizontal drain holes.

In the Lumina (Ausenco, 2013) pit design, the overall slope angle (which included haulage ramps) on the eastern side of the pit, where it encroaches onto the salar, was approximately 43°. This resulted in a significant increase in the distance that the proposed pit encroaches onto the salar and adds additional waste material that is required to be excavated.

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In order to determine the viability of other options for the 'E Central' slope sector, W&N completed for FQM, a reassessment (W&N, 2016) of their previous work, specifically looking at the steepest overall slope angle that could be adopted for the eastern wall, without encroaching onto the salar surface.

The conclusions from the W&N reassessment of the eastern pit slope are:

1. It is preferred to avoid encroaching onto the salar to avoid the cost and geotechnical/engineering risks associated with the proposed bund/clay core concept.
2. If the eastern pit wall was to daylight through the salar, the pit slopes could experience infinite recharge.
3. There are two main modifications that could be made to the eastern wall of the pit design to reduce the pit encroachment into the salar:
 - a) Remove haul ramps from the eastern wall, such that the inter ramp angle would equal the overall slope angle (48°).
 - b) The overall slope angle could be increased to 55° in the same area, provided that support and/or catch measures (e.g. cable bolting, rock catch fences and the like) are put-in-place to manage potential wedge instability.

In view of the above conclusions, subsequent FQM open pit optimisation placed a constraint on the eastern side of the deposit to prevent the Whittle algorithm from producing shells which encroached onto the salar surface. The optimisation approach to compensate for the potentially "lost" open pit resource on the eastern side of the deposit is discussed in Item 15.3.

The revised geotechnical parameters (i.e. overall slope angles) adopted following the W&N (2016) reassessment are shown with the corresponding design sectors, in Figure 17-3.

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 17-3 shows that the bearing capacity was determined as a range from 3.9 kg/cm² (382 kPa) to 10.9 kg/cm² (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.

Figure 17-3 Revised pit slope design parameters

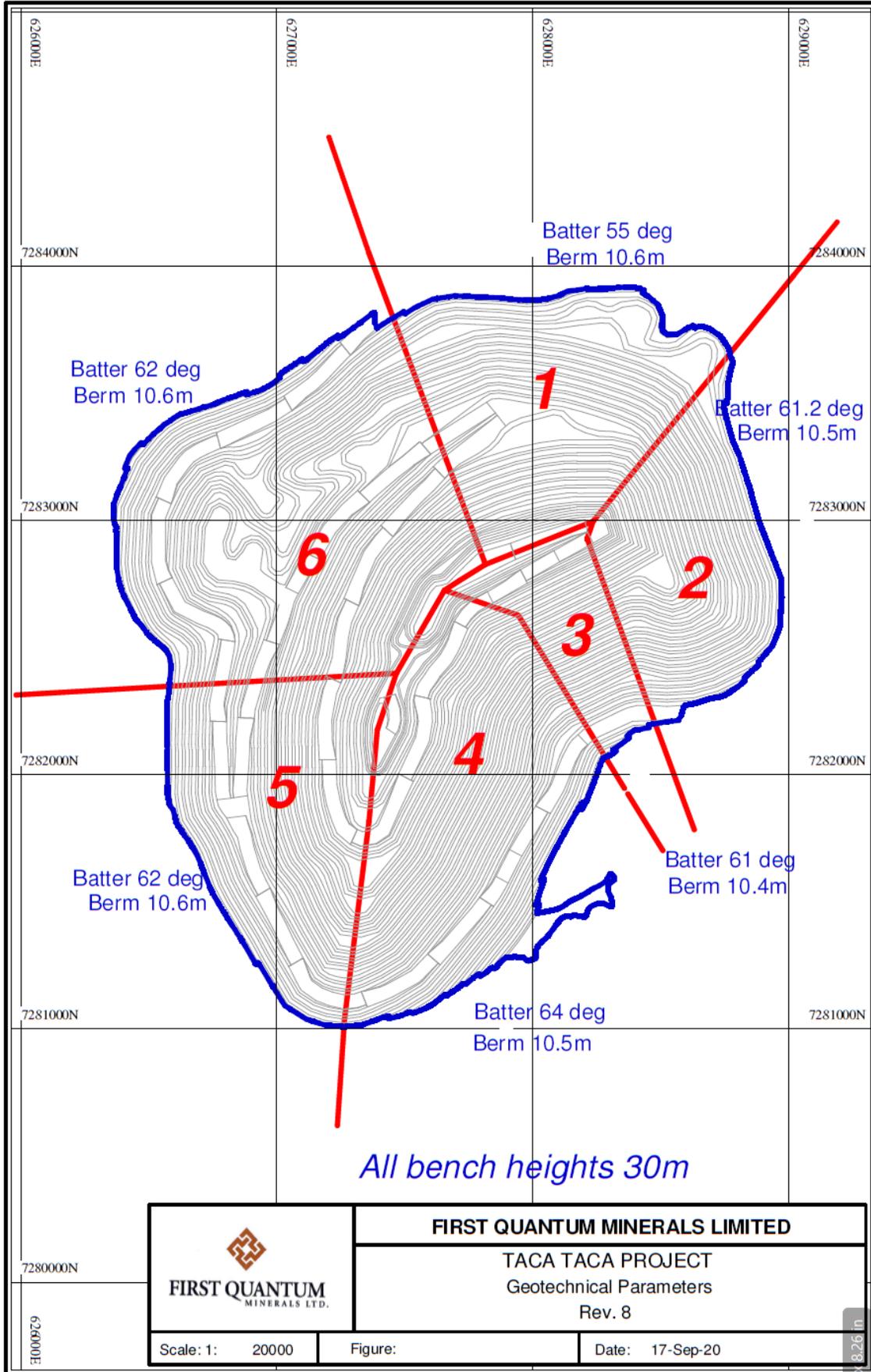


Table 17-3 Geotechnical analysis for the waste dump

	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

The waste dump will be significantly higher than the limiting heights indicated in Table 17-3, hence it is possible that the salar surface could compact/heave in places, leading to localised slumping of the dump outer slope. GP concludes that the subsidence would not be a sudden occurrence, due to the progressive heightening of the dump over time. For such relatively flat ultimate dump slope angles, the slumping could be managed.

17.2.4 Open pit inflow and pit slope depressurisation

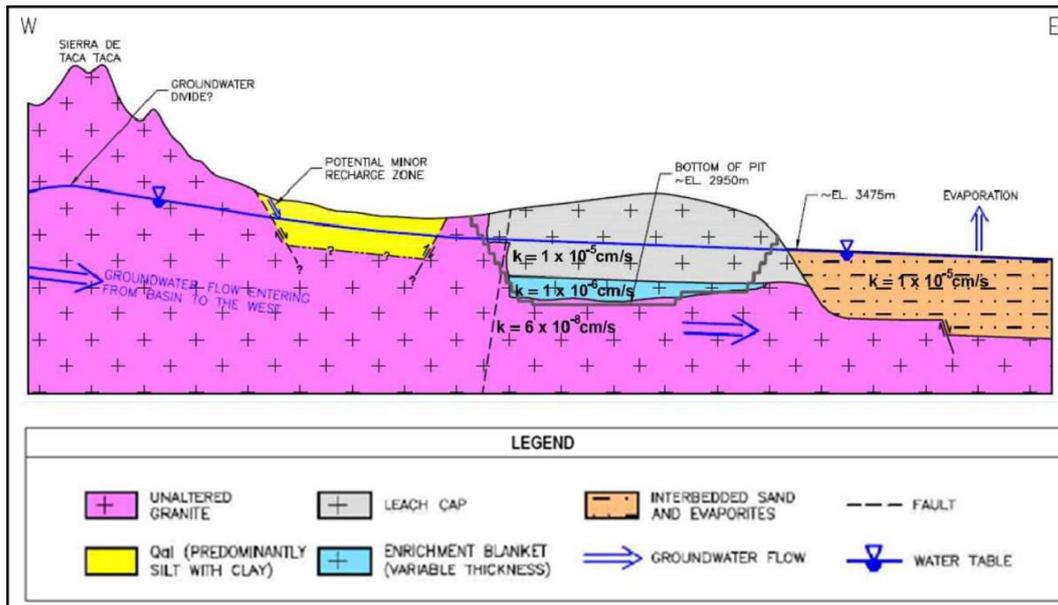
In terms of water inflow to mine workings, an evaluation of the effects of faulting and rock mass alteration on rock mass properties has not been completed. However, localised packer testing at the Project site indicates that permeability across fractures often results in higher hydraulic conductivity 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).

Ausenco (2011) produced a hydrogeological model to assess the groundwater response to open pit mining drawdown. Figure 17-4 shows a hydrogeological cross section, west to east, through the proposed mining area, as modelled.

Ausenco modelled a flow system characterised by:

- groundwater flowing from west to east, from recharge sources in the distant mountains and flow towards the low point of Salar de Arizaro
- depth to water in the pit area of 125 m to 150 m below surface
- groundwater flows through fractured rock zones
- from rock quality designation (RQD) data, a rockmass which is more fractured in the leach cap and less fractured at depth
- hydraulic conductivities determined from borehole packer testing

Figure 17-4 Hydrogeological cross section, Taca Taca Project site



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/sec to 54 L/sec
- a cone of depression arising from pit dewatering 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

Active depressurisation of pit walls was modelled by Ausenco in support of pit slope stability analyses carried out by W&N (2012). Groundwater pressure information was provided to W&N in the form of pressure grids generated specifically for cross-sections through the open pit walls. The pressure grids provided were for a natural drainage condition developed solely by excavation of the pit and also for an active depressurisation system in which dewatering wells and horizontal drains are employed.

The sulphide deposit is said to be much less permeable than the leach cap and therefore, in the absence of active slope depressurisation, water would tend to exit the pit slope well above the base of the pit, leading to the development of excess pore water pressures within the pit walls. Accordingly, the sulphide deposit was considered saturated under conditions of natural drainage (W&N, 2012). Further analysis by W&N 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.

In considering the east wall of the open pit and its close proximity to the Salar de Arizaro, geotechnical engineers Wyllie & Norrish (2016) were of the opinion that:

1. The sub horizontal layering within the salar suggests the probability of a series of aquifers and aquicludes. Thus, the horizontal permeability of the unconsolidated salar deposits will be greater than the vertical permeability.
2. If the eastern pit wall was to daylight through the salar, the pit slopes could experience infinite recharge.

For the Phase III water supply definition programme (Item 24), a borehole is planned for the eastern edge of the open pit at the edge of the salar, to check the inflow rates and quantify of brine that could be supplied from pit slope depressurisation bores and drains, as part of the overall brine supply.

17.2.5 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 reblocking impact of this regularisation is indicated in Table 17-4, where the inventory comparison is reported when the two models are constrained by the same indicative ultimate pit optimisation shell. The difference in contained metal is less than 1%.

Table 17-4 Impact of model reblocking

Parameter	Units	Original Resource Model	Reblocked Resource Model	Planned 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%

In the Whittle optimisation inputs, “unplanned mining dilution” of 5% (at zero grade) 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 considered reasonable for the bulk mining of a large orebody.

17.2.6 Waste dump design

The overall final slope angle for the waste dumps would be <25°. The dump(s) would be constructed from 20 m high dump lifts, with 34° angled batters, and with 36 m wide berms. Dumping will generally progress from west to east (i.e. progressing out towards the salar).

17.3 Mining and processing schedule

With the completion of the detailed ultimate and phased pit designs, detailed life of mine (LOM) production scheduling was completed using MineSched software. Scheduling assumptions included:

- minimum mining block size for phase 1,2,3 = 30 m x 30 m x 15 m (X, Y, Z); for phase 4 and ultimate pit = 45 m x 45 m x 15 m (X, Y, Z)
- mining flitch height = 15 m
- sinking rate generally a maximum of six benches (90 m) per year, with minor increases when mining small areas (e.g. at the top of the deposit)

Other scheduling constraints and strategies were as follows:

- The ore processed in Year 1 will be 30 Mt, increasing to 40 Mtpa in Year 2 to 6, then to 50 Mt in Year 7, and then to 60 Mtpa from Year 8 onwards.
- Provide a relatively consistent Cu production level, whilst maximising the annual profile over the first ten years of processing so as to compensate for the high strip ratio initial mining.
- Schedule the initial mining phases to enable ore encountered during pre-stripping to be stockpiled and reclaimed to the plant sufficient for processing at up to 40 Mtpa in the first two years.
- Marginal ore is to be stockpiled on the waste dump:

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- where possible, this material will not be direct fed or actively reclaimed to the plant until after open pit mining has been completed

The schedule level of detail is annually for all periods. There is an accompanying schedule which is detailed into quarters for the three year pre-strip period, then monthly for Years 1 and 2, then quarterly for Year 3, and annually thereafter.

Features of the LOM mining and production schedule as listed in Table 17-5 are as follows:

- Mining (i.e., starting with the pre-strip period) commences in Year -3 whilst processing commences in Year 1. The Project life (processing years) is 32 years.
- 240.1 Mt of waste is mined in the three-year pre-strip period, during which time 17.4 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 4,543.0 Mt (1,737.0 Mbcm) of which:
 - 1,758.5 Mt is ore with average grades of 0.44% Cu, 0.012% Mo and 0.09 g/t Au, and
 - 2,784.5 Mt is waste
- The overall life of mine strip ratio (waste tonnes: ore tonnes) is 1.6 : 1.
- The direct feed ore to the plant is 1,390.4 Mt at an average grade of 0.50% Cu, whilst 57.1 Mt at an average grade of 0.43% Cu is ore reclaimed from active stockpiles, and 311.0 Mt at an average grade of 0.15% Cu is ore (marginal ore) reclaimed from longer term stockpiles (mostly in Years 27 to 32).
- The total ore mined includes 39.0 Mt of ore grading 0.46% Cu from the near-surface “leached cap”, of which over 15 Mt is mined to stockpile during the pre-strip years. Most of this ore is then processed over the following three years.
- The Inferred Mineral Resource that is mined as waste amounts to 69 Mt at an average grade of 0.31% Cu (i.e., about 2.5% of the total waste mined). This material is encountered in the mining schedule after Year 6, and following completion of mining phases 1 and 2.
- The crusher feed ramps up from Year 1 at 30 Mt to 40 Mt in Year 2, at which level it remains until Year 6. The feed rate then rises to 50 Mt in Year 7, and thereafter to 60 Mtpa until Year 32.
- In terms of total plant feed (after mining dilution/recovery):
 - the average copper grade is 0.72% Cu for the first six years when processing at up to 40 Mtpa,
 - then 0.45% Cu to Year 27 when processing at up to 60 Mtpa,
 - and finally 0.15% Cu for the remaining five years of Project life when reclaiming from longer term stockpiles
- Before the final five years of marginal ore reclaim, the total plant feed is 1,476.3 Mt at an average grade of 0.50% Cu.
- The annual average copper metal production to Year 6 is 227.0 kt, and ranging between 97.5 kt and 275.2 kt. Thereafter, the annual average is 200.4 kt, and ranging between 72.5 kt and 271.1 kt (ignoring the final year of processing). In terms of life of Project totals:
 - 1,362.0 kt of copper is recovered in the first six years,
 - then 4,869.3 kt of copper to Year 27,
 - and finally 341.8 kt of copper for the remaining five years of Project life when reclaiming from longer term stockpiles

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- The annual average molybdenum metal production to Year 6 is 2,205 t, and ranging between 1,434 t and 2,912 t. Thereafter, the annual average is 2,776 t, and ranging between 1,745 t and 4,147 t (ignoring the final year of processing).
- The annual average gold recovered into concentrate to Year 6 is 106.3 koz and ranging between 90.9 koz and 134.1 koz. Thereafter, the annual average is 92.9 koz, and ranging between 45.7 koz and 156.6 koz (excluding the final year of processing).
- Of the total 6,573.1 kt of copper recovered over the Project life, only 5,335 tonnes of this (0.08% of the total) would be attributable to ore mined between Years 8 and 15, from within a mining concession having 50% ownership with a third party group.

Table 17-5 Life of mine production schedule

Year	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 (ktoz)
-3					43.5	43.5							
-2	3.6	0.22	85.80	0.10	97.1	100.6							
-1	13.9	0.39	107.20	0.12	99.5	113.4							
1	33.6	0.45	115.90	0.16	171.2	204.7	29.9	0.52	119.80	0.16	97.5	1.4	91.8
2	40.4	0.60	93.18	0.11	205.4	245.8	39.9	0.63	106.56	0.14	193.2	1.7	104.1
3	34.8	0.94	135.26	0.13	210.8	245.6	39.9	0.82	128.22	0.13	275.2	2.0	98.5
4	37.4	0.74	145.01	0.12	208.7	246.2	39.9	0.76	143.66	0.12	262.1	2.3	90.9
5	43.7	0.74	167.10	0.16	202.2	245.8	39.9	0.79	178.34	0.17	274.6	2.8	134.1
6	57.8	0.61	166.46	0.15	185.3	243.0	39.9	0.74	182.47	0.15	259.3	2.9	118.4
7	63.1	0.53	134.67	0.12	174.4	237.5	49.9	0.63	163.15	0.15	271.1	3.3	142.8
8	61.2	0.49	128.28	0.12	174.3	235.5	59.9	0.52	147.62	0.14	268.5	3.5	156.6
9	79.9	0.44	102.43	0.10	156.3	236.2	59.8	0.52	126.57	0.12	270.9	3.0	137.0
10	78.2	0.41	96.23	0.09	158.2	236.4	59.9	0.48	116.80	0.11	246.9	2.8	122.3
11	74.5	0.36	79.27	0.08	96.7	171.2	59.9	0.42	90.02	0.09	214.6	2.2	102.9
12	77.1	0.38	80.36	0.08	74.3	151.5	59.9	0.44	92.14	0.09	225.4	2.2	103.8
13	73.9	0.39	77.01	0.07	67.4	141.3	59.9	0.45	84.64	0.08	227.2	2.0	92.3
14	72.5	0.40	83.89	0.07	67.6	140.1	59.9	0.45	90.85	0.08	231.0	2.2	92.8
15	76.6	0.38	93.84	0.08	63.9	140.5	59.9	0.45	101.85	0.09	231.3	2.4	107.2
16	80.0	0.37	110.48	0.08	59.6	139.7	59.9	0.45	117.28	0.09	230.1	2.8	105.4
17	75.8	0.34	107.37	0.08	56.1	132.0	59.8	0.40	119.66	0.10	203.2	2.9	109.7
18	79.8	0.37	103.36	0.07	50.9	130.8	59.9	0.44	108.84	0.08	227.4	2.6	92.7
19	77.6	0.35	114.73	0.07	36.3	113.9	59.9	0.41	119.37	0.08	208.8	2.9	95.7
20	78.4	0.36	128.68	0.07	34.3	112.7	59.9	0.43	135.04	0.08	219.6	3.2	92.6
21	75.8	0.37	135.20	0.08	27.7	103.5	59.8	0.43	138.43	0.09	221.2	3.3	107.5
22	80.4	0.37	151.16	0.08	23.2	103.6	59.9	0.44	152.79	0.09	230.1	3.7	102.8
23	67.8	0.39	156.85	0.07	14.4	82.2	59.9	0.43	164.26	0.08	221.8	3.9	92.6
24	65.3	0.43	167.85	0.07	10.8	76.1	59.9	0.46	173.23	0.08	238.9	4.1	88.2
25	67.7	0.44	158.83	0.07	8.0	75.8	59.9	0.48	160.99	0.08	248.5	3.9	90.7
26	59.2	0.49	161.29	0.08	5.5	64.8	59.9	0.48	160.91	0.08	251.3	3.9	91.9
27	28.4	0.56	187.44	0.08	0.7	29.1	59.9	0.35	133.30	0.06	181.8	3.2	71.2
28							59.9	0.15	72.89	0.04	72.5	1.7	45.7
29							60.0	0.15	72.89	0.04	72.7	1.7	45.8
30							59.9	0.15	72.89	0.04	72.5	1.7	45.7
31							59.9	0.15	72.89	0.04	72.5	1.7	45.7
32							42.7	0.15	72.89	0.04	51.7	1.2	32.6
33													
Total	1,758.5	0.44	121.40	0.09	2,784.5	4,543.0	1,758.5	0.44	121.40	0.09	6,573.1	85.4	3,052.0
Year -3 to -1	17.4	0.36	102.82	0.12	240.1	257.5	0.0	0.00	0.00	0.00	0.0	0.0	0.0
Year 1 to 6	247.6	0.67	140.13	0.14	1,183.6	1,431.2	229.4	0.72	144.19	0.14	1,362.0	13.2	637.8
Year 7 to 27	1,493.4	0.40	118.51	0.08	1,360.8	2,854.3	1,246.9	0.45	128.19	0.09	4,869.3	63.9	2,198.6
Subtotal	1,741.1	0.44	121.59	0.09	2,544.4	4,285.5	1,476.3	0.50	130.67	0.10	6,231.3	77.2	2,836.4
Year 28 to 32	0.0	0.00	0.00	0.00	0.0	0.0	282.2	0.15	72.89	0.04	341.8	8.2	215.6
Total	1,758.5	0.44	121.40	0.09	2,784.5	4,543.0	1,758.5	0.44	121.40	0.09	6,573.1	85.4	3,052.0

17.3.1 Mining material movements and phases

Figure 17-5 accompanies the production schedule table and shows a chart of the annual ore and waste mining volumes, together with the decreasing strip ratio that follows the initial waste pre-strip.

Figure 17-6 shows a chart of the mining progression through each of the pit phases. From the information in Figure 17-5 and Figure 17-6 it can be appreciated that:

- the Domain 203 ore is mined in Phase 1a and 1
- the lesser Domain 304, 305, 310 ore is mined in Phases 2 to 4
- the larger Domain 307, 308, 309 ore is mined in Phases 2 to 5
- the primary ore in Domain 306 is mined mostly from Phases 4 and 5

Figure 17-5 Chart of scheduled mining material movements

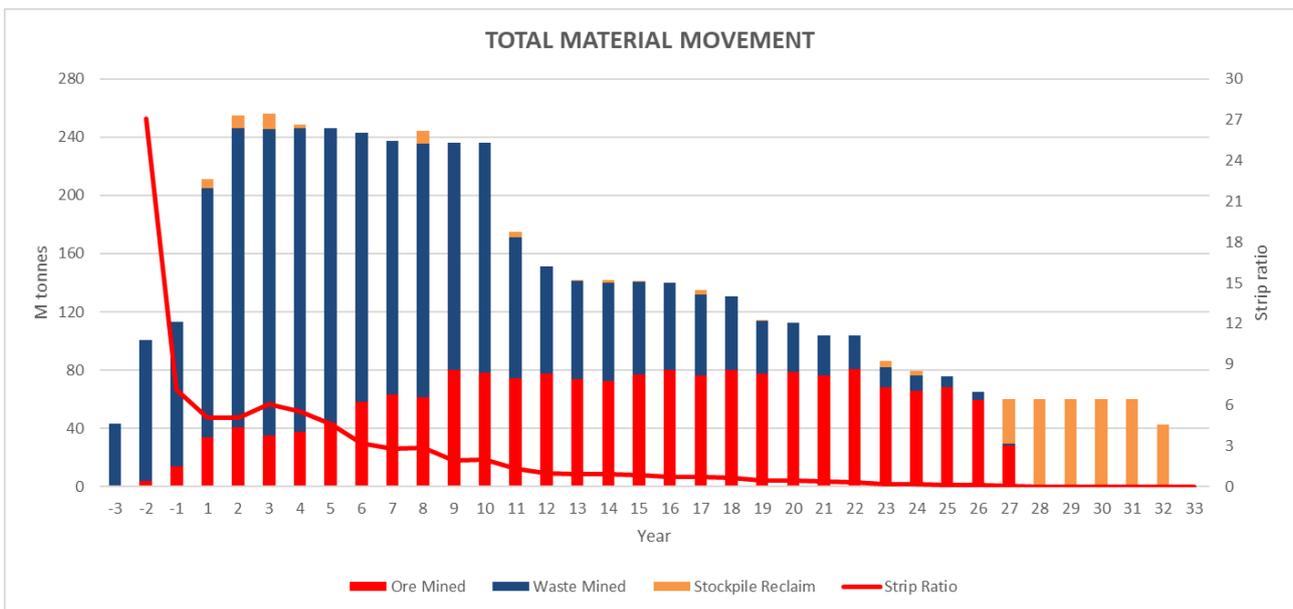
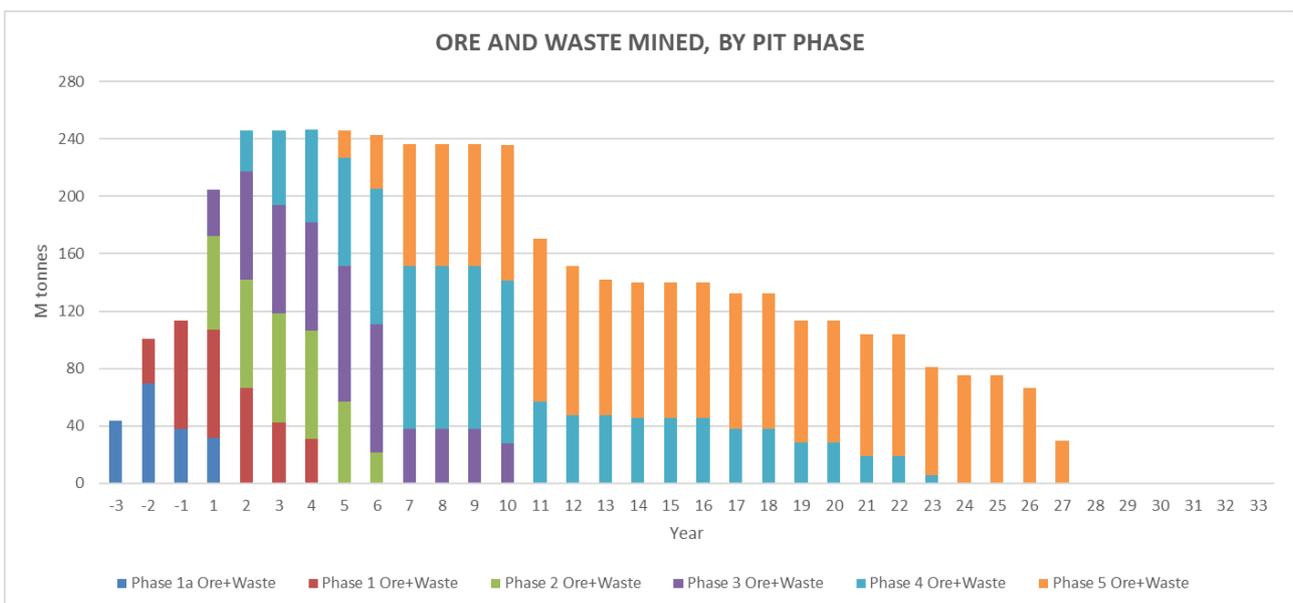


Figure 17-6 Chart of scheduled mining sequence



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Figure 17-7 shows the ore types mined according to the mineralisation domains and recovery clusters listed in Item 15, where:

- Domain 203 is the supergene mixed ore in the near-surface cap
- Domain 304, 305, 310 is the supergene secondary ore
- Domain 307, 308, 309 is the supergene mixed ore
- Domain 306 is primary ore

Figure 17-7 Chart of scheduled ore mining tonnes and Cu grade, by mineralisation domain

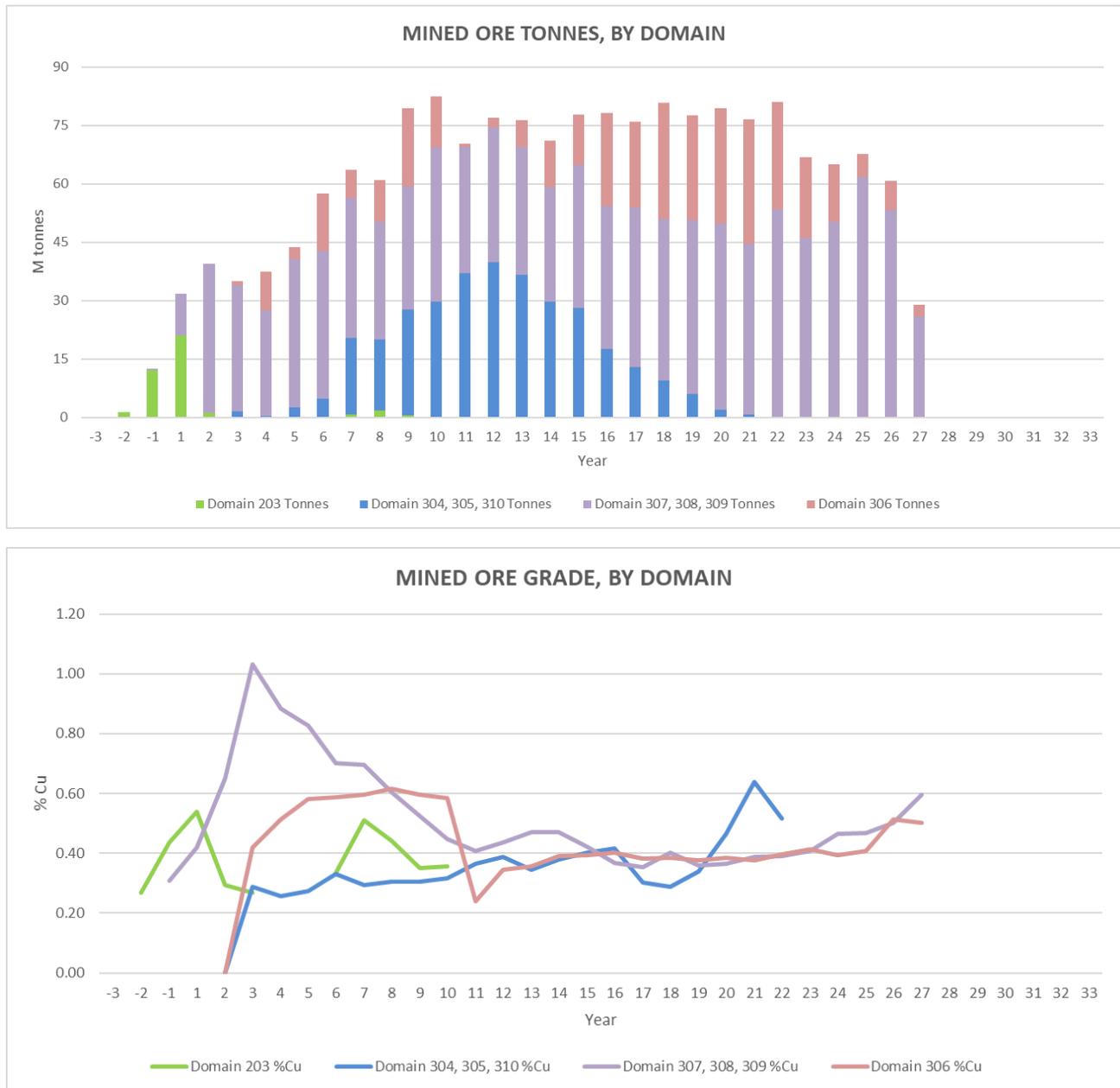


Figure 17-8 shows that the western edge of the design pit encroaches slightly onto the 50% joint venture concession '18049, Mina Francisco 2'. This encroachment amounts to approximately 1.7 Mt of ore at an average grade of 0.38% Cu (for 5,535 t of recovered copper) and approximately 47.5 Mt of waste.

The amount of recovered copper equates to about 0.08% of the total recovered. The Phase 5 limits transgress the concession boundary in Years 5 to 15 (Figure 16-9).

Figure 17-8 Taca Taca ultimate design pit, relative to concession boundary

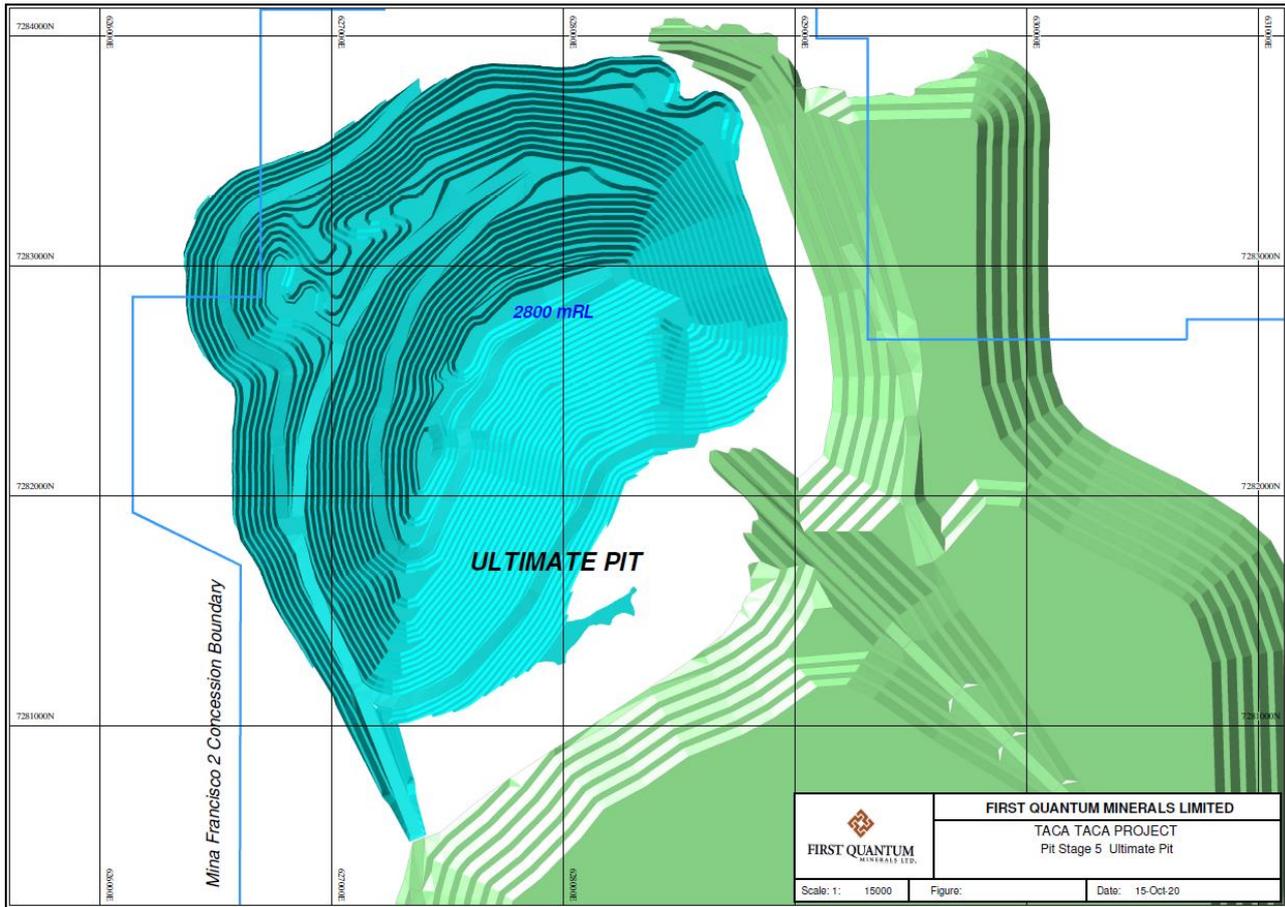
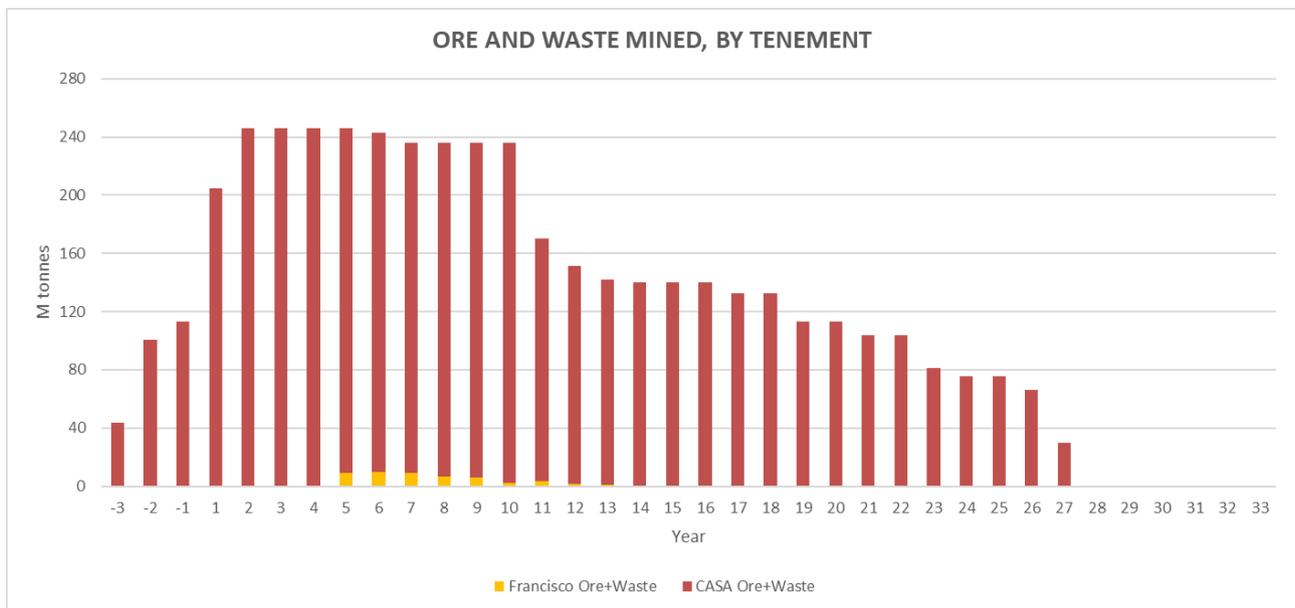


Figure 17-9 Chart of scheduled ore and waste mining tonnes, by concession (tenement)



17.3.2 Plant feed and recovered metal profile

Figure 17-10 shows the plant feed profile, with a short ramp-up in the first year of processing and a prolonged rise to the ultimate 60 Mtpa processing rate in Year 8. 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 79% to 21%. Figure 17-10 also highlights the overall feed grade trend attributable to the selection of the pit shell phases and mining sequence.

Figure 17-10 Chart of scheduled plant feed profile

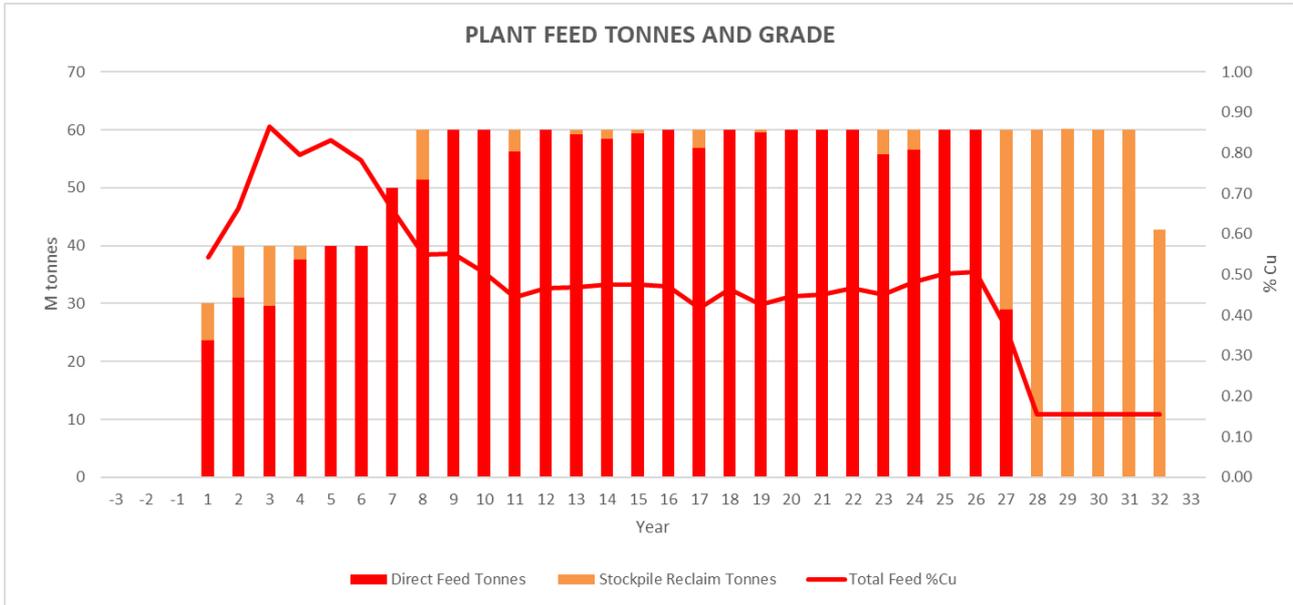


Figure 17-11 shows the ore feed types split into the metallurgical domains carried through from the Mineral Resource model. The supergene mixed ore is clearly shown for the period when direct feeding ore from the near surface Phase 1a and 1 pits. The large proportion of feed from the supergene ore sources is also shown, comprising mixed and secondary mineralisation. These would be processed in all years of the operation, supplemented with primary ore feed (Domain 306).

Figure 17-11 Chart of ore feed types to the plant

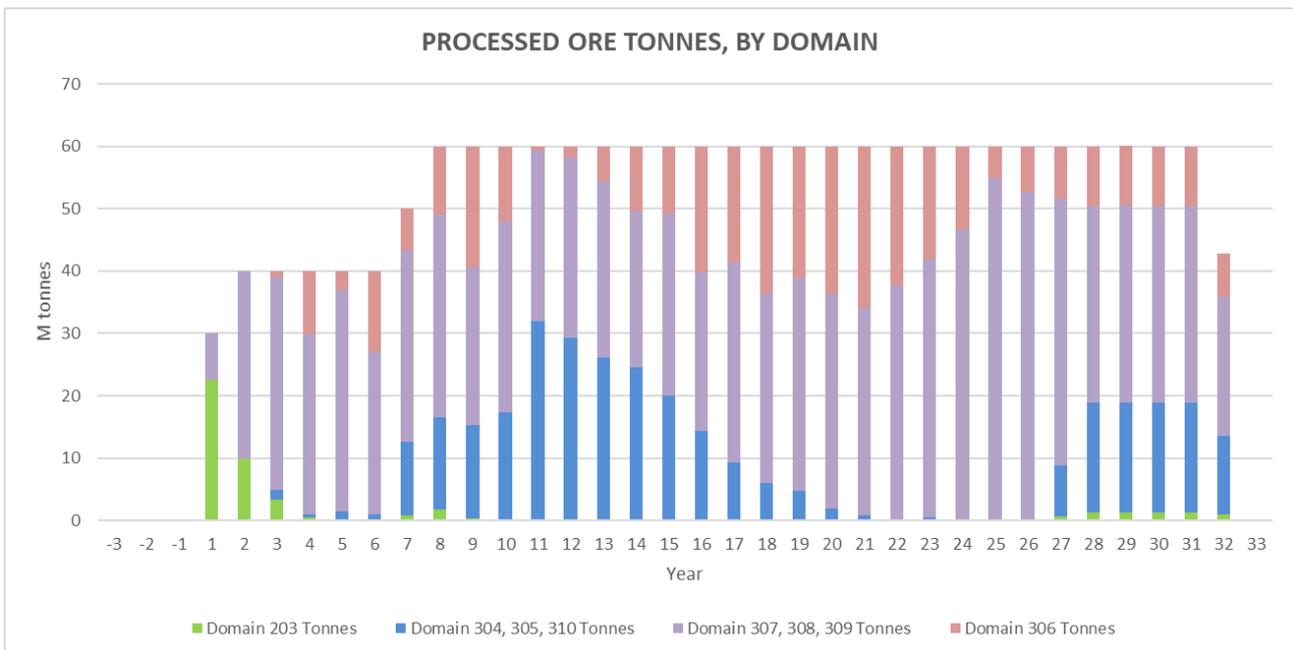


Figure 17-12 shows the annual recovered copper tonnages relative to the recovery rates carried in the model. Figure 17-13 shows the cumulative recovered copper trend; the average annual copper recovery for the first ten years of processing is 245.9 kt.

Figure 17-14 and Figure 17-15 show respectively, the recovered molybdenum metal and recovered gold (i.e., recovered into concentrate) profiles over the life of the Project.

Figure 17-12 Chart of scheduled recovered copper

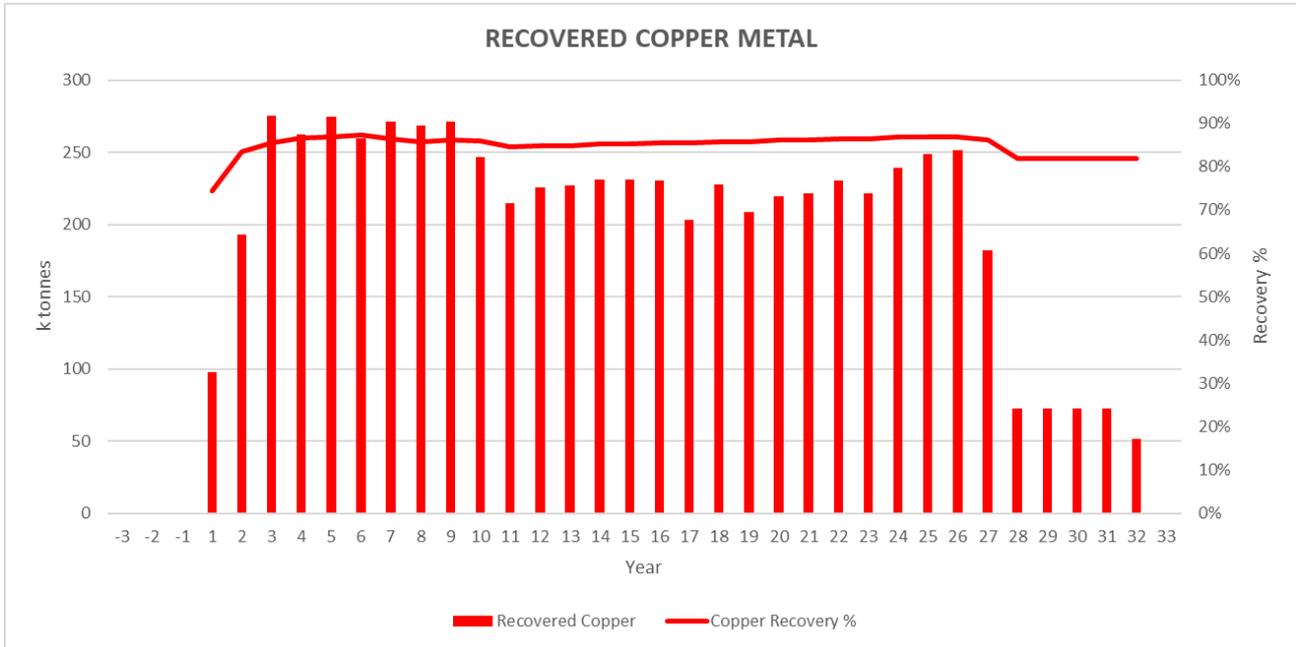


Figure 17-13 Chart of cumulative recovered copper

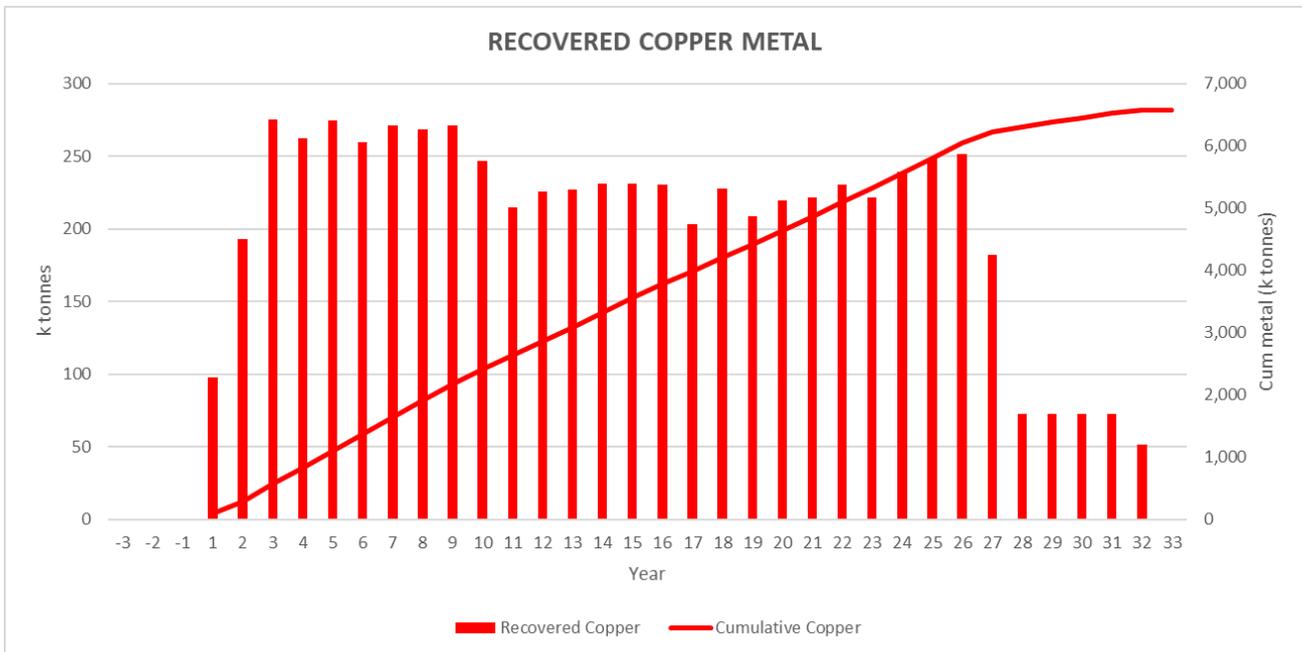


Figure 17-14 Chart of scheduled recovered molybdenum

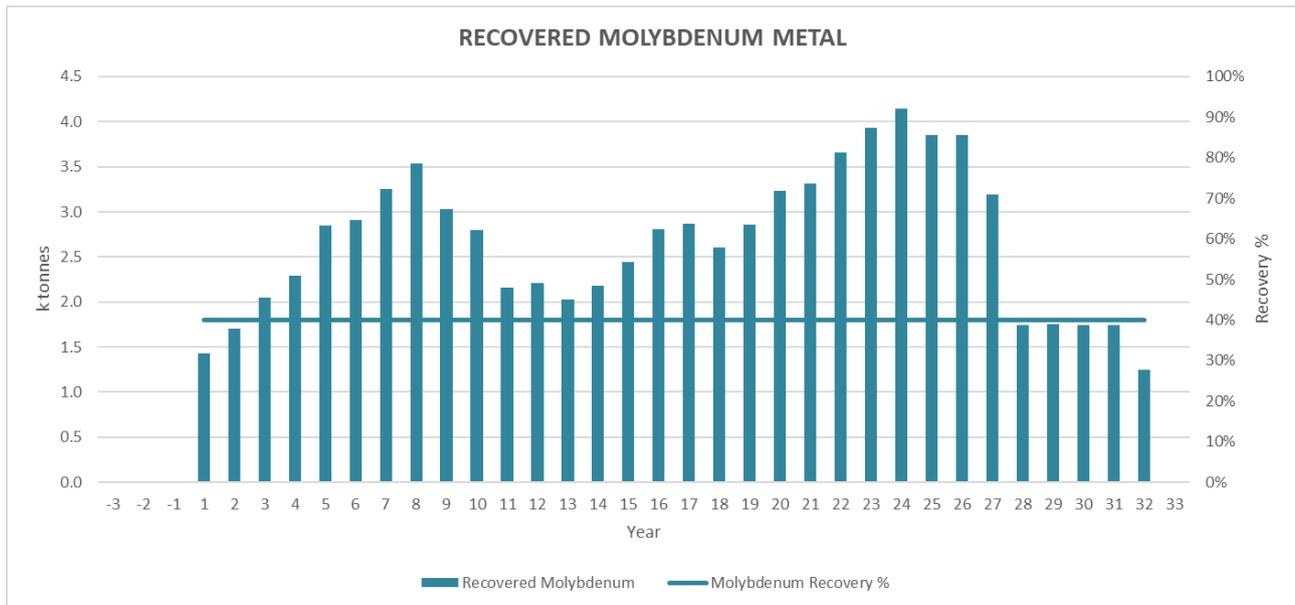
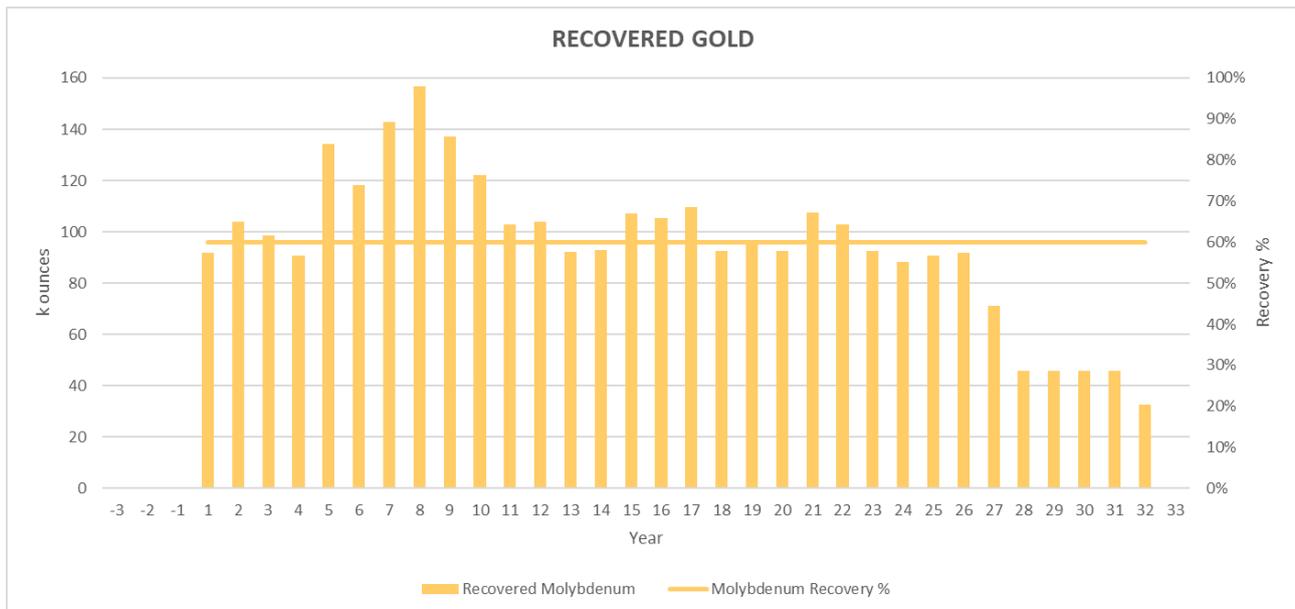


Figure 17-15 Chart of scheduled recovered gold



17.3.3 Waste dumping schedule

Figure 17-16 shows the scheduled annual volumes of waste mined, relative to ore mined, and split between PAF and NAF volumes. NAF waste is about 9% of the total waste mined and very little of it is expected to be generated after Year 14 of the Project. Table 17-6 lists the waste dumping schedule and shows the allocation of NAF and PAF waste material for haulage to the waste dump, to TSF embankment construction and for construction requirements.

The total waste tonnage shown in this table excludes the Domain 102 (gold) material that is mined as waste but dumped onto a separate stockpile.

Figure 17-16 Chart of PAF and NAF waste mined, relative to mined plant feed

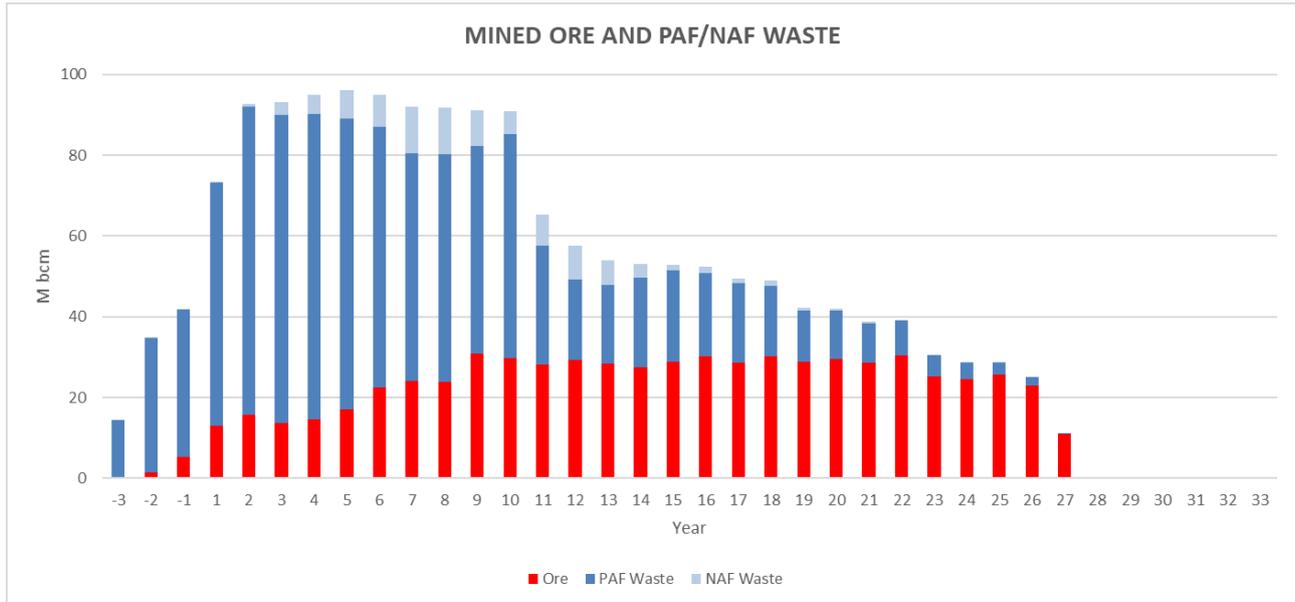


Table 17-6 Waste dumping schedule

Year	Waste Mined						NAF Allocation			PAF Allocation
	NAF (Mbcm)	NAF (Mt)	PAF (Mbcm)	PAF (Mt)	TOTAL (Mbcm)	TOTAL (Mt)	TSF (Mbcm)	Construction (Mbcm)	Dump/stockpile (Mbcm)	Dump (Mbcm)
-3	0.09	0.23	14.26	37.10	14.35	37.33	0.04	0.05		14.26
-2	0.06	0.15	33.40	87.04	33.46	87.19	0.04	0.02		33.40
-1	0.01	0.02	36.33	94.94	36.34	94.96		0.01		36.33
1	0.22	0.55	60.21	156.55	60.42	157.09			0.22	60.21
2	0.76	1.92	76.18	194.57	76.94	196.49			0.76	76.18
3	3.18	8.10	76.35	195.43	79.53	203.54			3.18	76.35
4	4.79	12.23	75.52	193.35	80.31	205.58			4.79	75.52
5	7.02	17.81	72.03	183.60	79.05	201.41			7.02	72.03
6	7.81	19.79	64.61	164.58	72.42	184.37			7.81	64.61
7	11.48	29.24	56.52	144.99	68.00	174.24			11.48	56.52
8	11.60	29.74	56.19	144.42	67.79	174.16			11.60	56.19
9	8.97	23.11	51.42	132.98	60.39	156.09			8.97	51.42
10	5.52	14.39	55.46	143.62	60.99	158.01			5.52	55.46
11	7.65	20.04	29.45	76.46	37.10	96.50			7.65	29.45
12	8.36	21.91	19.87	52.24	28.22	74.14			8.36	19.87
13	6.11	16.06	19.47	51.17	25.57	67.23			6.11	19.47
14	3.31	8.80	22.29	58.64	25.61	67.45			3.31	22.29
15	1.44	3.88	22.62	59.80	24.06	63.68			1.44	22.62
16	1.68	4.52	20.51	54.92	22.19	59.44			1.68	20.51
17	1.07	2.88	19.78	53.07	20.85	55.95			1.07	19.78
18	1.39	3.75	17.34	46.97	18.74	50.72			1.39	17.34
19	0.51	1.38	12.85	34.72	13.36	36.10			0.51	12.85
20	0.57	1.54	11.97	32.57	12.54	34.11			0.57	11.97
21	0.33	0.90	9.83	26.57	10.16	27.47			0.33	9.83
22	0.18	0.48	8.45	22.52	8.63	22.99			0.18	8.45
23	0.02	0.05	5.33	14.18	5.35	14.23			0.02	5.33
24	0.00	0.00	4.00	10.65	4.01	10.66			0.00	4.00
25	0.00	0.00	2.98	7.87	2.98	7.88			0.00	2.98
26	0.02	0.05	2.03	5.33	2.05	5.39			0.02	2.03
27	0.00	0.00	0.24	0.63	0.24	0.63			0.00	0.24
28										
29										
30										
31										
32										
33										
Total	94.14	243.51	957.50	2,481.51	1,051.64	2,725.02	0.08	0.08	93.98	957.50

As can be gleaned from Figure 17-16, there may be insufficient NAF waste available to provide an appropriate thickness of final cover on the waste dump. If required, additional NAF material could be obtained from the following:

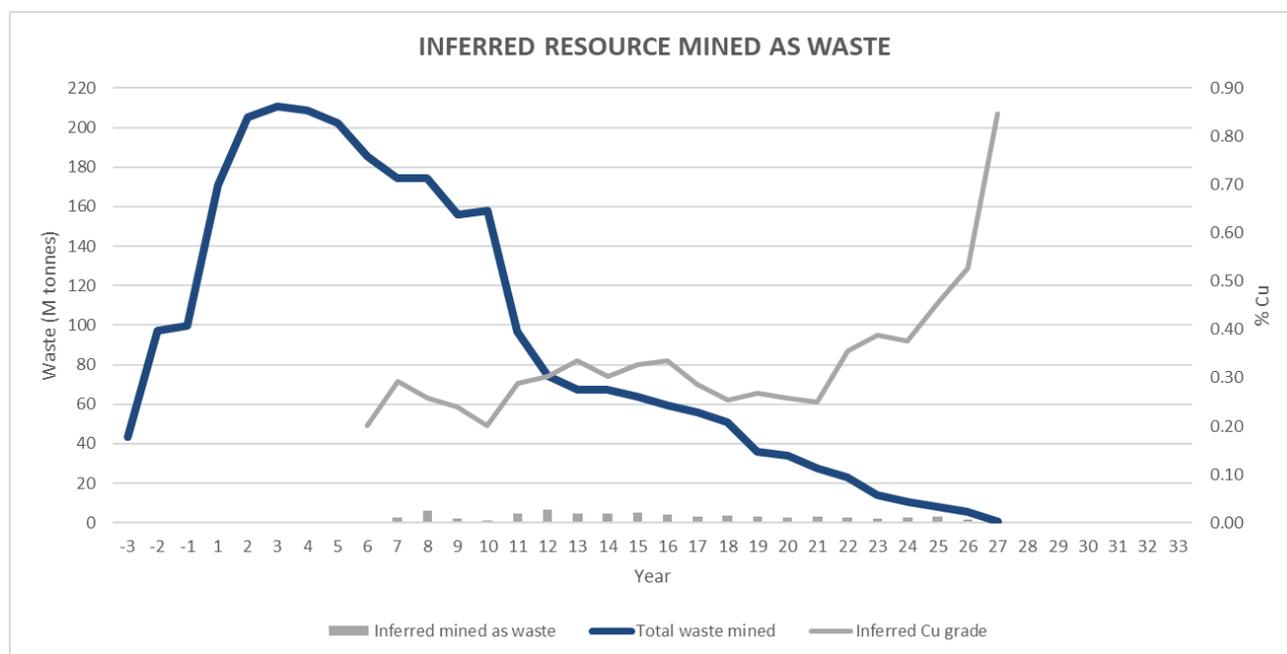
- use of locally quarried material for TSF embankment and other construction requirements, rather than using mined waste
- excavation of NAF material from the base of the proposed western stockpile site

These possibilities will be evaluated as the Project engineering phase proceeds.

17.3.4 Inferred Mineral Resource mined as waste

Relative to the total mined waste tonnage, Figure 17-17 shows the annual profile of Inferred Mineral Resource that is mined and hauled to the waste dump. The inventory amounts to 69 Mt at an average grade of 0.31% Cu, encountered in pit phases 3, 4 and 5, and between Years 6 and 27.

Figure 17-17 Chart of scheduled Inferred Mineral Resource mined as waste



Subject to confirmatory drilling and optimisation during the course of mining the initial pit phases, this Inferred Mineral Resource could be reclassified for future plant feed.

17.3.5 Ore stockpiles

Figure 17-18 shows the on/off active stockpile movements and the end of year balance tonnages.

A surface stockpile provided for active ore stockpiling/reclaim is shown in Figure 16-11. The maximum size of the ore stockpiles is up to 13.8 Mt (In Year -1 and again in Year 7) or 6.9 Mm³.

Figure 17-19 shows the on/off long term stockpile movements and the end of year balance tonnages. The marginal ore that is set aside for long term stockpiling and reclaim could be stored at the northern end of the waste dump such that it could be reclaimed to the processing plant in the final years of the Project.

The design waste dump capacity is 1,698 Mm³, which is in excess of that required. The long term stockpile reaches an eventual volume of 126.2 Mbcm (or 164 Mlcm) and the NAF and PAF waste volume to be dumped totals 1,369 Mlcm.

Figure 17-18 Active stockpile movements and balance

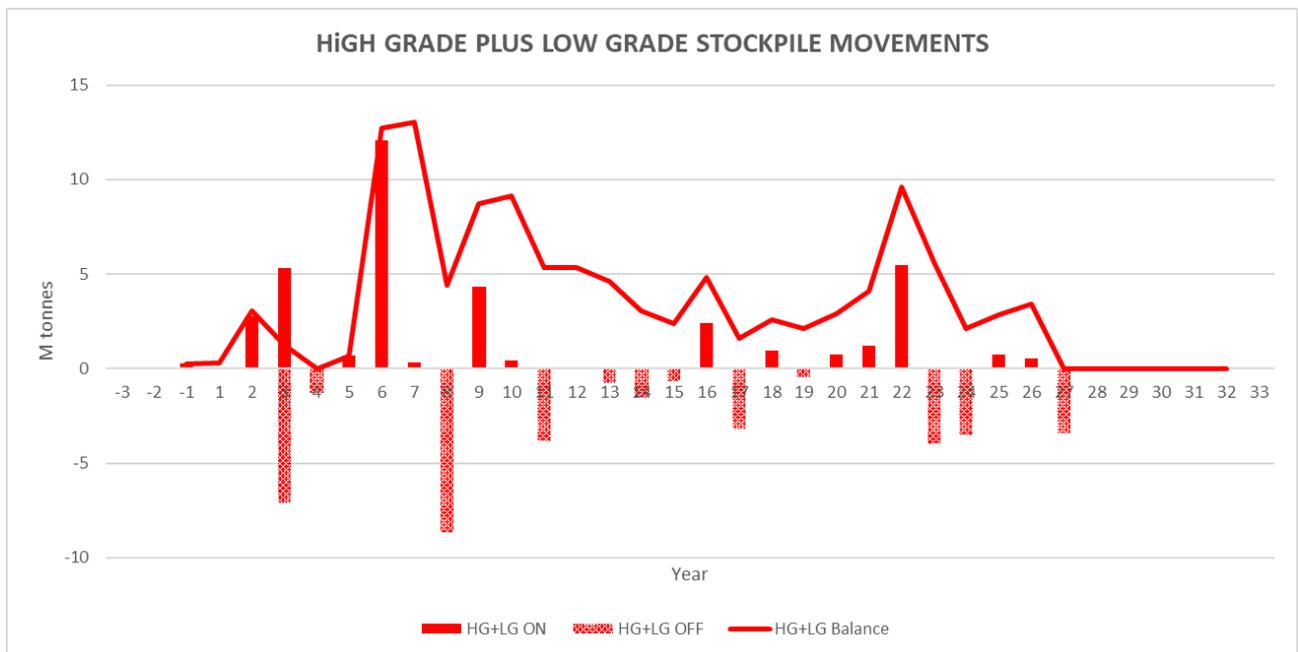
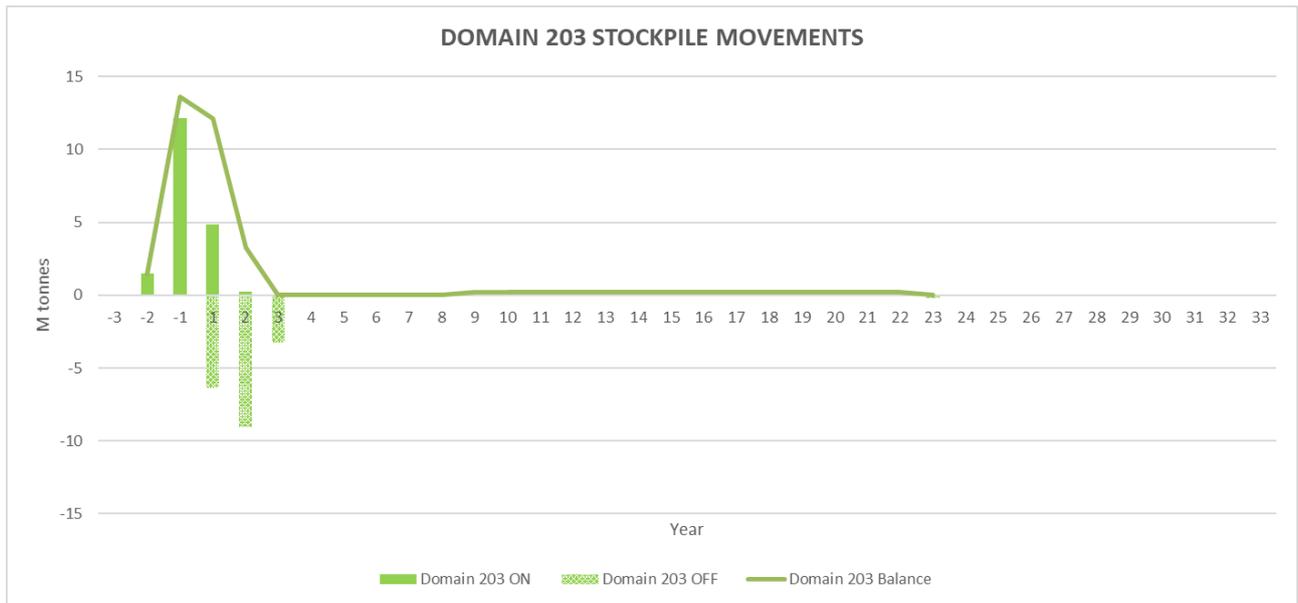
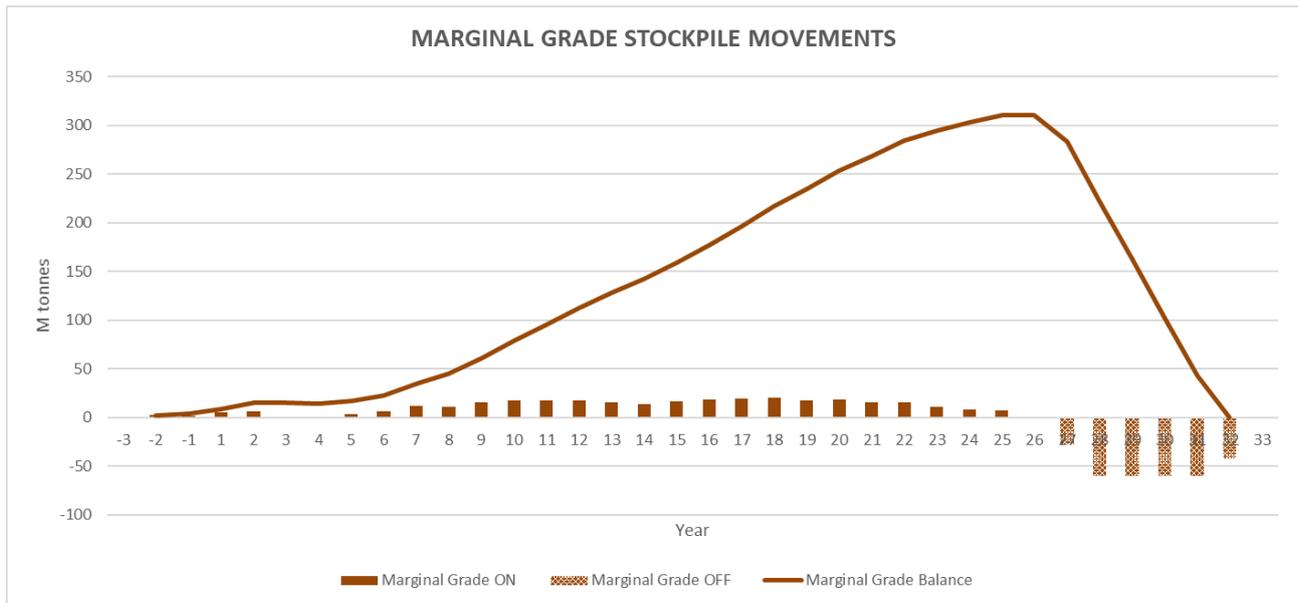


Figure 17-19 Long term stockpile movements and balance



17.4 Mining equipment requirements

The schedule of material quantities listed in Table 17-5 serves as the starting point from which to calculate the primary and secondary mining equipment requirements listed in Table 17-7 and Table 17-8, respectively.

Primary equipment represents those units which are dependent upon production, as opposed to secondary items that are subordinate to the production equipment and are estimated based on historical practice.

Table 17-7 List of annual primary mining equipment requirements

Operating Fleet	Period	-3 to -1	1 - 3	4 - 6	7 - 9	10 - 12	13 - 15	16 - 18	19 - 21	22 - 24	25 - 27	28 - 30	31 - 32
Production drill	No.	3	9	10	10	9	7	7	6	5	3	0	0
Electric rope shovel	No.	2	5	6	6	5	4	3	3	2	1	2	2
Large Excavator (800t)	No.	1	1	1	1	0	0	0	0	0	0	0	0
Large Front End Loader	No.	1	3	3	3	2	1	1	1	1	1	1	1
Haul trucks (360t)	No.	20	63	62	66	55	43	44	43	38	30	11	10
Dozers	No.	4	11	11	11	8	6	6	5	4	3	3	3
Graders	No.	5	18	17	18	15	12	12	12	11	8	3	3
Water carts	No.	4	14	14	15	12	10	10	10	8	7	3	2

Assumptions for annual primary equipment numbers include:

- production drills are planned to be electrically powered and capable of drilling 270 mm to 311 mm diameter holes to a depth of 15 m (plus sub-drill), and with drilling productivity based on a drilling rate of 25 to 30 linear m/h and 5,400 operating hours per year
- diesel powered excavators required for construction work and near-surface mining, with productivity based on a digging rate of 1,200 bcm/h and 6,000 operating hours per year
- electric rope shovels would be required for waste and ore digging, with productivity based on a digging rate of 2,450 bcm/h and 6,000 operating hours per year
- the productivity of 360 t capacity haul trucks, matched to the electric rope shovels, is based on an average trucking rate (dependant on haul profile) of 215 bcm/h and 6,000 operating hours per year

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- a large front end loader (FEL) is required for support of shovels during maintenance, blasting, relocation, etc
- a large excavator is required for waste and ore mining in the first thirteen years of mining operations, with productivity based on a digging rate of 1,900 bcm/hr and 6,000 operating hours per year
- the estimated number of graders and water carts is based on truck hours and reflects long in-pit and surface hauls

Table 17-8 List of maximum secondary mining equipment requirements

Category	Description	No of Units
Mining Support	350 t hydraulic excavator	1
Mining Support	120 t hydraulic excavator	1
Mining Support	85 t payload haul truck	4
Mining Support	Hydraulic hammer/impactor	1
Mining Support	45 t backhoe/loader	1
Mining Support	Prime mover for equipment float	1
Mining Support	Cable reeler	1
Mining Support	Stemming/roadbase crushing plant	1
Mining Support	Lighting plant	30
Mining Support	Vibratory compactor	5
Maintenance	200 t capacity diesel crawler crane	2
Maintenance	80 t rough terrain crane	1
Maintenance	5 t all-terrain forklift	2
Maintenance	16 t all-terrain forklift	3
Maintenance	30 t tyre handling forklift	1
Maintenance	Off-highway fuel/lube truck	3
Maintenance	On-highway fuel/lube truck	3
Maintenance	Light truck for field mechanics/welders	4
Maintenance	Aerial boom Truck	3
Maintenance	Utility truck with crane	3
Maintenance	Tyre handler	1
Vehicles	4x4 diesel pickup - single cab	12
Vehicles	4x4 diesel pickup - dual cab	20
Vehicles	Crew bus, diesel - 30 seat	20
Miscellaneous	Truck dispatch system	1
Miscellaneous	Ambulance	1
Miscellaneous	Mine and geology software	1
Total		127

In terms of the maximum secondary (ancillary) equipment requirements, assumptions include:

- a 350 t excavator and 85 t capacity dump trucks would be required for general haulage duties, particularly for use in restricted working areas
- track dozers would be required for clearing and stripping, for dozing on waste dumps and TSF embankment, and for establishing ramps and access ways in the pits
- wheel loaders would be required for loading from stockpiles
- water carts and a road roller/compactor would be required for road maintenance
- low bed trailers and prime movers would be required for relocating heavy equipment
- site services vehicles would include maintenance and in-pit lube trucks
- mobile cranes would be required for general service duties in and around the pit, including extracting sump and in-pit bore pumps, in-pit equipment maintenance and servicing

Other ancillary equipment would include light vehicles, crew buses and tyre handlers. A crawler/tractor would be required for picking up and relocating IPCs (if IPCC technology were to be adopted). Whilst a major

item, this crawler/tractor would only be required intermittently and the Company’s intention is to relocate such a machine between the Company’s operations as and when required.

17.5 Mine (and infrastructure) water requirements

Table 17-9 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:
 - CAT 785 (or equivalent) 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 32 years of mining, consuming 130 kL/day
 - life of mine average is 142.5 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 load-out 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 17-9 Demand and balance for mine (and infrastructure) water consumption

Water Demand	Average				Peak			
	ML/annum	kL/day	m ³ /hr	L/sec	ML/annum	kL/day	m ³ /hr	L/sec
Mining operations								
fresh water for drilling	1.6	4.5	0.2	0.1	6.8	18.6	0.8	0.2
fresh water for equipment washdown	3.7	10.0	0.4	0.1	15.1	41.4	1.7	0.5
saline water for dust suppression	7,899.4	21,642.2	901.8	250.5	15,798.8	43,284.4	1,803.5	501.0
Subtotal	7,904.7	21,656.7	902.4	250.7	15,820.7	43,344.4	1,806.0	501.7
Camp, plant and administration								
potable water	52.0	142.5	5.9	1.6	127.0	348.0	14.5	4.0
Other								
fresh water for site services, construction etc	8.4	23.0	1.0	0.3	21.9	60.0	2.5	0.7
saline water for road maintenance etc	32.5	89.0	3.7	1.0	81.2	222.5	9.3	2.6
Subtotal	40.9	112.0	4.7	1.3	103.1	282.5	11.8	3.3
TOTAL	7,997.6	21,911.2	913.0	253.6	16,050.8	43,974.9	1,832.3	509.0
Water Balance								
Mining operations								
water into the ground	897.0	2,457.6	102.4	28.4	1,795.3	4,918.7	204.9	56.9
evaporation	7,008.0	19,200.0	800.0	222.2	14,026.0	38,427.5	1,601.1	444.8
Subtotal	7,905.0	21,657.6	902.4	250.7	15,821.4	43,346.2	1,806.1	501.7
Camp, plant, administration and other								
sewage treatment	55.2	151.2	6.3	1.8	131.4	360.0	15.0	4.2
evaporation	46.4	127.2	5.3	1.5	109.5	300.0	12.5	3.5
Subtotal	101.6	278.4	11.6	3.2	240.9	660.0	27.5	7.6
TOTAL	8,006.6	21,936.0	914.0	253.9	16,062.3	44,006.2	1,833.6	509.3

17.6 Mining consumables

17.6.1 Diesel fuel and lubricants

Diesel fuel and lubricants for the mining fleet would be delivered by a combination of road and rail tankers to the main storage facility adjacent to the MSA, which would be sized to provide two weeks requirements. Average hourly and annual fuel and lubricant consumption is listed in Table 17-10.

Table 17-10 Estimated diesel fuel and lubricant consumption/annum for the mine

Equipment	Diesel fuel (litres/hour)	Fleet average (hrs/annum)	Lubricant usage (litres/hour)
800 t Diesel hydraulic excavator	500	4,900	20.0
360 t Haul trucks	195	247,000	2.8
350 t Diesel hydraulic excavator	160	2,500	8.0
85t Dump trucks	65	12,000	0.8
Track dozers	75	36,500	0.5
Large Front End Loaders	180	5,800	1.2
Graders	30	65,000	0.4
Water carts	95	52,000	0.8
Explosives Manufacturing Trucks	25	12,000	0.2
Road rollers/compactors	25	20,000	0.3
Cable reel trucks	15	2,500	0.2
Low bed trailer and prime mover	200	1,000	0.3
Mobile crane	15	10,000	0.2
Site services vehicles	15	10,000	0.2
Lighting towers	10	120,000	0.2
Light vehicles	5	150,000	0.1
Crew buses	10	25,000	0.2

17.6.2 Explosives

The blasting powder factors for ore and waste rock are likely to be different, and this is reflected in the estimated explosives consumption figures listed in Table 17-11. These figures are based on the blasting design parameters listed in Table 17-1.

Table 17-11 Estimated explosives consumption/annum

Year	Volume mined (Mm ³)		Explosives (t) Ore	Explosives (t) Waste	Total explosives (t)
	ore	waste			
-3 to -1	1.9	98.9	1,678	78,093	79,770
1 - 3	43.6	170.7	39,220	134,872	174,091
4 - 6	54.2	187.2	48,741	147,889	196,631
7 - 9	67.1	178.6	60,403	141,127	201,530
10 - 12	76.7	115.6	68,995	91,296	160,291
13 - 15	77.5	88.5	69,714	69,949	139,663
16 - 18	74.8	76.4	67,325	60,358	127,683
19 - 21	70.2	92.4	63,176	73,004	136,180
22 - 24	72.5	64.4	65,215	50,904	116,119
25 - 27	73.2	18.3	65,924	14,473	80,398
28 - 30	52.2	3.8	47,019	2,965	49,983
31 - 32	0.0	0.0	0	0	0
Tot.	663.8	1,094.8	597,409	864,930	1,462,339

17.6.3 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 17-12.

Table 17-12 Estimated lifespan and consumption/annum for mining tyres

Equipment type	Av. operating (hours/year)	Tyre life (hours/year)	Sets of tyres (#)	No. tyres (#/year)
360 t haul trucks	247,000	6,500	6	228
85 t dump trucks	12,000	5,500	6	13
Front end loaders	5,800	10,000	4	2
Graders	65,000	3,000	6	130
Water carts	52,000	4,000	6	78
Light vehicles	150,000	2,000	4	300
Others	80,500	4,000	4	81

ITEM 18 RECOVERY METHODS

18.1 Preferred processing route

The Taca Taca processing feed would be both supergene (plus mixed) and hypogene (sulphide) primary ores. Primary ores are defined as those containing more than 50% of the copper present as chalcopyrite. Consequently, when treating primary ores, significant amounts of secondary sulphides will be present in the feed, and there may also be some tarnished minerals.

The preferred process route follows that of conventional porphyry copper-molybdenum concentrators common throughout South America, but including a sulphidising flotation circuit, using sodium hydrosulphide (NaHS) for sulphidising oxide and tarnished minerals. When treating primary ores with low acid soluble copper, the same circuit will be used, but the NaHS addition would be switched off.

A gold recovery circuit would not be constructed to treat the auriferous oxide cap during the pre-strip phase of mine development. However, this material would be stockpiled separately from waste material and subject to extensive future testwork in order to define whether or not to treat it at a later stage in the Project life.

Flowsheet design and equipment selection would be based on experience gained at the Company's recent concentrator installations in Zambia (Sentinel at 55 Mtpa) and Panamá (Cobre Panamá at 100 Mtpa).

18.2 Plant location

Further to the Project Alternatives Analysis (Item 1.4) and the preferred siting for the processing plant, the location with respect to the terrain is shielded from prevailing winds coming from the north west at speeds up to 23.2 m/s (83.5 km/h). On approach from the east, the plant site would also be partially shielded from view by the topography and by the waste dump on the salar.

The milling and flotation areas will be shielded from the wind, either by sheeting of two sides and the roof of these facilities, or by erection of wind barriers on the north and west sides. Wind barriers will also be required on the flotation concentrate and tailings thickeners to reduce wave action on the surfaces which could otherwise interfere with settling rates. The crushed ore stockpile would also be covered and all conveyors would have removable covers (for maintenance), in addition to mounted dust suppression sprays.

The administration offices, warehouse, MSA, workshops and other infrastructure are proposed to be sited to the north west (i.e. upwind) of the pit and the plant area.

18.3 Process design basis and design criteria summary

The key process design criteria for the ultimate throughput of 60 Mtpa are provided in Table 17.1. The grade and recovery figures presented in this table are the average figures for the mine life. Year on year recoveries and grades will vary from the average.

Table 17.1 Summary of key process design criteria

Parameter	Units	Criteria
Annual Treatment rate	Mtpa	60
	dry tpd	180,000
Crusher Utilisation	%	70
Crusher Throughput	dry tph	10,800
Mill & Flotation Utilization	%	91.3
Mill & Flotation Throughput	dry tph	7,500
Ore Head Grades (average)	% Cu	0.44
	ppm Mo	121.4
	ppm Au	0.09
Recoveries to Final Concentrates (average)	Cu %	85.0
	Mo %	40.0
	Au %	60.0
Annual Production (average)	Cu tonnes	224,400
	Mo Tonnes	2,914
	Au kg	3,030
Copper Concentrate	% Cu	25.3
Concentrate Produced (at 10% moisture)	wet tpa	985,500
Molybdenum Concentrate	% Mo	47
Concentrate Produced (dry)	tpa	6,200
Primary Grind Size	P ₈₀ µm	180
Regrind Size	P ₈₀ µm	20 to 30
Mass Recovery to Rougher Concentrate	%	20
	dry tph	1,500
Crusher Work Index (CWi)	kWh/t	8.0
JK Parameter A*b (for SAG Mill sizing)	kWh/t	44.9
Rod Mill Work index (RWi)	kWh/t	14.0
Bond Ball Mill Work index (BWi)	kWh/t	15.6
Abrasion Index (Ai)	g	0.18

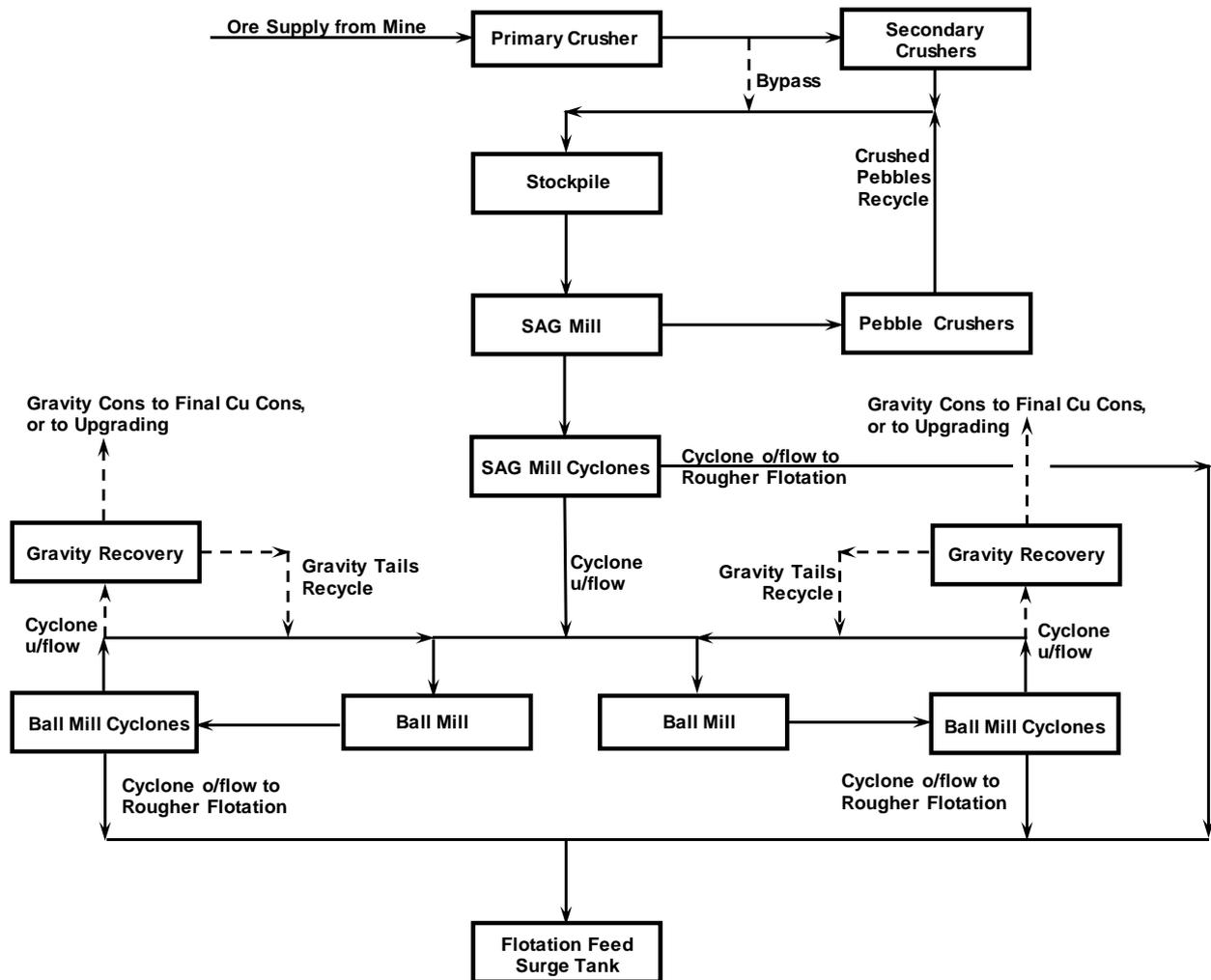
The process plant facilities would be designed for an annual throughput of up to 180,000 tpd, or 60 Mtpa. The mining ramp-up schedule shows ore production rates of up to 40 Mtpa for the first six years of operations. Therefore, there is an opportunity to defer some capital costs by staging the plant construction. Options for doing this have not been evaluated in detail, but are noted in the process summary below.

The comminution circuit would comprise a conventional SABC circuit, with secondary crushing of the SAG mill feed required to increase the throughput from the initial 40 to 60 Mtpa. The grinding circuit would consist of two trains each of 30 Mtpa capacity operating to process 60 Mtpa of ore. Each circuit would be designed to treat 3,750 tph of material from a feed size of 80% passing 130 mm to product size of 80% passing about 180 µm. A single circuit is shown in Figure 17-1.

Rougher flotation can be operated efficiently in brine. However, testwork has indicated that acceptable recoveries and concentrate grades can only be achieved in the cleaner circuit using good quality water.

Rougher concentrates would be thickened prior to regrind and diluted with fresh water for cleaning to produce a bulk Cu-Mo concentrate. Separation of Mo and Cu sulphides would be accomplished by depressing chalcopryrite 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.

Figure 17-1 Block diagram for comminution circuit



Flotation concentrates would be dewatered prior to being sent off site for further processing. Copper concentrates would be shipped in bulk by rail to a coastal port in Chile, and molybdenum concentrates would be filtered, dried and bagged for transport.

Following delivery of run of mine (ROM) ore from the pit, the concentrator circuit would comprise:

- primary crushing in three 63 x 130 gyratory crushers, with a fourth crusher to suit the 60 Mtpa throughput
- secondary crushing of mill feed to a P_{80} of 75 mm, ahead of the crushed ore stockpile
- the secondary crushing circuit would not be installed until the throughput increases to 60 Mtpa (Year 8)
- conveying of crushed ore to the coarse ore stockpile
- a coarse ore stockpile with 12 hours of live capacity (90,000 t)
- SAG and ball milling of crushed ore, with size classification by means of hydrocyclones
 - two grinding circuits would be installed, each comprising a 28 MW SAG mill and two 22 MW Ball mills, for 60 Mtpa processing capacity
 - the target grind size would be 80% passing 180 μm
 - a block flowsheet of a single comminution circuit is presented in Figure 17.1
- a gravity recovery circuit on ball mill cyclone underflow for coarse gold recovery
- pebble crushing on scats generated from the SAG mills

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- two MP 1250 crushers would be installed for this duty
- rougher and scavenger flotation of cyclone overflow slurry with controlled sulphidisation flotation using NaHS for oxidised and tarnished supergene ores
 - two circuits in parallel would be installed for 60 Mtpa, each comprising a single train of seven 600 m³ flotation cells to give approximately 25 minutes residence time
 - for the initial throughput of 40 Mtpa, one circuit only would be installed, but with nine cells to provide sufficient residence time
 - a block flowsheet of the flotation and concentrate handling circuits is provided in Figure 17-2
- thickening of rougher flotation tails in three 50 m diameter thickeners operating in parallel
- pumping of thickened tailings to the TSF
- reclaim of decant water from the TSF for usage within the process
 - note that climatic conditions (very dry, constant winds, high solar radiation) may limit the amount of decant to be returned
- dewatering of rougher concentrates, to remove brine water
- regrind of dewatered rougher concentrates to 80% passing 30 µm in high intensity grinding (HIG) mills (12 MW of installed capacity), followed by dilution in fresh water
- 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
 - two parallel cleaner flotation circuits would be installed
 - the higher feed grades expected in the initial processing years would require both circuits to be installed from Year 1
- dewatering of cleaner scavenger tails in a single 50 m diameter thickener, followed by pumping of the thickened underflow to final tailings
- dewatering of bulk Cu-Mo concentrates in a 50 m diameter thickener, for recycle of Cu float reagents
- Cu – Mo separation of the bulk cleaner concentrates in a molybdenum flotation circuit, comprising five to seven stages of cleaning
- 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
- use of water reticulation systems
- use of compressed air systems to support instrumentation and for automatic valve activation
- low pressure air systems provided by blowers for the flotation cells

The circuit would be designed for an annual throughput rate of up to 60 Mtpa, with a plant availability of 91.3% (8,000 hrs per year). Hourly throughput rates would be 7,500 tph, which gives a daily throughput (full 24 hours operation) of 180,000 tonnes. 985,500 wet tonnes of copper concentrate is expected to be generated annually at an average grade of 25.3% Cu and 10% moisture along with 6,200 tonnes of molybdenum concentrate at a grade of 47% to 50% Mo.

Figure 17.2 Block flowsheet for flotation and tailings

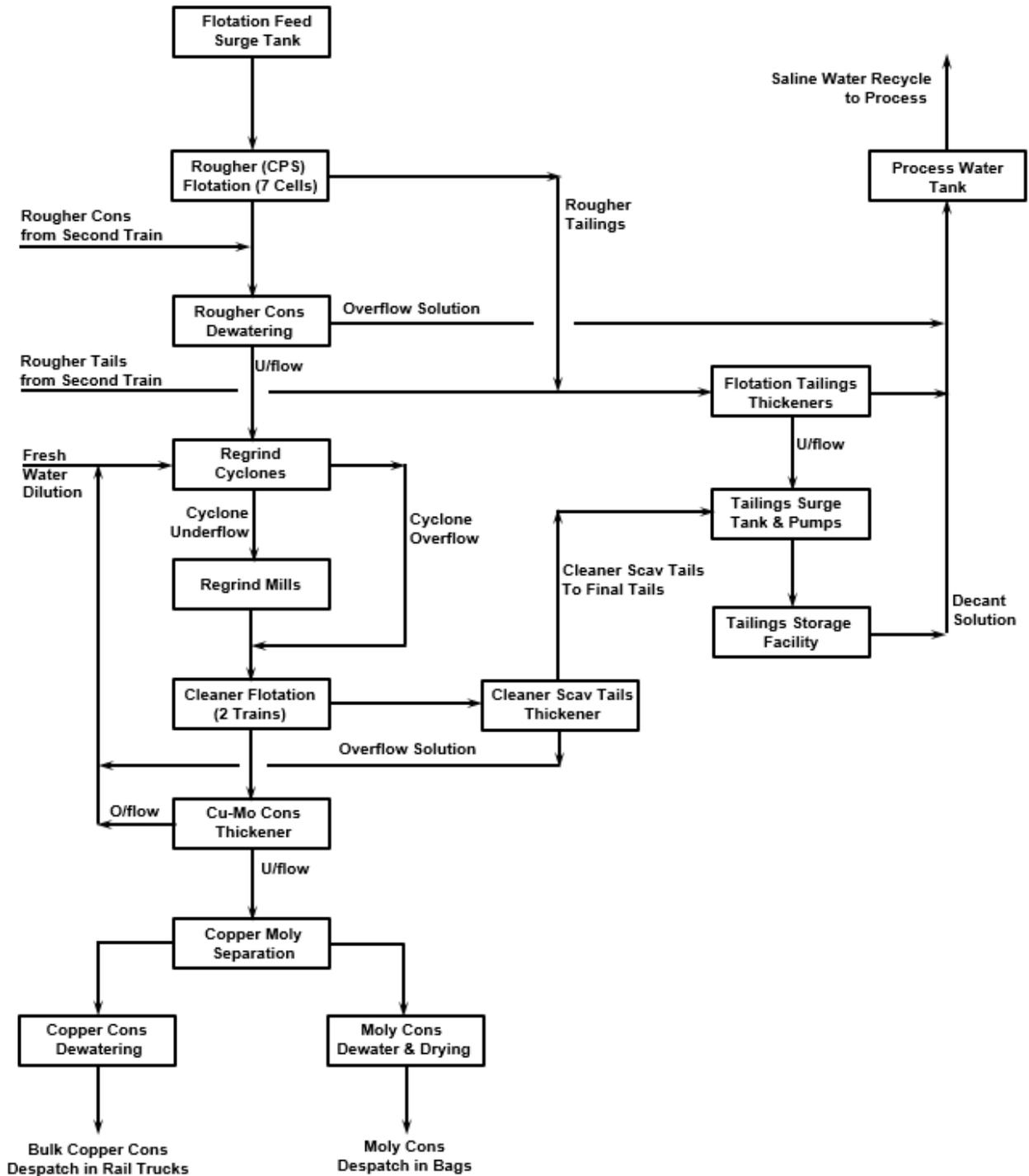


Figure 17.3 shows the location of the conceptual surface ROM pad and primary crushers, whilst the plant layout is shown in Figure 17.4.

Figure 17.3 Conceptual primary crushing layout

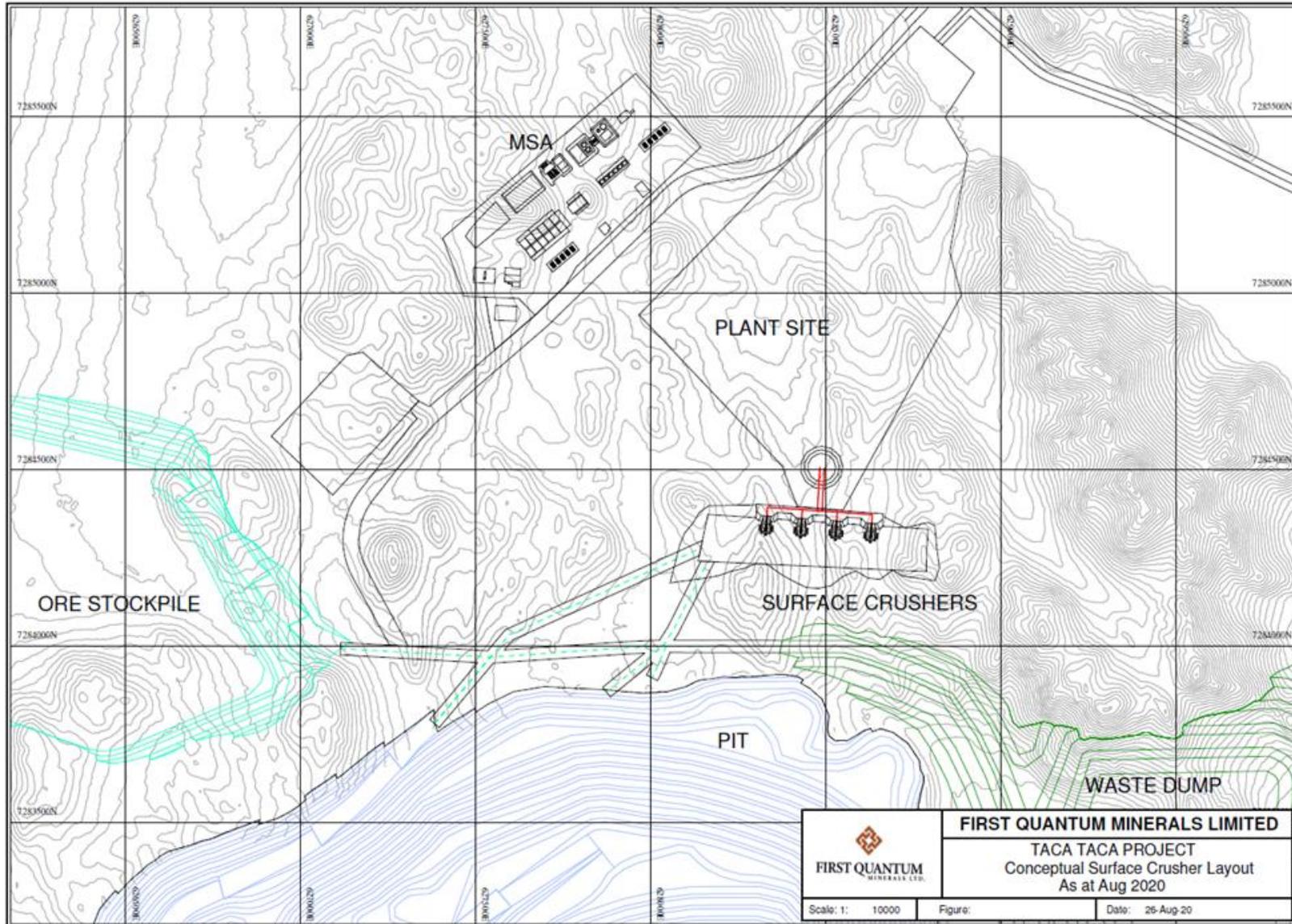
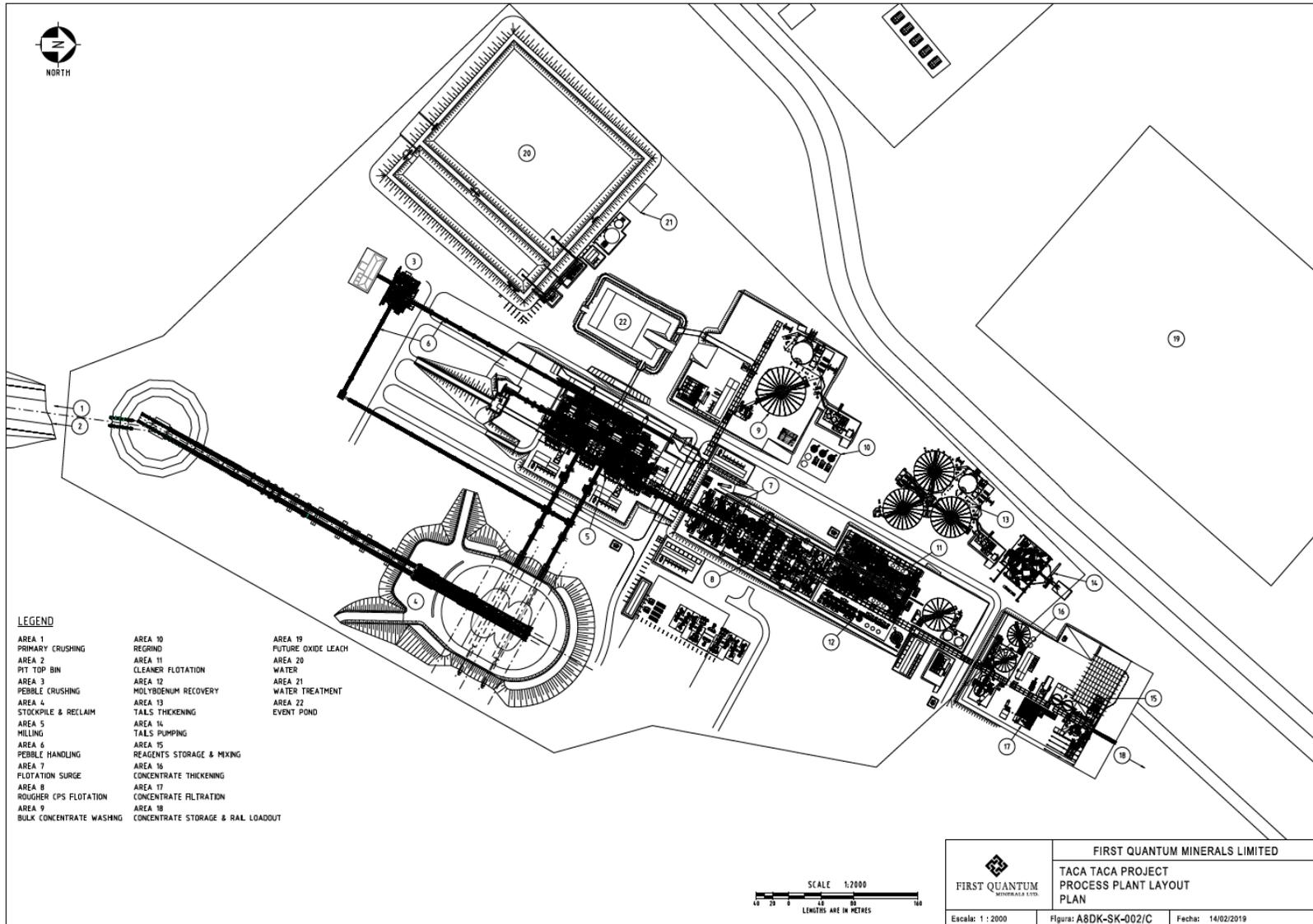


Figure 17.4 Conceptual Layout of flotation/concentrator plant



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The plant layout is indicative only; it has been adapted from the Sentinel plant layout (Sentinel design was 55 Mtpa), with minor adjustments to fit the site selected for the plant.

There has been no detailed engineering undertaken to date, and several differences between Sentinel and Taca Taca have not been incorporated into this layout. The major differences include:

Changes to the flowsheet and layout that have been identified to date include:

- Secondary crushing of ore from the primary crushers, prior to the stockpile. Bypass facilities around the secondary crushers would be allowed for.
- Rougher concentrate dewatering, to allow for a change in water quality ahead of cleaner flotation. Depending on the results from trade-off studies, dewatering of rougher concentrates might be achieved by filtration instead of thickening.
- Rougher flotation would comprise two rows of 600 m³ rougher flotation cells, instead of four rows of 300 m³ cells.
- Addition of Jameson cells and flotation columns in the cleaner circuit. The Jameson cells might operate as coarse cleaners discharging concentrate directly to final concentrate.
- An additional tailings thickener dedicated to cleaner scavenger tails, to allow recycle of good quality water.
- Installation of copper concentrate dewatering filters at the concentrate shed by the rail load-out facilities. Alternatively, concentrate could be filtered at the main concentrator and conveyed to the concentrate shed.

The site selected for the process plant should be re-evaluated as the Project engineering phase proceeds, to be absolutely sure it is the optimum location for the concentrator with respect to the other facilities required for the Project.

18.4 Summary of processing consumables

Concentrator reagent requirements have been defined following various testwork campaigns undertaken by the previous owners in 2011, the recent testwork conducted in brine, or estimated based on design experience at the Company's other sites. The consumption rates detailed below refer to an annual treatment rate through the concentrator of 60 Mtpa.

Table 17-2 summarises the estimated annual quantities of concentrator consumables.

Table 17-2 Estimated concentrator consumables/annum

Consumables		Consumption Rate, t		
		Design Basis	tpd (24 hrs)	tpa (8,000hrs)
Grinding Media				
SAG Mill Balls	140 mm	0.28 kg/t ore	51.0	16,800
Ball Mill Balls	65 mm	0.45 kg/t ore	81.0	27,000
Regrind Mill Balls	Ceramic Beads	0.02 kg/t ore	3.4	1,140
Lime Ball Mill Media	50 mm	0.80 kg/t lime	115 kg/d	38
Reagents				
Frother	MIBC	100 g/t ore	18.0	6,000
Collector	SEX	120 g/t ore	21.6	7,200
NaHS for CPS	NaHS	1 kg/t ore	180.0	Note 1*
NaHS for Mo recovery	NaHS	15 kg/t cons	34.0	12,515
Lime (pH Modifier)	Quicklime	800 g/t ore	144.0	48,000
Mo Promoter		10 g/t ore	1.8	600
Mo Collector	Pine Oil	50 g/t conc	115 kg/d	42
Diesel Oil		15 g/t conc	35 kg/d	13
Mo Feed pH Control	Sulphuric Acid	allowance	1.0	330
Dispersant	Sodium Silicate	0.13 kg/t conc	330 kg/d	110
Scrubber Solution	Sodium Hydroxide	0.33 kg/t conc	840 kg/d	280
Flocculant	Flotation Tails	50 g/t feed	9.0	3,000
Flocculant	Concentrate	25 g/t conc	66 kg/d	22

Notes:

1. NaHS consumption in the main sulphide flotation circuit is expected to be intermittent, depending on ore feed quality. Therefore an annual consumption figure cannot be determined.
2. The consumable consumption rates detailed above are the design quantities for the reagent make-up systems; these are often higher than the anticipated actual requirements used to define the process operating costs.

18.5 Process plant water balance

Brine water would be used for milling and for rougher flotation, but may not be suitable for cleaner flotation, as it would affect recoveries and concentrate grades.

At 60 Mtpa, the processing facilities would require approximately 6,086 m³/h of new water, which would be made up of:

- 1,546 m³/h (546 L/sec) of fresh water,
- 4,540 m³/h (1,261 L/sec) of brine water

These quantities of new water assume that no TSF decant water is available. If in fact it is possible to return water from the tailings facilities, the quantity of brine water, the brine water make-up would decrease accordingly.

The summary water demand and balance for the processing plant is summarised in Table 17-3 and shown in Figure 17-5.

Table 17-3 Summary water demand and balance for the processing plant

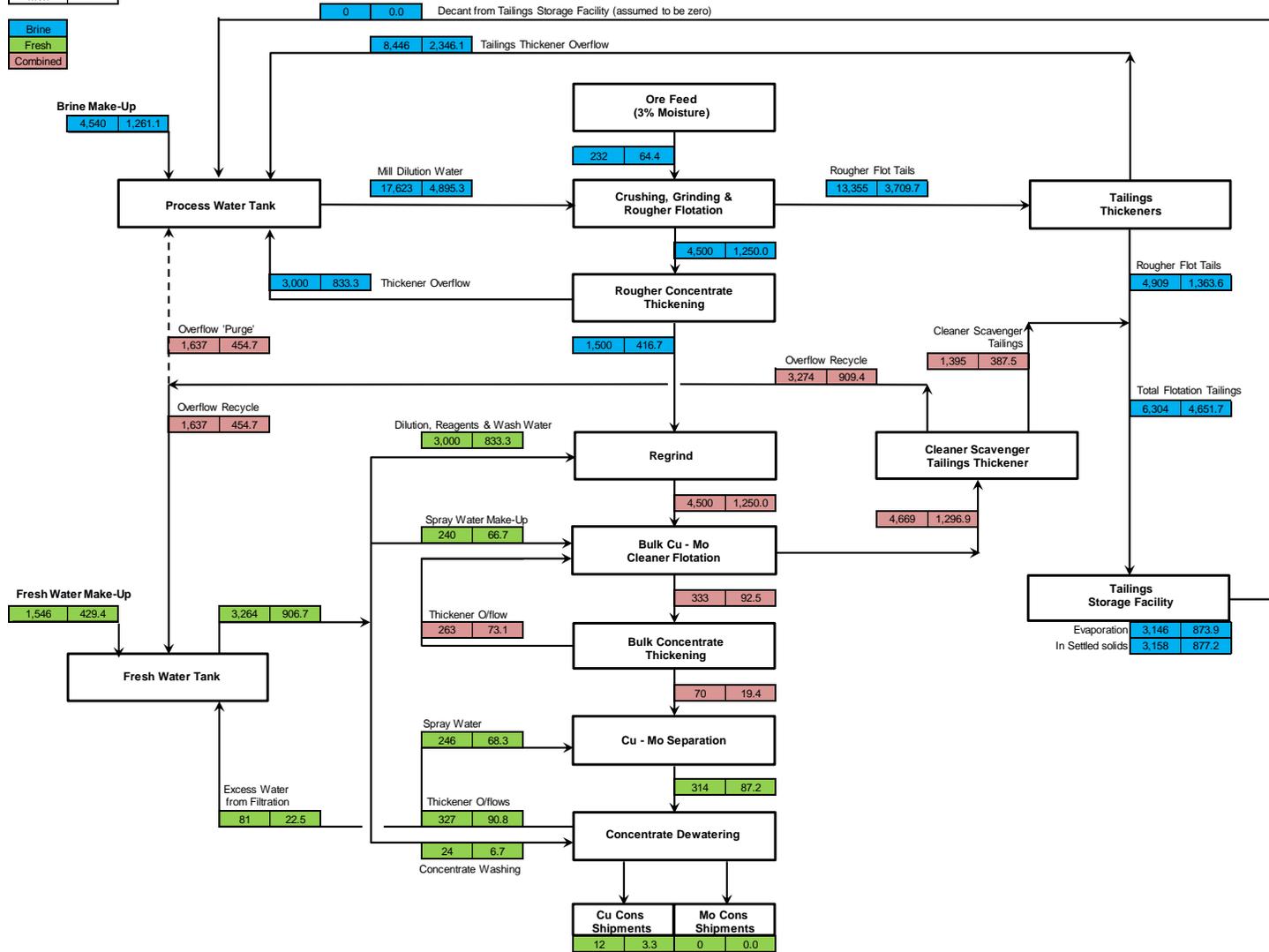
Water Demand		Overall			
		ML/annum	kL/day	m ³ /h	L/s
	With ore processed	1,856	5,568	232	64
	Brine make-up	36,310	108,960	4,540	1,261
	Fresh water make-up	12,365	37,104	1,546	429
	Subtotal	50,531	151,632	6,318	1,755
Water Loss		Overall			
		ML/annum	kL/day	m ³ /h	L/s
	In settled solids in TSF	25,257	75,792	3,158	877
	Evaporation at TSF	25,161	75,504	3,146	874
	In concentrates	96	288	12	3
	Subtotal	50,515	151,584	6,316	1,754

Figure 17-5 Plant water demand and balance

Summarised Process Water Balance
Based on an ore throughput of 60 Mtpa (7,500tph)

m ³ /h	L/s
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Brine
Fresh
Combined



18.6 Outstanding process design items

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 new design considerations and enhancements will be incorporated as the Project engineering phase proceeds. The outstanding items that have been identified are listed below:

- review the location of the primary crushers (in conjunction with the mine planning engineers in respect of the potential for in-pit crushing and conveying)
- trade-off study for options on rougher concentrate dewatering (filtration vs. thickening)
- configuration of the cleaner flotation circuit and inclusion of flotation columns and Jameson cells
- review concentrate optimal regrind size and define the regrind circuit power requirements and mill sizing
- review the economics of molybdenum production (production of less than 10,000 tpa of concentrate is indicated) and design the Cu-Mo separation circuit
- review the sizing of the reagent makeup systems in light of the uncertainty with respect to some reagent consumption rates
- define the optimum location for copper concentrate dewatering (main plant or load-out facilities)
- review of the economics of leaching the auriferous material from the near-surface leached cap
- specification of construction requirements to suit the climatic conditions, notably wind loads and catering for the corrosive aspects of brine
- review the options for stage construction of the processing facilities
- specify major equipment that can be deferred, but ensure constructability at a later date in an operating plant
- update of the concentrator flowsheet, the layout plan and related facilities

ITEM 19 PROJECT INFRASTRUCTURE

19.1 Introduction

The information provided under this Item reflects the preliminary level of engineering that has been completed to date, now providing a focus for continuing efforts as the Project engineering phase proceeds.

19.2 Seismic conditions

Seismic criteria and regulations 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 are no specific regulations 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 Taca Taca 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 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 based on a detailed engineering risk analysis. A site specific seismic hazard study will be performed during the on-going engineering phase and will be used for the design of infrastructure including the TSF. This study will include a review of the historical seismicity records, the regional tectonic features, and the definition of seismic sources and their respective recurrence.

19.3 Power supply

The nearby town to the Project site, Tolar Grande, generates its own power requirements from diesel fuel. The nearest power transmission line to the Project site is to the north in the vicinity of La Puna (Figure 19-1). This is a 345 kV line from the Güemes generating station in Salta Province, extending to Los Andes in Chile. The line is privately owned and operated by *Termoandes SA* (*Termoandes*).

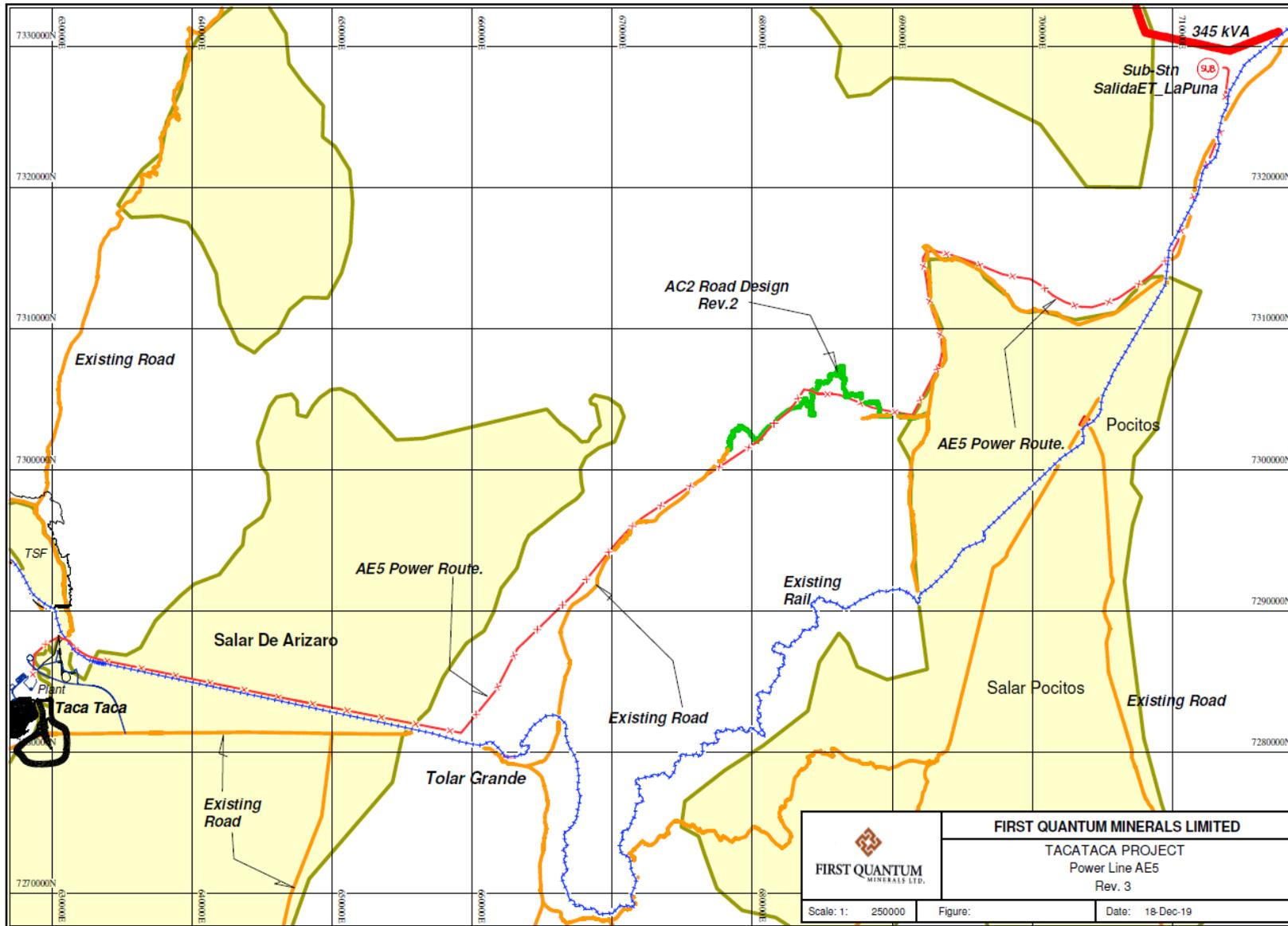
From a number of power generation alternatives including on-site diesel fuelled generation, wind-turbine generation, natural gas fuelled generation, and solar powered generation, a straight-forward connection to an existing 345 kV transmission line was selected as the preferred alternative in the Project Alternatives Analysis (Ausenco, 2018). This was primarily for reasons of available capacity and reliability.

Five grid connection alternatives were considered in the Project Alternatives Analysis for power transmission to the Project site. Each of these involved a new switching station, at varying locations along the existing line. *Tranelsa Transmision Electrica* (Tranelsa, May 2017) reviewed the power transmission alternatives for the Project and their preferred alternative is for a new line branching off from a switching station at La Puna near Olacapato, and then proceeding south westerly and south of the Pocitos Volcano. The route is then south west over the Cerro Maçon, across the Salar de Arizaro and into the Project site. This transmission preference was made on the basis of shortest distance and lower relative cost.

The *Tranelsa* recommended route (AE5), after some further modification by FQM, is shown in Figure 18-1.

A power supply study and report was completed by *Tecnolatina SA* in November 2019 (Tecnolatina, 2019) in which it was concluded that the proposed power supply to Taca Taca will not compromise the existing transmission.

Figure 19-1 Preferred power transmission line route



19.3.1 Maximum installed power requirements

The estimated total demand for the mine, processing facilities and infrastructure is expected to be in the range of 180 MW to 240 MW at a processing rate of 60 Mtpa. To meet the requirement of supply authorities, the site power factor would be corrected to 0.95 lagging or better, resulting in a site peak demand of 260 MVA.

19.3.2 Supply requirements and substations

Power distribution for the Project site would be primarily for the following areas, at 9 kV, 4.16 kV, and 33 kV supply:

- the process plant
- the mine
- MSA
- ancillary facilities, administration, workshops, warehouse, etc.
- camps
- remote areas, e.g. TSF, water reservoir, water return pumping system (should that be feasible), potable water pumping system, etc.

There will also need to be new-build 66 kV transmission lines radiating from a central site 345/66/33kV substation to each of the outlying fresh water borefields at Valle de las Burras, Valle de Arizaro, Valle de Chaschas and Socompa.

The overhead line design, method of construction and supply network tie-in point at the Project site are yet to be determined.

Project site

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 (subject to further evaluation during the continuing engineering phase). 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), set of pole fuses, set of lightning arrestors for lightning protections, motor control centre (MCC), lighting and small power board. The MCC and the lighting distribution board would be housed in a standard substation building. The power transformer would be oil filled, oil-natural-air-natural (ONAN) cooling with off-load tap changes.

The mine services, ancillary facilities and man camps 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.

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 for surface and in-pit dewatering, and lighting equipment. Power is planned to be transmitted to the operating benches by a series of portable substations (approximate capacity 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 (to the extent adopted) is planned to be distributed by a series of

dedicated overhead power lines on each haulage ramp and associated substations ('e-house') located at approximately 350 m intervals along the haul road ramp.

19.3.3 Renewable energy

A proposed connection to the existing 345 kV transmission line from Güemes is the current basis for the Company's estimated power supply requirements and capital cost provisions.

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.

19.4 Water supply

Regional borefields will be developed to supply a combination of high and low salinity water for the Project. Brine water from a proposed borefield on the adjacent Salar de Arizaro is intended for use in milling and rougher flotation. The balance of the process water supply is intended to be fresh or brackish water abstracted from borefields in regional water storage catchment areas. Investigations by Ausenco (May, 2016), SRK (2015) and latterly by FQM indicate that there are several preferred regional areas for priority fresh water exploration studies and for the required development of a sustainable fresh water supply. These are at Valle de Arizaro, Valle de las Burras, Valle de Chaschas and Socompa.

Because of the fundamental importance of this water supply to the Project, detailed information on supply investigations is provided under Item 24.

19.4.1 Desalination assessment

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 report (Ausenco, 2013) stated that fresh water of not greater than 1,500 mg/L TDS (total dissolved solids) is preferred for cleaner flotation, cooling, reagent mixing and concentrate washing. On the other hand, brine water at up to 300,000 mg/L TDS may be used for grinding and rougher flotation.

Information in the PEA report 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.

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 brine waters 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, fresh and brine water abstraction quantities at suitable TDS levels may be possible.

19.4.2 Project water requirements

Detailed information on water consumption (and balance) requirements is provided in:

- Item 17.5 Mine (and infrastructure) water requirements - in relation to the mine site and Project infrastructure (e.g. camp and related facilities)

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- Item 17.5 Process plant water balance – in relation to the process plant

Table 19-1 summarises the complete water demand from all sources, summarised by fresh and saline (“brine”) demand quantities, and by processing, mining, camp and other consumption activities. A summary water balance is provided in Table 19-2 for all consumption activities including water lost into the ground, into tailings, and due to evaporation.

These tables reflect the overall Project water demand and balance chart shown in Figure 19-2.

Table 19-1 Summary water demand for all consumption activities

Water Demand	Average				Peak			
	ML/annum	kL/day	m ³ /h	L/s	ML/annum	kL/day	m ³ /h	L/s
Fresh water demand								
for processing	13,543.0	37,104.0	1,546.0	429.4	13,543.0	37,104.0	1,546.0	429.4
for the camp	52.0	142.5	5.9	1.6	127.0	348.0	14.5	4.0
for mining	5.3	14.5	0.6	0.2	21.9	60.0	2.5	0.7
for site services, rail load-out & construction	8.4	23.0	1.0	0.3	21.9	60.0	2.5	0.7
Subtotal	13,608.7	37,284.0	1,553.5	431.5	13,713.8	37,572.0	1,565.5	434.9
Brine demand								
for processing	39,770.4	108,960.0	4,540.0	1,261.1	39,770.4	108,960.0	4,540.0	1,261.1
for mining	7,899.4	21,642.2	901.8	250.5	15,798.8	43,284.4	1,803.5	501.0
for road maintenance & construction	32.5	89.0	3.7	1.0	81.2	222.5	9.3	2.6
Subtotal	47,702.3	130,691.2	5,445.5	1,512.6	55,650.4	152,466.9	6,352.8	1,764.7
Water in ore processed	2,032.3	5,568.0	232.0	64.4	2,032.3	5,568.0	232.0	64.4
TOTAL	63,343.3	173,543.2	7,231.0	2,008.6	71,396.5	195,606.9	8,150.3	2,264.0
Processing summary								
fresh make-up	13,543.0	37,104.0	1,546.0	429.4	13,543.0	37,104.0	1,546.0	429.4
brine make-up	39,770.4	108,960.0	4,540.0	1,261.1	39,770.4	108,960.0	4,540.0	1,261.1
water in ore processed	2,032.3	5,568.0	232.0	64.4	2,032.3	5,568.0	232.0	64.4
Subtotal	55,345.7	151,632.0	6,318.0	1,755.0	55,345.7	151,632.0	6,318.0	1,755.0
Mining summary								
fresh	5.3	14.5	0.6	0.2	21.9	60.0	2.5	0.7
brine	7,899.4	21,642.2	901.8	250.5	15,798.8	43,284.4	1,803.5	501.0
Subtotal	7,904.7	21,656.7	902.4	250.7	15,820.7	43,344.4	1,806.0	501.7
Camp and other								
fresh	60.4	165.5	6.9	1.9	148.9	408.0	17.0	4.7
brine	32.5	89.0	3.7	1.0	81.2	222.5	9.3	2.6
Subtotal	92.9	254.5	10.6	2.9	230.1	630.5	26.3	7.3
TOTAL	63,343.3	173,543.2	7,231.0	2,008.6	71,396.5	195,606.9	8,150.3	2,264.0

Table 19-2 Summary water balance for all consumption activities

Water Consumption	Average Water Balance				Peak Water Balance			
	ML/annum	kL/day	m ³ /h	L/s	ML/annum	kL/day	m ³ /h	L/s
Processing summary								
water in settled solids at the TSF	27,664.1	75,792.0	3,158.0	877.2	27,664.1	75,792.0	3,158.0	877.2
TSF evaporation	27,559.0	75,504.0	3,146.0	873.9	27,559.0	75,504.0	3,146.0	873.9
water in concentrates	105.1	288.0	12.0	3.3	105.1	288.0	12.0	3.3
Subtotal	55,328.2	151,584.0	6,316.0	1,754.4	55,328.2	151,584.0	6,316.0	1,754.4
Mining summary								
water into the ground	876.0	2,400.0	100.0	27.8	1,753.3	4,803.4	200.1	55.6
evaporation	7,023.4	19,242.2	801.8	222.7	14,056.8	38,511.9	1,604.7	445.7
Subtotal	7,899.4	21,642.2	901.8	250.5	15,810.1	43,315.3	1,804.8	501.3
Camp and other								
sewage treatment	52.0	142.5	5.9	1.6	131.4	360.0	15.0	4.2
evaporation	41.8	114.5	4.8	1.3	109.5	300.0	12.5	3.5
other	21.9	60.0	2.5	0.7	17.3	47.5	2.0	0.6
Subtotal	115.7	317.0	13.2	3.7	258.2	707.5	29.5	8.2
TOTAL	63,343.3	173,543.2	7,231.0	2,008.6	71,396.5	195,606.9	8,150.3	2,264.0

19.4.3 Water supply summary

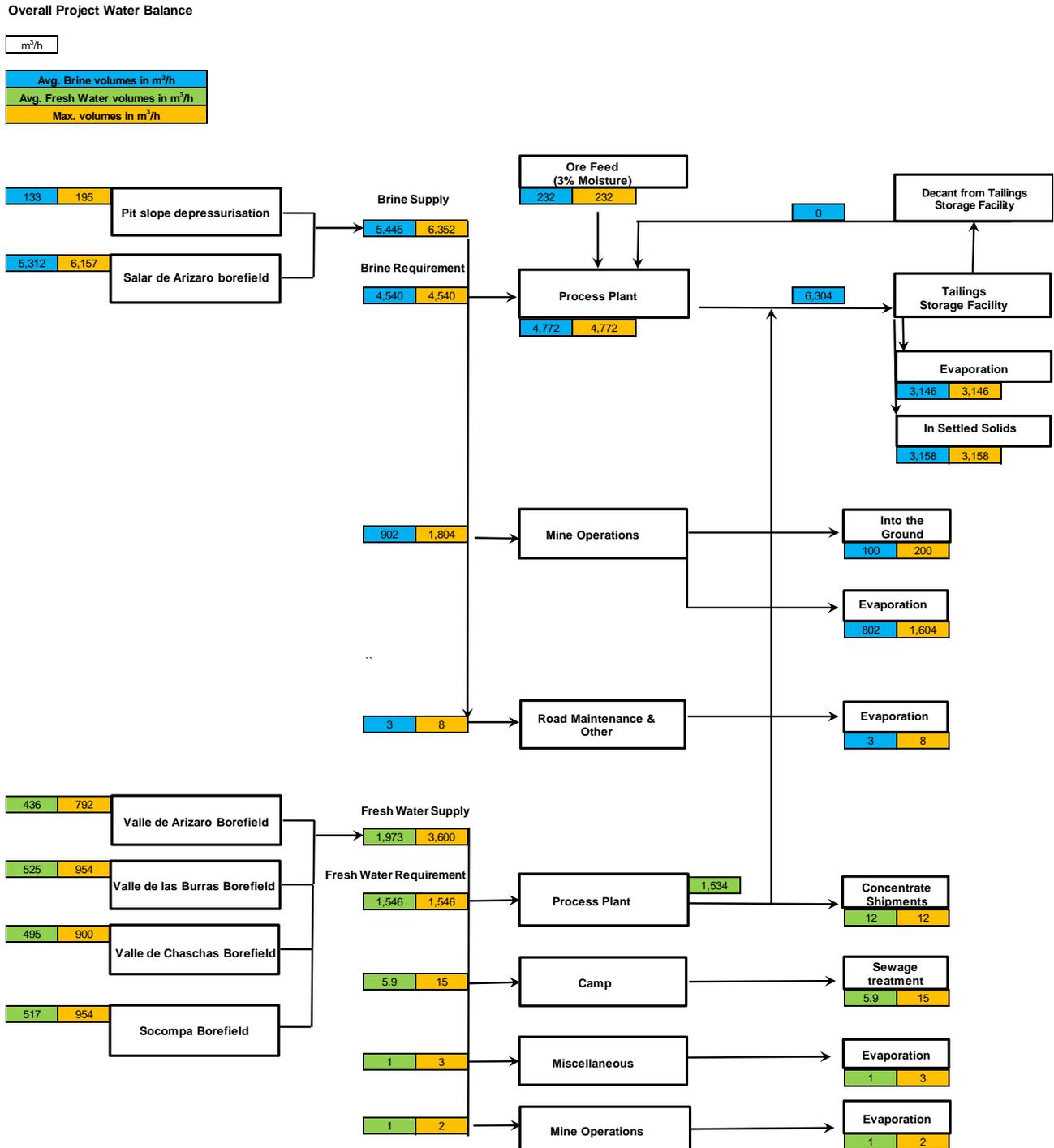
Montgomery & Associates (M&A, December 2018) completed a review of Project water supply and operational requirements for inclusion with the Project ESIA submission. In their 2018 review, M&A focussed the fresh water supply source alternatives assessment on Valle de Arizaro, Valle de las Burras and Caipe.

Fresh water borefields were envisaged as two within the Valle de Arizaro basin, two within the Valle de las Burras basin and one at Caipe. The borefields were conceptualised as a central tank and pump station

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associated with five to seven pumping bores at each site. Nominal bore spacing was considered to be 800 m to 1,000 m and preliminary average pumping rates of 15 L/sec to 25 L/sec were nominated.

Figure 19-2 Project water demand and balance



Following regional reconnaissance work in late 2018 and 2019, the likely fresh water sources were revised. In place of Caipe, Valle de Chaschas and Socompa were substituted. Table 19-3 lists the estimated number of bores required at these proposed borefields, along with the required production rates.

Consistent with previous concepts, M&A assumed that brine water supply would come primarily from a number of bores on the Salar de Arizaro, located in close proximity to the processing plant and with several of these providing a dual role of depressurising the open pit eastern wall.

Table 19-3 Borefield summary

Source/location	Average water supply		Peak water supply		Borefield summary		
	m ³ /h	L/s	m ³ /h	L/s	no. of	av. bore	
					production	production rate	
					bores	m ³ /h	L/s
Fresh water supply							
Valle de Arizaro	435.5	121.0	792.0	220.0	9	90.0	25.0
Valle de las Burras	525.0	145.8	954.0	265.0	11	90.0	25.0
Valle de Chaschas	495.0	137.5	900.0	250.0	10	90.0	25.0
Socompa	517.0	143.6	954.0	265.0	8	126.0	35.0
Subtotal	1,972.5	547.9	3,600.0	1,000.0	38	99.0	27.5
Brine supply							
pit depressurisation	133.0	36.9	194.8	54.1	4	6.8	1.9
Salar de Arizaro	5,312.1	1,475.6	6,157.0	1,710.3	115	60.8	16.9
Subtotal	5,445.1	1,512.5	6,351.8	1,764.4	119	67.6	18.8

The peak fresh water supply potential from the four borefields is in excess of the total peak demand of 1,565.5 m³/h (Table 19-1). The listed peak brine water supply matches the peak demand (Table 19-1).

Drilling and borefield evaluation is currently on-going (Phase III, refer to Item 24), and as the Project engineering phase proceeds, regional knowledge of fresh water sources and sustainability will increase, thereby allowing further potential borefield production rate vs engineering/logistical trade off studies to be undertaken.

In terms of brine water supply, possible alternative layouts for wellfields, pit dewatering, conveyance systems, and potential infrastructure will also be evaluated as the engineering phase proceeds.

19.4.4 Supply pipelines

Valle de Arizaro, Valle de Las Burras, Valle de Chaschas and Socompa have been selected as the preferred primary sources of fresh water supply for the Project. In addition to the above water sources, it is also planned to develop a borefield on the Salar de Arizaro, adjacent to the proposed open pit, to supply brine water for the processing plant.

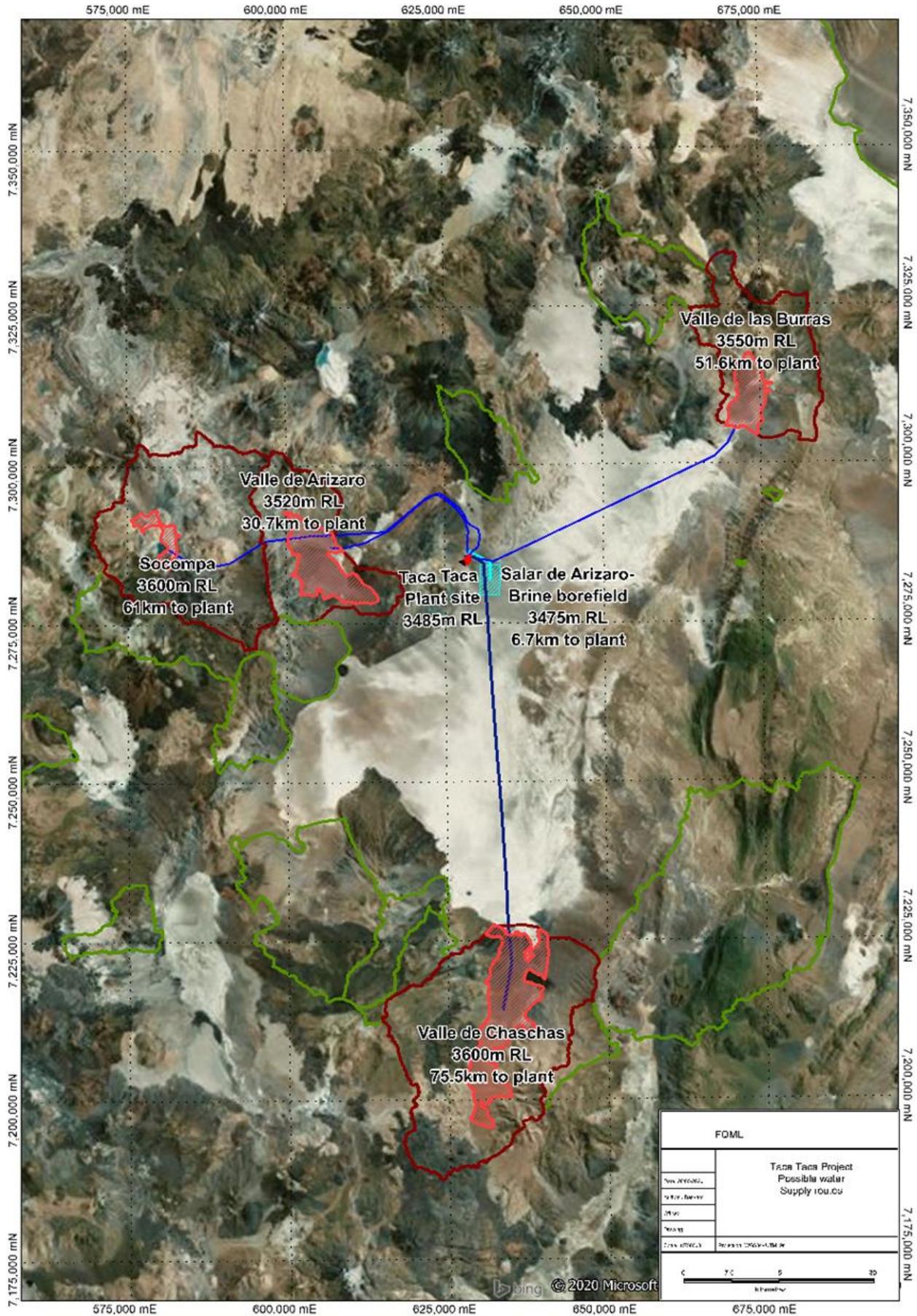
Figure 19-3 is a plan view of the proposed pipeline routes. This information can be translated into the pipeline distances and pumping heads listed in Table 19-4.

Details of suitable bore pumps and pipeline specifications, as assessed by the Company, are provided in the capital cost estimates of Item 22.2.7.

Table 19-4 Pipeline distances and pumping head

Source/Location	Overland pipeline distance (km)	Elevation of Bore Field (m)	Maximum elevation of route (m)	Lift required (m)	Elevation of Plant (m)
Fresh Water Supply					
Valle de Arizaro	30.7	3,520	3,660	140	3,485
Valle de Las Burras	51.6	3,550	3,550	0	3,485
Valle de Chacchas	75.5	3,600	3,600	0	3,485
Socompa	25.9	3,600	4,570	970	3,485
Subtotal	183.7				
Brine Supply					
Pit Dewatering	-				
Salar de Arizaro	6.7	3,475	3,485	10	3,485
Subtotal	6.7				

Figure 19-3 Water supply pipeline routes



19.5 Road access

Existing public roads provide access to the Project site from Salta, Argentina. To the north east of the Project site, a bypass of the existing road is envisaged to avoid a section with narrow switchbacks and another section which is subject to seasonal weather disruptions.

Existing public roads may also be used to access the site from Chile through border crossings at either Paso de Sico or Paso Socompa.

19.5.1 Proposed road deviation

Four possible Project access routes were identified in the Project Alternatives Analysis (Ausenco, 2018) in order to avoid the switchbacks (“the seven curves”) on RP N° 27, south of Cauchari and in the vicinity of Los Colorados (Figure 19-4), and to also bypass Tolar Grande. Each of these alternatives has a common route corresponding to RN N° 51 between Salta and Cauchari. A preferred alternative has been selected involving a deviation at a point south of Cauchari (at km 28), passing over Cerro Macón in the north to later re-join RP N°27 near Tolar Grande, and thence to the Project site.

The route has been planned using detailed satellite imagery survey data. The route, referred to from the alternatives analysis as AC 2, is shown on Figure 19-4 with the length requiring new construction shown in detail in Figure 19-5. The AC 2 deviation has been designed with a maximum 1 : 20 road gradient and minimum curve radius of 100 m to suit heavy haulage vehicles. The total distance is 147 km and the length of new road to be constructed is about 26.5 km in length. An additional 18.5 km length of existing road is to be deviated around Tolar Grande and 31 km deviated around the Project site.

Figure 19-6 shows the existing AC2 road route, the proposed new length of construction and the proposed bypasses around Tolar Grande and the Project site.

A supplementary access route through Chile, referred to AC 5, is also shown in Figure 19-4 as coming into the Project site from the north. This route will be considered at a later Project phase as a means of shortening the road access distances through Chile.

Figure 19-4 Alternative road access routes, including the AC 2 and AC 5 routes

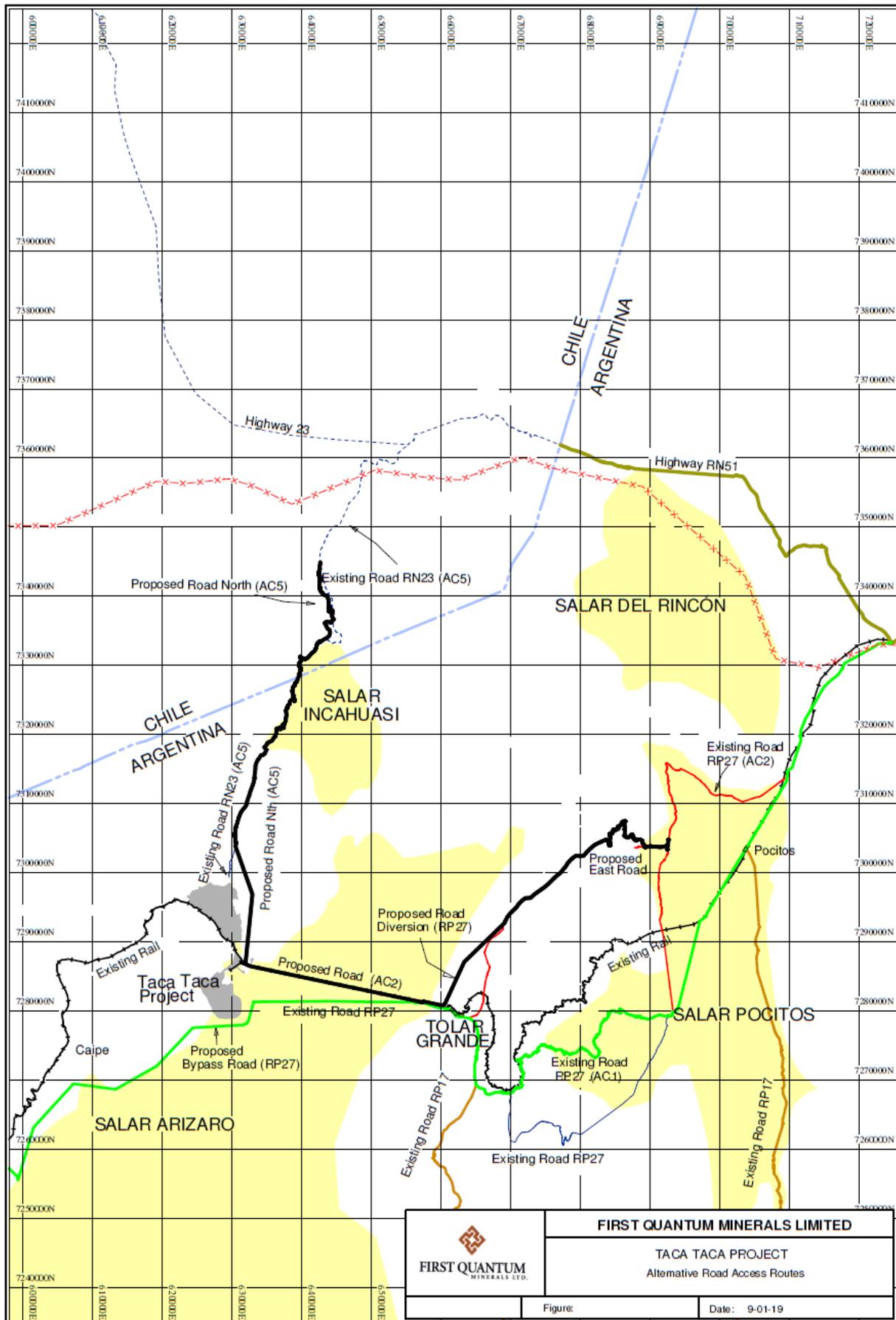


Figure 19-5 Road access route AC 2 showing detail in area of required new construction

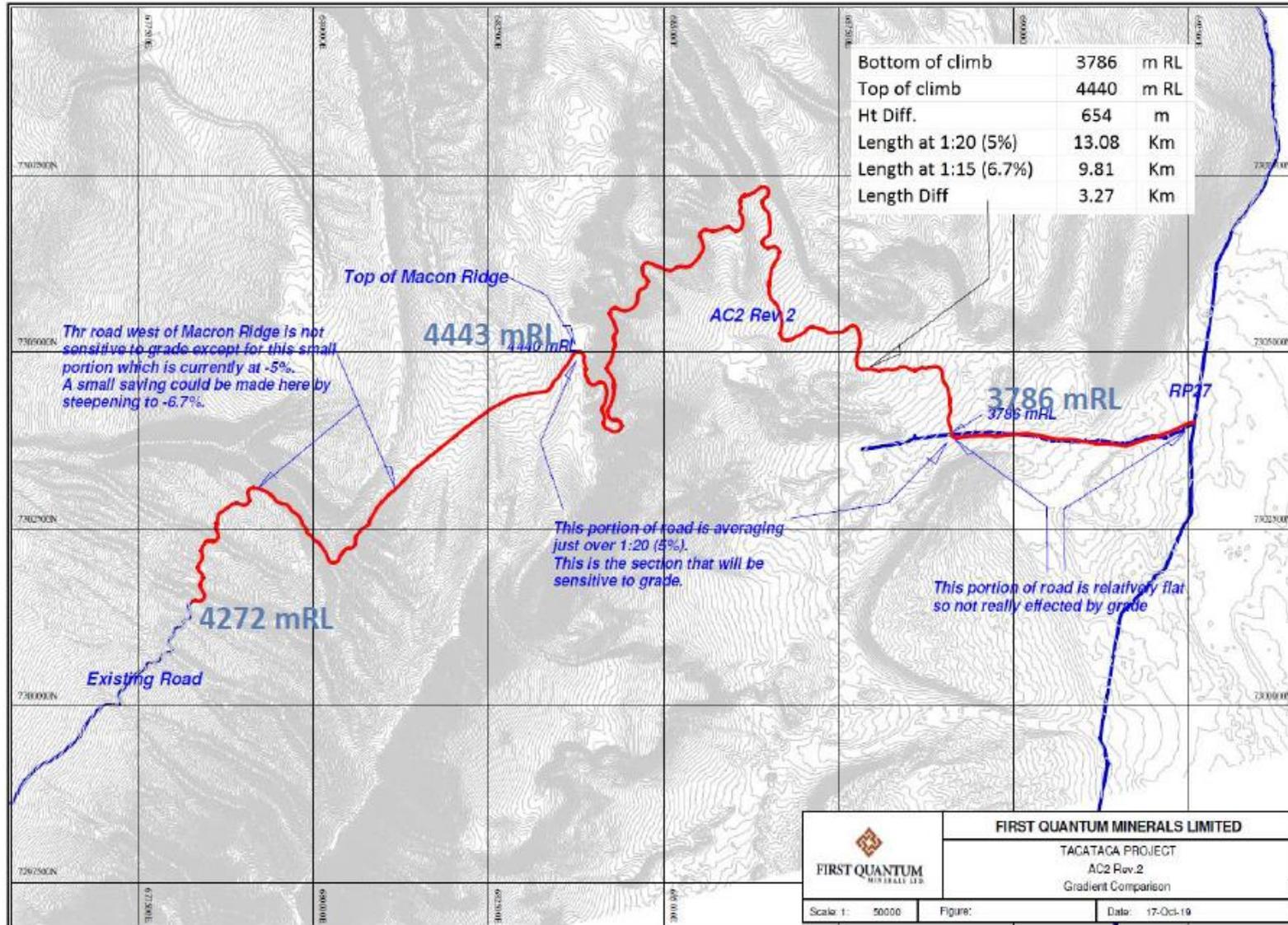
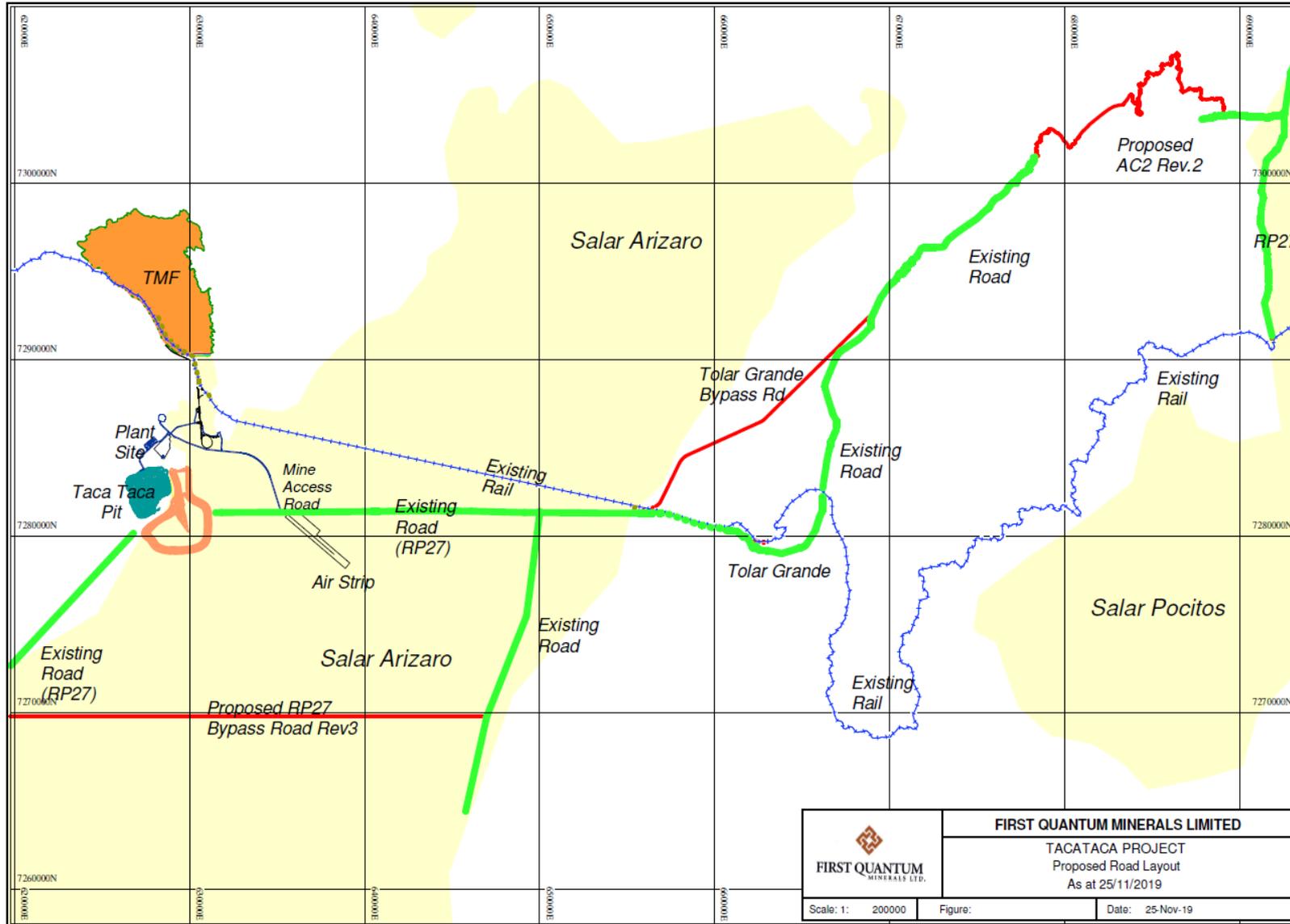


Figure 19-6 Road access route AC 2 showing overall layout inclusive of new road segments and bypasses



19.6 Rail access

A report by *TFP Construcciones SRL* (TFP, 2013) considered the condition of the existing railway line (plus available locomotives and rolling stock) and its suitability for transporting construction materials and consumables into the Project, and for transporting concentrate from the Project to a coastal port.

The main conclusions from the TFP report were that:

1. Substantial reconstruction of the line between Salta and Taca Taca would be required for timely rail transport of construction and consumable items.
2. With tunnels, tight radius curves, zigzag switchbacks and old bridges of restricted load capacity, this section of track would be of limited logistical benefit for the transport of concentrate product; it could be of benefit for the railing of construction equipment and materials, and for operations consumables.
3. In terms of concentrate transport, the preferred rail route into and from the Project would be from Taca Taca to Socompa (requiring 40.5 km of rehabilitation along the 134.4 km length of track), and thence to Mejillones, Chile.

Consistent with the TFP recommendations, Ausenco (May, 2016) noted that the distance between the Project site and the port of Mejillones in Chile is about 538 km compared with about 1,800 km to a coastal port in Argentina. Ausenco also noted that the Pacific ports are capable of handling bulk mining products, whereas the Atlantic ports are more suited to bulk grain handling.

In late 2018, a specialist railway engineering consultant, *Auraxis SA*, produced a preliminary rail logistics assessment for the Project (*Auraxis*, 2018). This assessment included:

- a review of the conditions of the existing railway lines and related infrastructure (confirming the TFP information)
- an opinion on the feasibility of rail freight logistics
- a review of the prevailing legal and contractual circumstances for potential rail operations to and from the proposed Project

19.6.1 Upgrade scenarios

Auraxis (2018) opined that the existing rail formation between Taca Taca and Mejillones would not be safely operable, in its current condition, for the volume of concentrate haulage traffic that is envisaged for the Project. This is in respect of:

- limitations on the track itself (e.g. aged track, light rail weight, inadequate ballasting, wooden sleepers, adverse grades and curvature, rock fall hazards, extreme weather conditions and lack of maintenance),
- running conditions (e.g. speed restrictions, rolling stock with limited capacity), and
- operability (short length trains, lack of automated signalling)

Auraxis (2018) described two potential upgrade scenarios:

1. A scenario involving the upgrades required for operation of a 16 tonne/axle system, specifically:
 - a) retaining the existing track with allowable axle loadings of 16 tonne/axle in Chile and 17.5 tonne/axle in Argentina
 - b) replacing 30% of the rails and 50% of the sleepers along 538 km of track
 - c) stone ballasting along the entire length
2. A scenario involving the upgrades required for operation of a 20 tonne/axle system, and involving:

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- a) replacing 100% of the rails to 46 kg/m weight
- b) replacing 100% of the sleepers with concrete ones
- c) stone ballasting along the entire length
- d) improving the alignment and flattening the curves

In the ledger of updated capital costs in Item 21, the rail infrastructure cost for the 16 tonne/axle scenario is assumed to be borne by the Company rather than by each of the rail network owners in Argentina and Chile. Subject to negotiations, it may transpire that the cost of these upgrades could instead be factored into the concentrate and freight carrying costs imposed by these owners on the Project.

Further work will be required during the on-going engineering phase to develop these scenarios and enter into commercial negotiations with the rail network owners.

19.6.2 Rail deviation

From the TSF design information in Item 18.10:

- the salar surface is at 3,470 mRL (assumed horizontal)
- the existing railway line is at 3,471 mRL rising to 3,535 mRL adjacent to the western side of the TSF
- at Year 13, the volume of tailings would be 530 Mm³, with the tailings level rising to 3,510 mRL

On this basis, it is apparent that the existing railway formation could be used for at least ten to fifteen years before needing to be deviated to a higher level. When the deviation becomes necessary, a railway embankment would be required to connect the high level deviation to the existing track in the proximity of Taca Taca station. This arrangement is shown on Figure 19-7 relative to tailings deposition modelled extents (Hillerton, 2019).

Further work will be carried out during the engineering phase to evaluate deferral of the railway deviation. It remains to be determined whether the tailings deposition arrangement could be practically and economically modified such that the existing railway line is not impacted at all.

19.6.3 Railway rolling stock

From preliminary calculations for the 16 tonne/axle scenario, *Auraxis* (2018) determined that the Project would require:

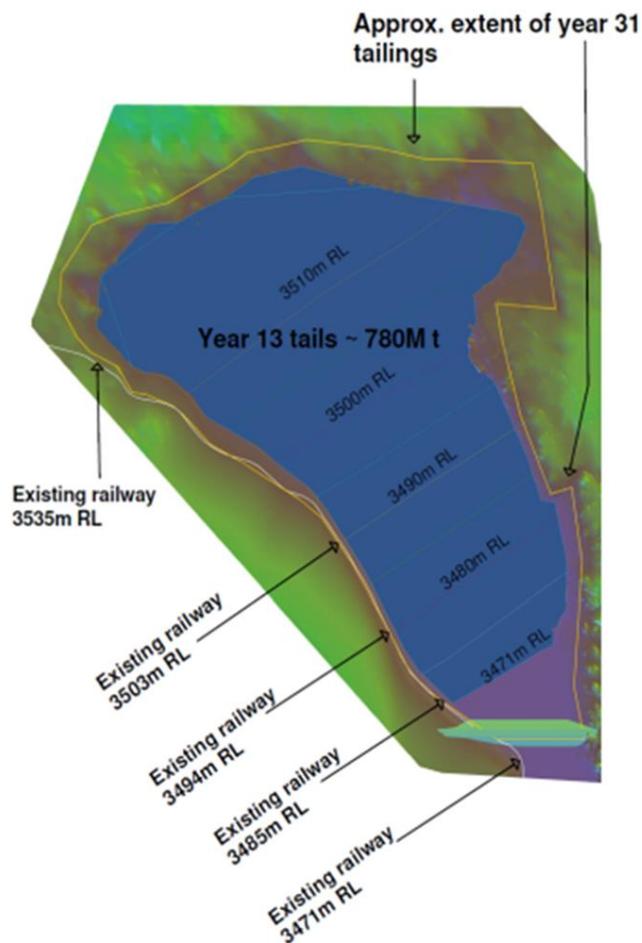
- 13 x locomotives (of 16 tonne/axle size and with AC traction technology), plus additional locomotives for shunting
 - the tare weight could not exceed 96 tonne
- 172 x flat wagons
 - either bottom discharge / bulk container tipping type, or alternatively
- 336 x half height containers (rotainers) for concentrate freight

The *Auraxis* estimated cost for this rolling stock is included in the Project capital cost estimates (Item 21).

19.6.4 Rail operations framework

Given the potential complexity of rail operations and upgrades along rail corridors owned by separate stakeholders (ADIFSE, *Ferronor* and FCAB), *Auraxis* (2018) recommended that a new third-party operating entity be created, e.g. 'Taca Rail Co'.

Figure 19-7 Rail deviation around the TSF; plan showing tailings extent at Year 13 relative to the existing rail formation



19.7 Project port

Chile has many major port facilities in operation and has experience in exporting copper products, including concentrates. The preferred export option for the Project would be via ports in or near to the city of Antofagasta. In particular, Mejillones Bay offers a sheltered bay with deep water close to shore and has become the preferred port of shipping for many industrial imports and exports. Mejillones is well connected with the rest of the region, with access to the main highways and railway, and it is 60 km from Antofagasta, the main city of the region.

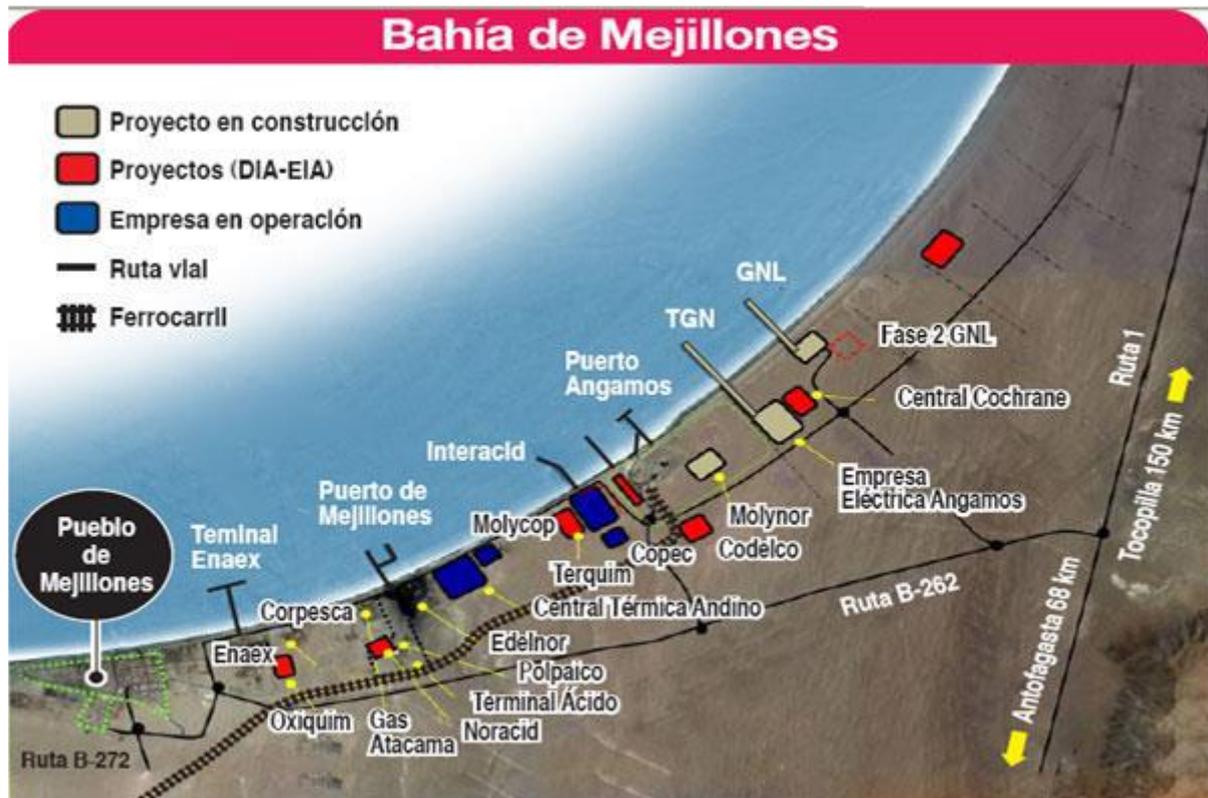
A reconnaissance visit to potential ports in Mejillones Bay was carried out by Company personnel in October 2018 (FQM, 2018). Two potential concentrate export ports were visited, and further possibilities were identified from ensuing discussions and investigations. Figure 19-8 shows the location of ports on Mejillones Bay, north of Antofagasta.

In the ledger of Project capital cost estimates in Item 21, an estimated cost is included for upgrade or expansion of one of the several identified potential Project ports.

19.7.1 Concentrate handling facilities

Auraxis (2018) observed that containerisation of copper concentrate and the use of half height containers on railway wagons has become common in the region. These particular containers (referred to as rotainers) offer the flexibility of being able to be carried on road trucks or rail wagons. It is anticipated that the Project would utilise containerised transport from the Project to the port.

Figure 19-8 Ports on Mejillones Bay



19.8 Mining facilities

The mining facilities include the mobile equipment workshops (i.e. the mine services area or MSA), and the explosives manufacturing facility. Figure 19-9 shows the planned locations of these facilities.

19.8.1 Maintenance workshops and mine services area (MSA)

Several maintenance shops are envisaged for the Project in addition to a mine equipment maintenance shop and light vehicle maintenance shop (the MSA). Figure 18-10 shows a concept layout for the MSA.

The MSA would include the heavy equipment workshop, light vehicle workshop, mine maintenance personnel offices, tire shop, wash down bay, water services, refuelling station, go-line and mine control facilities for the mining fleet. The MSA building would include mine maintenance and some administrative offices including the following; reception, offices, cubicles / workstation areas, conference room, store, copy/PABX, and main operations area kitchen. Potable water, raw water and firefighting water services would be available in the MSA.

To support the process facilities, a mechanical and electrical equipment maintenance shop would be provided for maintenance and rebuilding of equipment. The maintenance shops would be located in close proximity to the process facilities. The mechanical and electrical shops would have personnel and equipment for rebuilding equipment such as pumps and motors. The maintenance shops would include the following:

- electrical shop
- mechanical / machine shop
- welding shop
- light vehicle maintenance shop

Figure 19-9 Site layout plan showing location of mining facilities

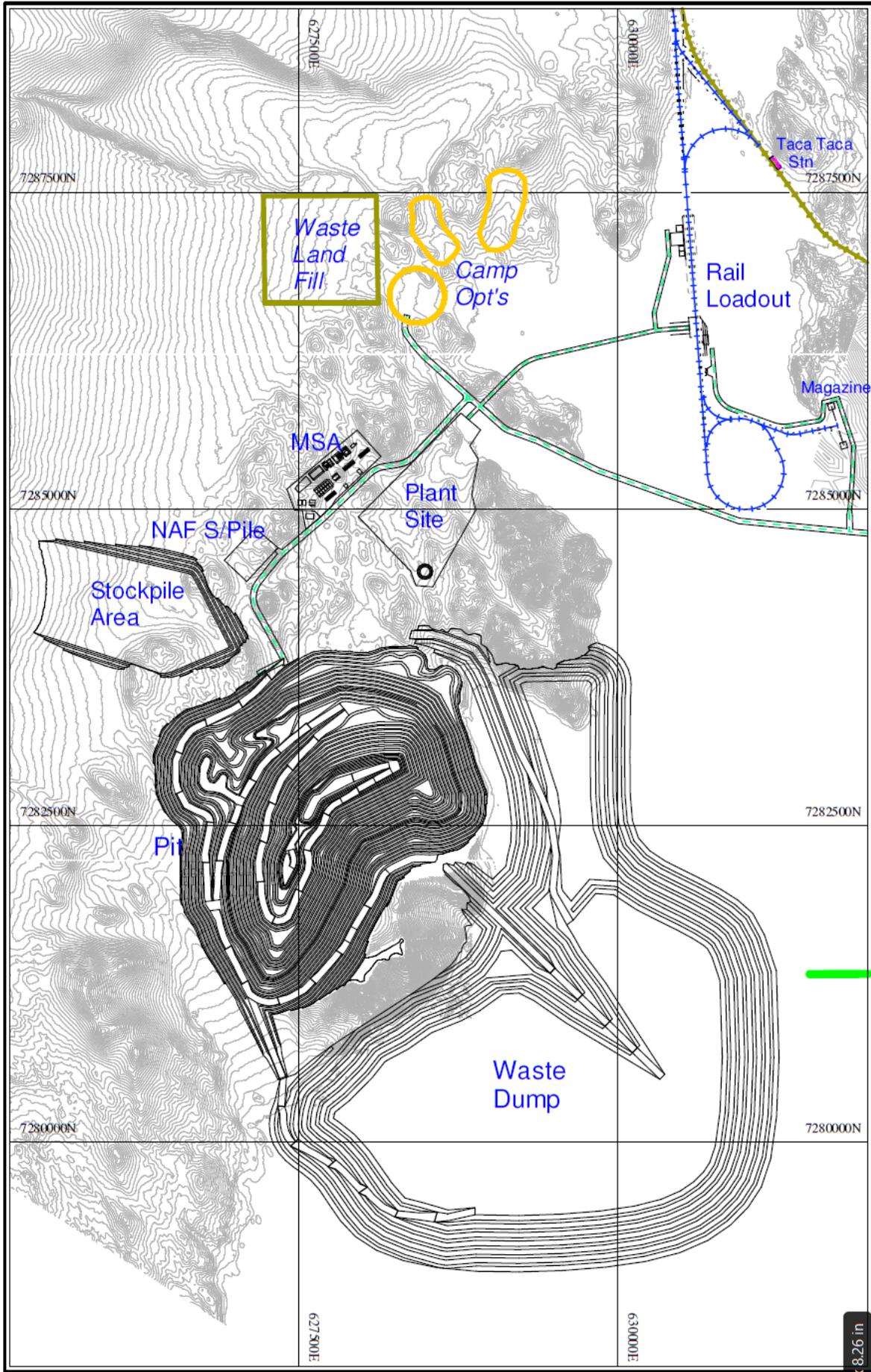
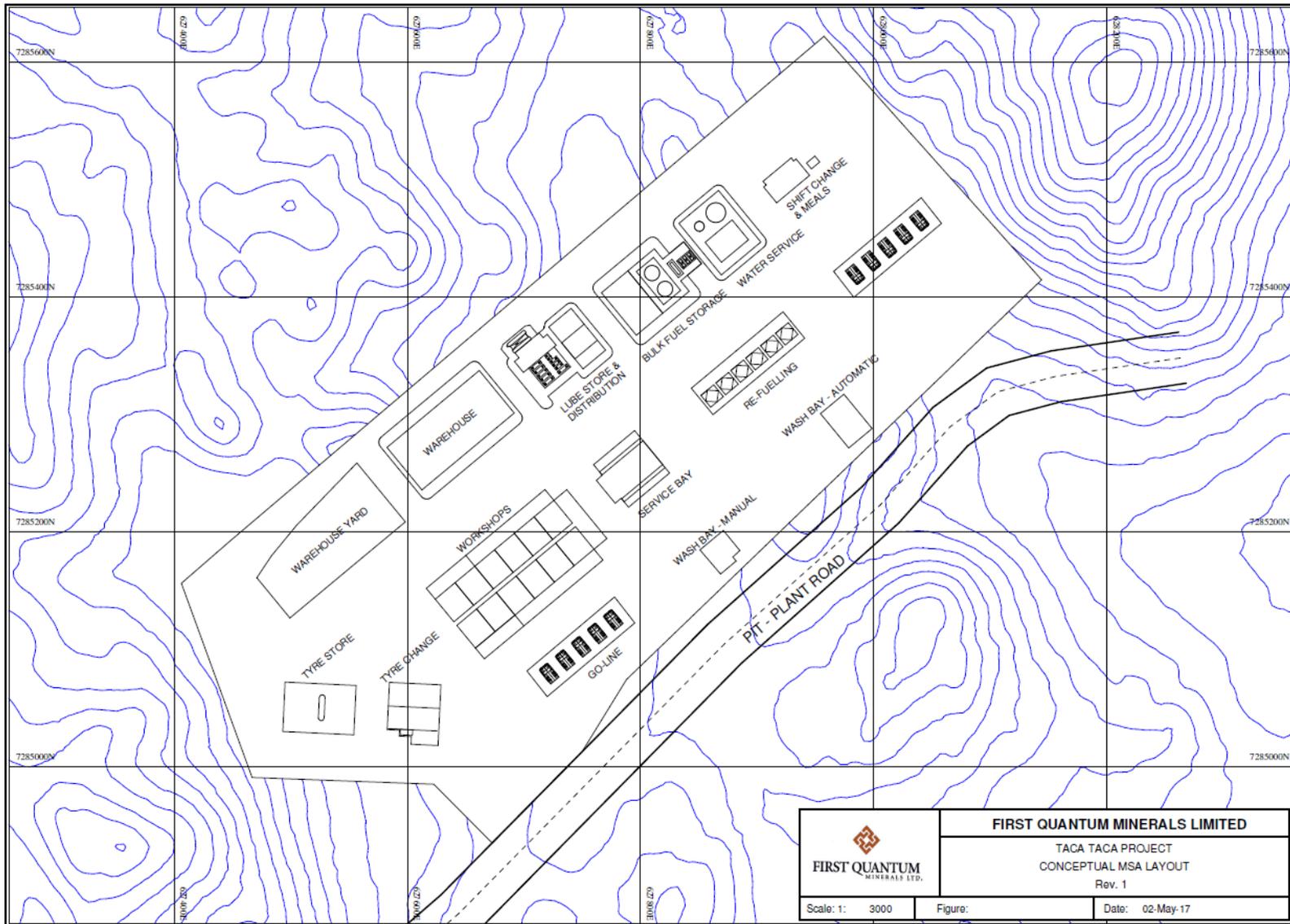


Figure 19-10 Layout of the MSA



19.8.2 Explosives manufacturing facility

A bulk explosives manufacturing facility site is envisaged at a location shown on Figure 19-9, opposite a suitably sized magazine. The separation distance between the magazine and the manufacturing facility would be 500 m to 750 m, to meet storage regulations.

19.8.3 Quarries

There could be a need for the development of local quarries for use in both the construction and operations phases (for supplementary TSF embankment material, blasthole stemming material, etc.), although specific locations have yet to be identified.

The potential location and extent of suitable rock quarries will be examined in the next phase of engineering. It is likely that an amount of construction rock would come from the abundant waste mined during the Project pre-strip. The base case plan does not require supplementary rock for TSF embankment lifts, as this is intended to be accomplished using cyclone tailings. No quarry rock is anticipated for blasthole stemming, as mined waste is intended to be crushed and sized to use for stemming material.

Some gravel regolith in the Project area may be suitable for road building and other construction purposes.

19.9 Plant site and administration facilities

The preferred siting for the processing plant is to the north east and adjacent to the open pit. The location with respect to the terrain is shielded from prevailing winds emanating from the north west at speeds up to 23.2 m/s (83.5 km/h). On approach from the east, the plant site would also be partially shielded from view by the topography and by the waste dump on the Salar de Arizaro.

The administration offices and related infrastructure are proposed to be sited to the northwest (i.e. upwind) of the pit and the plant area.

19.9.1 Administration buildings

An operations administration building would be located near to the entry to the Project site. This building would provide private offices and common office areas for personnel, in addition to common areas for meetings, filing rooms and kitchens.

Offices within the administration building would accommodate management personnel, as well as technical and operations superintendents and area managers, and all supporting professionals such as accountants, safety and loss prevention personnel, clerical and secretarial staff.

Facilities for training would be included, consisting of large and medium sized conference rooms and visitor offices. The administration building would also accommodate a medical clinic to provide emergency treatment to all site personnel. Emergency medical equipment would be located within close proximity of the medical clinic, including ambulance, rescue and firefighting equipment.

During the construction phase, modular type units would be used as administration and technical offices, for a medical clinic and for accommodating related construction services. These units could be removed from the site once the permanent facilities are constructed, or alternately, continue to be used for the same purpose during operations.

19.9.2 Metallurgical laboratory

A metallurgical laboratory would be located within the plant area, adjacent to the process plant building. The laboratory building would be of steel construction with roof and wall cladding, and would be fully

equipped with all analytical and sample preparation equipment including dust and fume hoods and associated air handling treatment systems.

Waste streams from the laboratory would be pumped to the process plant where they would join the main process streams and be neutralised.

19.9.3 Process water storage

A process water storage pond could be located in a small valley immediately to the south east of the processing plant. Subject to further investigation during the engineering phase, and if required, this potential pond could be used for the storage of fresh water or brine. Evaporation minimisation and retention durations will be important considerations to avoid fresh water losses or salt encrustation in the event of brine storage.

The valley impoundment could be 1 million m³ behind a 30 m high, 200 m wide embankment. At 15 m height, the storage capacity would be 200,000 m³, or at 10 m height, 70,000 m³ of capacity.

19.10 Tailings storage facility

The preferred TSF site is within a natural impoundment provided by the Salar de Taca Taca, which is an embayment of the Salar de Arizaro. This site, located to the north of the proposed processing plant, was selected for the following primary reasons:

1. A TSF at this site has the shortest and lowest embankment height of the several alternatives considered, and is located at the lowest elevation in the immediate Project area.
2. The preferred site presents the least risk of embankment failure, and the least potential impact in the event of embankment failure. This is because there is no major down-gradient landform, community or infrastructure (other than the railway line) for the tailings to gravitate to in the event of an embankment failure.
3. The preferred TSF site is on a salar characterised by brine saturated halite and transported sediments in hyper-saline conditions. As such, there is no risk of contaminating a source of fresh, potable water.
4. At the narrow southern end of the salar, the granite base rises and forms a natural barrier to water flow into the Salar de Arizaro.

In the proposed location, the TSF would be almost entirely enclosed by the natural land mass on the north, west and east sides, requiring only a relatively low height, short length and unobtrusive embankment at the entrance to the salar, on the south side. The ultimate storage capacity of the TSF at this site is 1,405 Mm³, with a final northern elevation at 3,540 mRL (approximately 70 m above the salar surface). Due to tails beaching, the final southern elevation at the embankment position will be at 3,496 mRL (approximately 25 m above the salar surface).

The Salar de Taca Taca is essentially a basin of accumulated sedimentary and evaporitic deposits, of varying permeability, on top of granite bedrock.

Figure 19-11 shows a view looking north over the proposed site of the TSF on the Salar de Taca Taca.

Figure 19-11 View looking north across the Salar de Taca Taca (Cerro Aracar in the background)



19.10.1 Site characteristics

Hydrogeology

Observations at the TSF site indicate that the Salar de Taca Taca is for much of its extent, saturated with brine, from less than a metre below surface.

Hydrology

A map showing the 450 km² extents of the hydrographic basin surrounding the Salar de Taca Taca is shown in Figure 19-12. The hydraulic gradient in this basin is from north to south; Montgomery & Associates (M&A, May 2020) advise of the following hydrological conditions:

Inflows:

- recharge from precipitation = 66.5 L/s

Outflows:

- evaporation from the salar and surrounds = 66.15 L/s
- below surface lateral discharge = 0.4 L/s

Figure 19-12 Hydrographic basin surrounding the Salar de Taca Taca (M&A, May 2020)



There are no major surface water flows within the preferred TSF site. However, there is some minor water inflow to the upper reaches of the salar via the alluvial gravels near the north eastern corner¹³.

Water flowing into the salar from the north, and/or collecting from precipitation is either evaporated or stored within the salar basin. At the narrow southern end of the salar, the granite base rises and forms a natural barrier to water flow into the Salar de Arizaro.

Vegetation

There is no vegetation on the salar surface and the surrounding area is very sparsely vegetated.

Surficial soils

There are no surficial soils on the salar surface. The salar deposits are essentially lacustrine sediments overlain with a saline crust (typically halite and gypsum).

19.10.2 Storage concept and design basis

The TSF will be an upstream raised structure, with the retaining embankment built initially from available NAF mine waste and if required, locally quarried material, and thereafter followed by continuous upstream construction using cycloned tailings.

¹³ This particular corner of the salar could be bunded-off to prevent interaction with the tailings.

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Hillerton Consulting Ltd (Hillerton, 2019) produced a conceptual design for the TSF in May 2019, the basis for which is listed in Table 19-5¹⁴.

The eventual upstream raised embankment would be formed from cycloned coarse tailings, with the fine cyclone fraction being directed towards the centre of the TSF. It should be noted that although cyclone tailings for the embankment lifts has been assumed for the Project base case, this will be reviewed at a later phase of engineering to determine whether cyclone tailings are required or if the embankment may be raised using spigotted tailings lifts.

Table 19-5 TSF design parameters (Hillerton, 2019)

Item	Value	Comment
Tailings disposal method	Conventional slurry tailings (thickened)	Assumed
Annual tailings production		CASA
Year 1	30 Mt	
Year 2	42 Mt	
Year 3	51 Mt	
Year 4	57 Mt	
Years 5 to 31	60 Mtpa	
Total Tailings production	1,800 Mt	CASA
Tailings slurry concentration	55% to 60%	CASA
Average tailings stored density	1.25 t/m ³	Hillerton assumption
Average tailings beach slope	1:150	Hillerton assumption
Max tailings particle sizing	150 µm	CASA
Salar groundwater quality	Brine, TDS>300,000 ppm	CASA - TBC
Seepage control measures	Not Required	Hillerton assumption - TBC
Tails deposition arrangement	Multiple spigot discharge	Hillerton assumption
Seismic coefficient (pga)	0.18g	INPRES

19.10.3 TSF operations

The following information is reproduced from the Hillerton (2019) report ...

- *Tailings will be pumped, through a HDPE pipeline(s), from the process plant to the northern part of the facility. Initially, the pipeline(s) will extend some 8 km along the western side of the facility and deposition will occur from the north-west corner¹⁵. During the early years of facility development, the pipeline(s) will be gradually extended eastwards, to facilitate tailings deposition from all along the northern perimeter of the facility.*
- *Tailings will be discharged via spigots located along the pipeline, creating a tailings surface that slopes downwards to the south. The topography of the salar is virtually flat with a fall of approximately 1 m over 7 km and this exceptionally shallow gradient will limit the spread of tailings from the deposition*

¹⁴ The annual tailings production rate to Year 9 has since been revised. The grind size was subsequently revised to 80% passing 180 µm.

¹⁵ Further work on the spigotting arrangement will be carried out during the continuing engineering phase, to assess the viability of prolonging the life of the existing railway line immediately to the west of the TSF. It is conceivable that deposition could commence from the north-east corner of the TSF.

point. With ongoing tailings deposition, the depth of tailings at the northern end of the facility will increase, and the edge of the tailings beach will gradually migrate southwards.

Area requirements

The recognised ratio linking the beaching (disposal) area of the storage to the tonnage of tailings production is 40 ha/Mt/annum. With an insitu dry density of 1.25 t/m³, this ratio would result in a rate of rise of around 1.8 m/annum. For a throughput of 60 Mtpa, a beaching area of approximately 2,400 ha could be expected. The actual area available for tailings deposition is 3,650 ha, which is in excess of the minimum required.

Tailings pipelines and spigots

Two pipelines would be installed from the concentrator to the TSF, both operational. These lines would direct pump tailings to different sections of the TSF for controlled distribution of tailings into the facility.

Embankment building would be undertaken using cyclones on the tailing to provide coarse particles for the wall, with fine cyclone overflow being directed into the centre of the TSF.

Around the TSF perimeter, tailings would be spigotted into the facility through 150 mm diameter spigot lines spaced at 20 m apart. Up to twenty-four of these spigots may be used at any one time, to direct tailings to specific areas of the facility. 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.

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 peripheral tailings lines would be installed approximately 3 m to 5 m higher than the beach of the TSF to avoid being inundated by the cones of coarser material. It is anticipated that the peripheral tailings line would require to be raised onto higher ground every three years.

The discharge of tailings around the TSF would be managed to drive the decant pool to the north western 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.

Tailings return water

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 at all times and tailings return water (decant water) would be used in the plant in preference to make-up water from borefields.

Under these circumstances, and as the tailings beaches develop, a pond of water could be expected to develop at a low point(s) on the TSF surface. Submersible pumps installed on barges would then remove water from ponds formed within the respective cells in the TSF and return the water to a central location from where booster pumps would pump the water to the process water pond in the plant area.

However, at this stage of Project engineering it is not fully understood whether or not a decant pool would develop, or whether any supernatant liquid would seep or evaporate. Operation of floating pontoons or barges may not be feasible. Methods for the remainder of excess water from the TSF will be investigated during the continuing engineering phase. The Project water balance currently assumes that there will be no tailings water return.

Initial decant arrangements

Further to the above, and again subject to further work during the Project engineering phase, 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 the water before it evaporates.

19.10.4 TSF embankment

Although the tailings surface is not projected to reach the southern extents of the TSF for some years (i.e. more than 13 years on an assumed gradient of 1 : 150), a 4 m high starter embankment will be constructed across the entrance to the salar embayment as part of the development of the initial facility. This embankment will be upstream raised during the life of the TSF (Figure 19-13).

The following information is reproduced from the Hillerton (2019) report ...

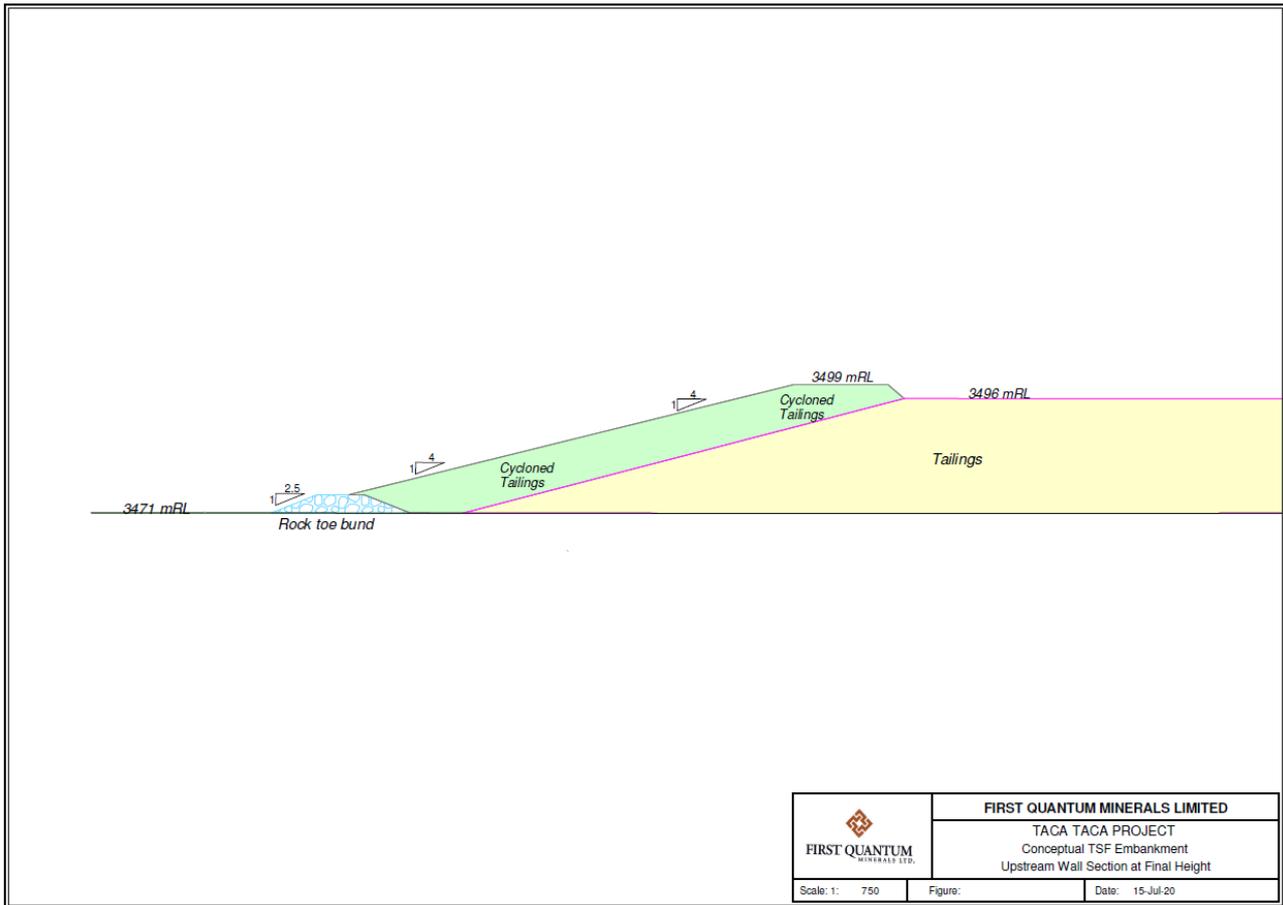
The embankment will be a homogeneous rockfill structure, constructed with material from open pit stripping operations or local borrow. The fill material will comprise clean sound durable rock, well graded, with a maximum particle size of 350 mm and not more than 5% passing 75 µm. The embankment crest will be 10 m wide with upstream and downstream slopes of 1V:2.5H. Construction of the 4 m high starter embankment will require the placement of approximately 60,000 m³ of rock-fill.

The embankment will be raised progressively, later in the life of the facility, using upstream construction methods and cycloned tailings. The downstream slope will be constructed at 1V:3.5H, with 6 m benches at 8 m height intervals, resulting in an overall slope of approximately slope of 1V:4.0H.

A 2 m wide erosion protection layer could be placed on the downstream slope, composed of mine waste rock.

Figure 19-13 shows the final elevation of the tailings at the embankment position as 3,496 mRL (25 m above the salar surface). An additional 3 m of cycloned tailings is shown as a freeboard provision.

Figure 19-13 Cross section through proposed TSF embankment



Deposition modelling and rates of rise

Hillerton (2019) carried out deposition modelling to simulate the filling of the TSF. Modelling was carried out using Rift TD, an advanced three-dimensional digital terrain modelling software package, specifically developed to model tailings deposition.

Table 19-6 is an extrapolation from the 2019 modelling to account for a subsequently revised processing schedule and the associated rate of tailings rise and required upstream embankment raising.

The ultimate TSF capacity at an embankment height of 25 m is 1,405 Mm³ (at 1.25 t/m³ tailings density). An allowance has been made in the modelling for the loss of capacity in the tailings dish, due to an assumed beach slope of 1%.

Table 19-6 TSF tails elevations (derived from modelling by Hillerton, 2019)

Year	Plant Feed		Tailings at 1.25 t/m ³		Max tailings elev. RL (m)	TSF embankment	
	Annual (Mt)	Cumulative (Mt)	Volume (Mm ³)	Cum Vol (Mm ³)		Height (m)	RL (m)
-3 to -1						4.0	3,475
1	30	30	24	24			
2	40	70	32	56			
3	40	110	32	88	3,490	4.0	3,475
4	40	150	32	120	3,490	4.0	3,475
5	40	190	32	152			
6	40	229	32	184	3,500	4.0	3,475
7	50	279	40	223	3,500	4.0	3,475
8	60	339	48	271			
9	60	399	48	319			
10	60	459	48	367			
11	60	519	48	415			
12	60	579	48	463			
13	60	638	48	511			
14	60	698	48	559	3,515	4.0	3,475
15	60	758	48	606	3,515	4.0	3,475
16	60	818	48	654			
17	60	878	48	702			
18	60	938	48	750		5.0	3,476
19	60	998	48	798			
20	60	1,057	48	846			
21	60	1,117	48	894	3,525	6.0	3,477
22	60	1,177	48	942		7.9	3,479
23	60	1,237	48	990		9.8	3,481
24	60	1,297	48	1,037		11.7	3,483
25	60	1,357	48	1,085		13.6	3,485
26	60	1,416	48	1,133		15.5	3,487
27	60	1,476	48	1,181		17.4	3,488
28	60	1,536	48	1,229		19.3	3,490
29	60	1,596	48	1,277		21.2	3,492
30	60	1,656	48	1,325		23.1	3,494
31	60	1,716	48	1,373	3,540	25.0	3,496
32	43	1,759	34	1,405	3,540	25.0	3,496
					freeboard	28.0	3,499

Note: The figures shown in red are extrapolated from the Hillerton modelling. The intermediate embankment heights shown in italics are estimated.

19.11 Camp

A construction camp with a capacity for approximately 4,300 people would be established to accommodate the construction workforce. This camp would be located near to the concentrator construction area and would be a full service facility including recreation areas and providing meals and laundry.

Subject to further evaluation during the engineering phase, the preliminary location is approximately 5 km from the entry to the Project site, where shown on Figure 19-14. Several potential sites are shown in close proximity to each other, some more sheltered from the prevailing wind than others.

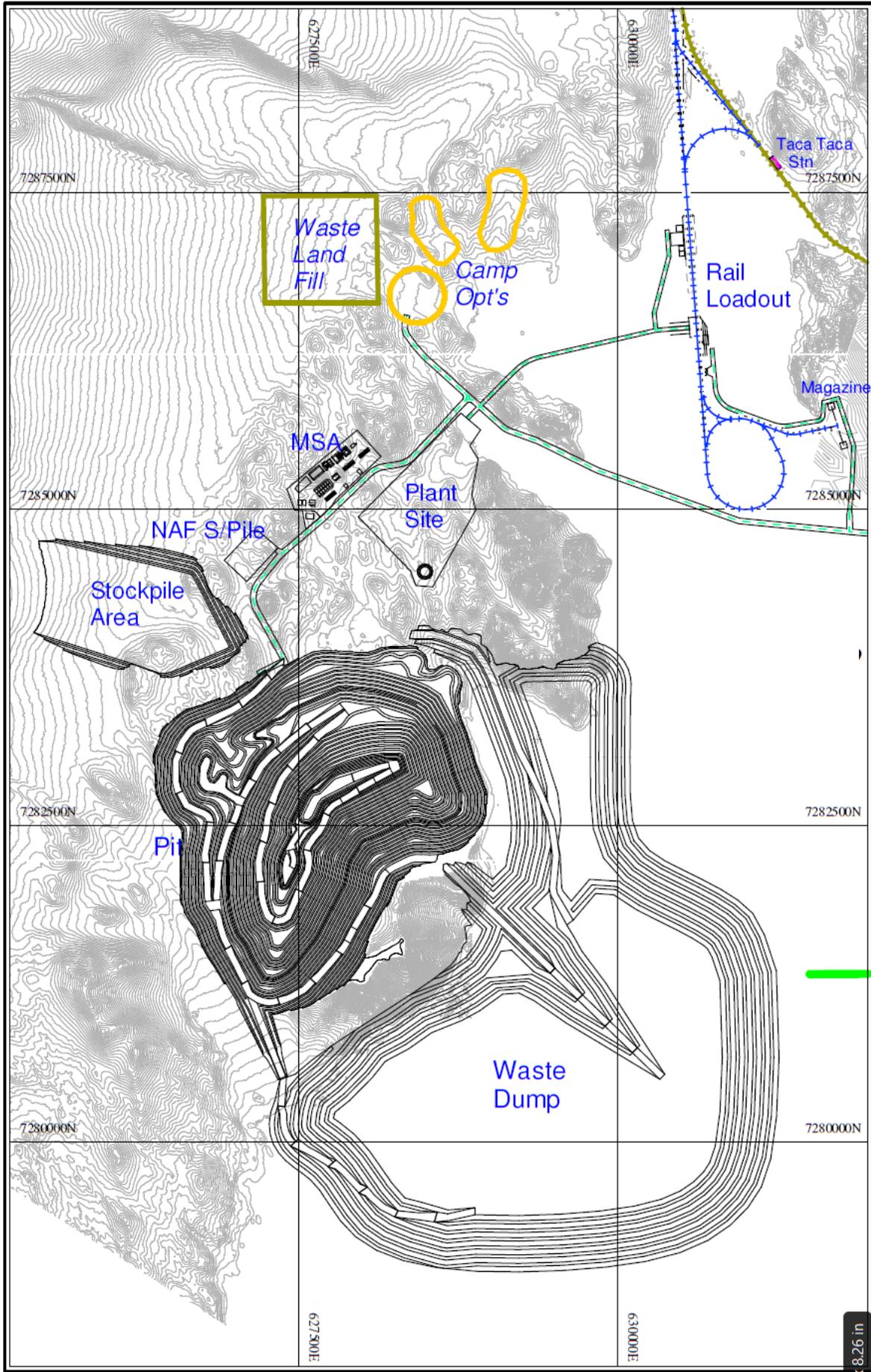
At the end of the construction phase, it is proposed that much of this initial camp complex could be converted into a permanent facility which is close to the operating mine and plant. The operational manning numbers will be fewer than during the construction phase, hence the permanent camp is expected to have a capacity for approximately 1,900 people.

The accommodation design and layout would be planned in liaison with the local authorities and regulations. The Company will review other appropriate community services and town planning requirements to support the accommodation camp, such as a health clinic.

19.11.1 Waste landfill

GT Ingenieria S.A. (January 2020) completed a domestic waste landfill study from which a preferred location was identified within the square shown on Figure 19-14, close to the conceptual camp site locations.

Figure 19-14 Site layout plan showing conceptual location of camp and waste land fill



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Wastes would be disposed of in an environmentally acceptable manner in accordance with legal requirements to prevent their direct or indirect discharge to the environment. The waste disposal facilities would include:

- a solid waste sorting facility
- hazardous waste storage buildings
- incinerators
- sanitary landfills

Wastes would be recycled wherever practicable. Materials such as tyres and conveyor belting may be buried within the waste dump. Other materials such as lead-acid batteries, fluorescent lamp ballasts, and chemicals would be shipped off site for recycling. The materials would be stored in a solid waste sorting yard at the Project site before being removed from site for recycling or permanent disposal.

Hazardous waste would be stored temporarily at the site in appropriate containers protected by secondary containments, before being sent off site for disposal at special handling facilities. Non-hazardous solid waste that cannot be burned would be disposed of in unlined sanitary landfills, which could include areas of the mine waste dump.

19.12 Other facilities and infrastructure

19.12.1 Concentrate load-out and rail spur lines

A concentrate load-out facility is planned at a site located between the processing plant and the existing Taca Taca railway station. The load-out would be accessed by rail turnout spurs from the existing rail line.

Copper concentrate would be pumped from the main plant to surge tanks located at the load-out facility, where it would be dewatered and conveyed to a covered stockpile. The stockpile would provide approximately four weeks of concentrate storage (80,000 tonnes) to act as surge capacity between production and load-out. Rail cars would be loaded by front end loader. Bagged molybdenum concentrate would be transported from the concentrator to the load-out facility by truck. A shed would provide for one week's production of approximately 140 tonnes of concentrate (70 x two tonne bags).

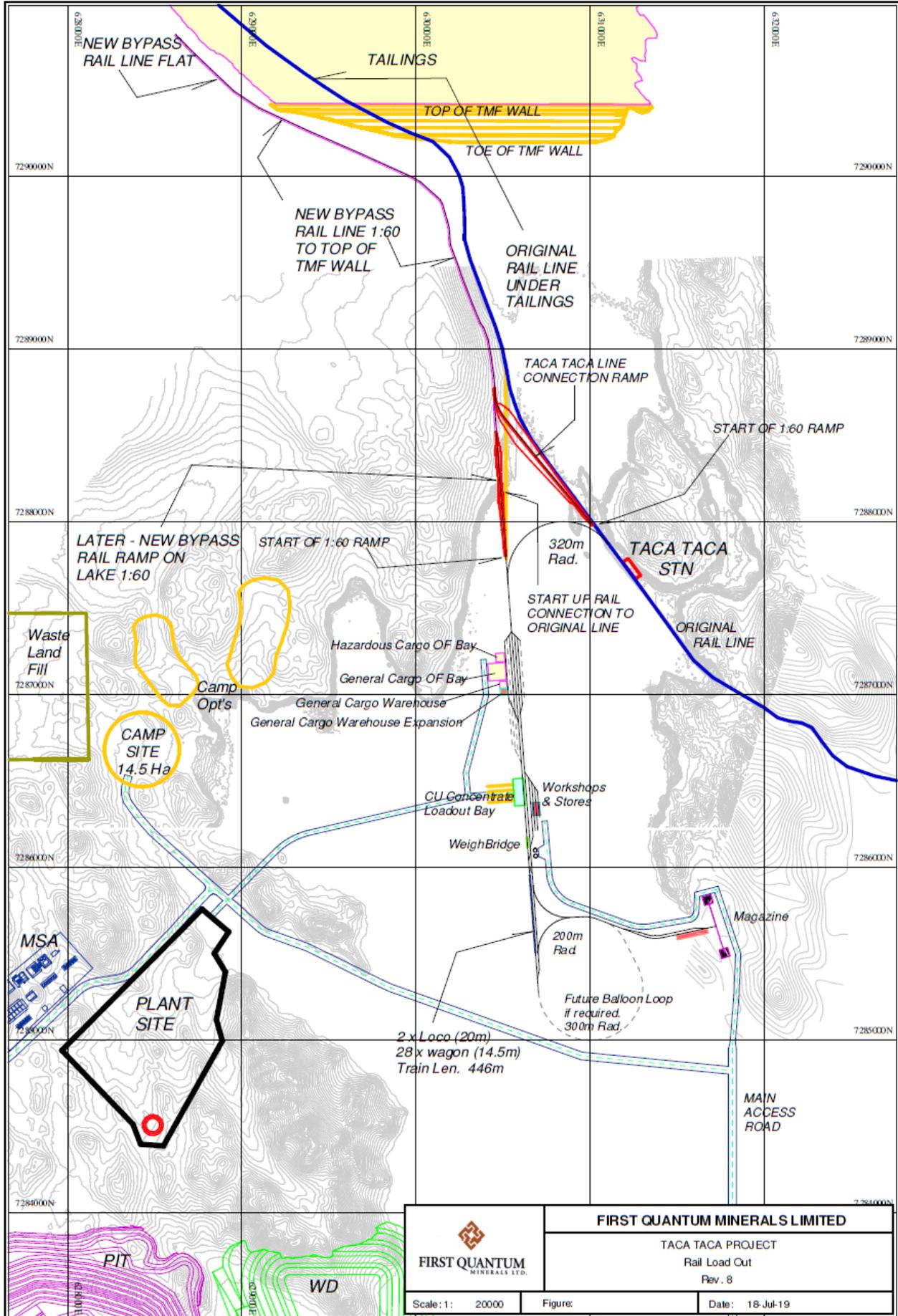
The expected train size is between 25 to 30 cars, each carrying between 45 wmt to 50 wmt of copper concentrate per car, which is expected to be loaded in under two hours. The facility is expected to be able to dispatch between two and three trains per day on a continuous basis.

A design for the load-out has been produced by railway engineering consultants *Auraxis* (2018). This work has involved preliminary engineering and cost estimates for the load-out components shown in Figure 19-15 and including rail sidings for multiple trains, inbound with freight and consumables and outbound with concentrate.

At the outset of the Project, two new spur lines would be required from the existing track to turn-out into the load-out facility. One (eastern) spur line would be required for trains operating along the Argentine sector of track, and another (northern) required for trains going to and from Mejillones, Chile. These are shown on Figure 19-15. The northern spur line is proposed to be part of a complete new section of track required to eventually pass above the TSF on the south western side (shown as a purple line in Figure 19-15). This deviation would require:

- a turn-out onto the main line for a distance of 1,255 m
- a new length of track rising over a distance of 3,300 m at a gradient of 1.7% (1 : 60)
- an embankment connecting the new, higher level length of track to the existing line near to the Taca Taca railway station

Figure 19-15 Rail access into the load-out facility



The eastern spur line would be required, and an initial northern spur line (without an embankment) would be required where shown as an orange line in Figure 19-15.

Beyond the western end of the railway embankment, the deviated route would skirt around the salar before re-joining the original formation at a distance of about 6,200 m. The existing railway formation could be used for a considerable period.

19.12.2 Marshalling yard and warehousing/storage

The marshalling yard at the plant construction site would be a suitably prepared area, contoured, compacted and covered with a locally crushed and screened aggregate / stone, sized appropriately to accommodate storage of delivered materials and equipment during construction.

Modular type offices would be located within each of the marshalling yards to control and monitor the dispensing and issue of all materials, equipment, hand tools and construction consumables. Receiving of deliveries would be conducted by personnel located in the same set of modular offices within each marshalling yard.

Construction equipment and consumables would be stored/warehoused adjacent to each marshalling yard, with site preparation conducted during the same time as the marshalling yard. Similarly, construction equipment would be monitored and controlled by an equipment dispatch office located within each secured and gated fenced area.

19.12.3 Fuel storage and dispensing

The main diesel fuel storage tanks could be located adjacent to the concentrator site and the MSA¹⁶. Fuel would be stored in single-walled, above-ground tanks installed within impermeable bunded enclosures to provide secondary containment in accordance with legal standards.

The diesel fuel tanks would be sized to store approximately two weeks requirements for the mining and process plant operations. Fuel (and lubricants) for the mining operation would be stored in the dispensing/refuelling facilities at the MSA.

The refuelling station would comprise fuel unloading pumps, diesel storage tanks, heavy vehicle fuel dispensing pumps and bowzers, a light vehicle fuel dispensing pump and bowser, and fuel transfer pumps to the plant area. Concrete pads would be provided for fuel unloading and loading vehicles, and storage tanks would be installed in a fully concrete bunded area, with a sump pump which would discharge any spillage or wash-down water to the oil/water separator sump.

The access roads to the refuelling station would be configured such that the travel paths of the heavy and light vehicles do not cross.

19.12.4 Warehouses

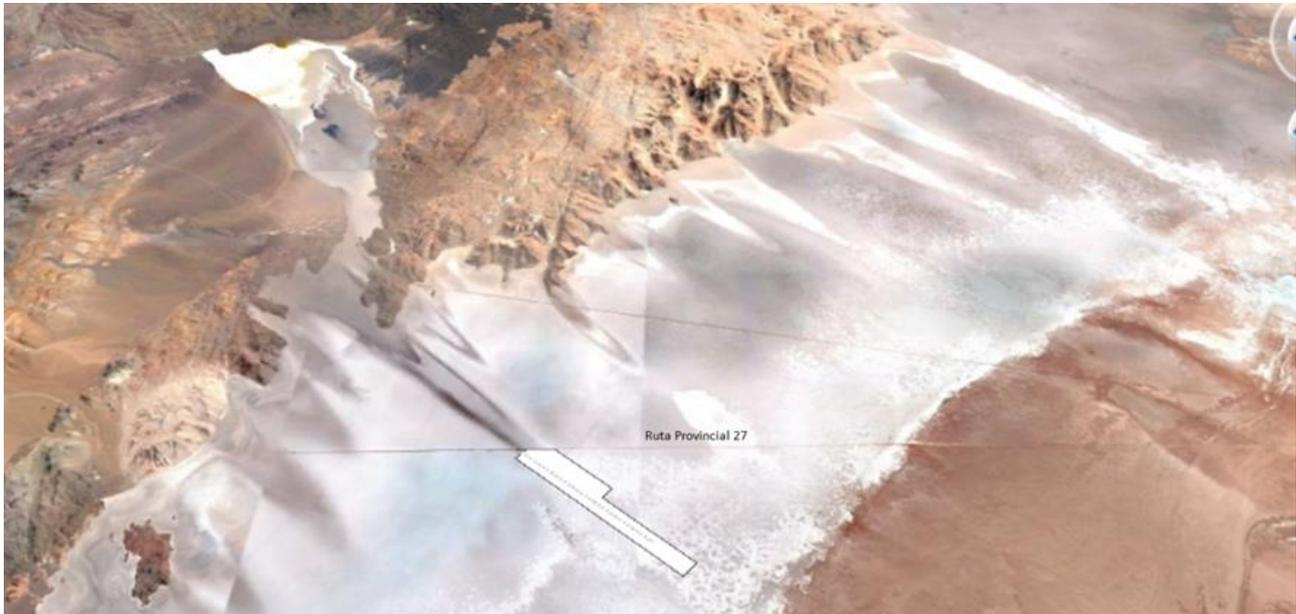
The main warehouse would be a steel construction building with roof and wall cladding. The facility would be complete with warehousing area, receiving and shipping dock and counters, materials dispensing counter, offices for warehousing personnel, small conference room and small kitchen. In addition, outdoor equipment and materials storage areas would be located adjacent to the warehouse. Perimeter security fencing would be installed.

¹⁶ Subject to further work during the Project engineering phase, fuel may be transported into the site by rail. Under these circumstances, the location of the fuel storage tanks may change.

19.12.5 Airstrip

The now defunct former airstrip on the Salar de Arizaro would be replaced to enable small planes (including medical evacuation planes) to land as required. The location of this new airstrip immediately to the east of the mine site is shown in the Figure 19-16 view.

Figure 19-16 Location of airstrip on the Salar de Arizaro



19.12.6 Security

The overall Project site security would be monitored and controlled by the use of guard houses at all entries to the Project site, with additional guard houses located at strategic locations to provide overall monitoring of site activities. In addition to site monitoring at the processing plants, security monitoring would also be conducted at the pit, TSF, the camp and the magazine.

Security fencing would be erected around the process facilities, warehouse, and material storage areas.

19.12.7 Construction site utilities and services

Construction site utilities and services would include power, water, waste treatment and management, communications, fuel storage and dispensing, and fire protection.

Construction power would be provided by diesel generators, sized and specified to be used as emergency generators during operations. The ideal siting of the diesel generators would be in the permanent location for back-up provision during operations, thereby avoiding relocation and providing uninterrupted power supply.

Potable water would be pumped from the regional borefields and treated as required. Initially, water would be reticulated to the accommodation units and into storage facilities for construction consumption. Non-potable water would be sourced from bores on the Salar de Arizaro.

Prior to having permanent waste water treatment facilities in place, the construction facilities would have to use chemical toilets.

Communications services would be established and accessible at all of the construction facilities, including the construction offices and camp. Communications would include telephone service and internet access, in addition to two-way radios and cellular telephones.

19.13 Communications

The Project plans to upgrade the available communications infrastructure utilising a combination of satellite and land based technology.

19.14 Project infrastructure layout

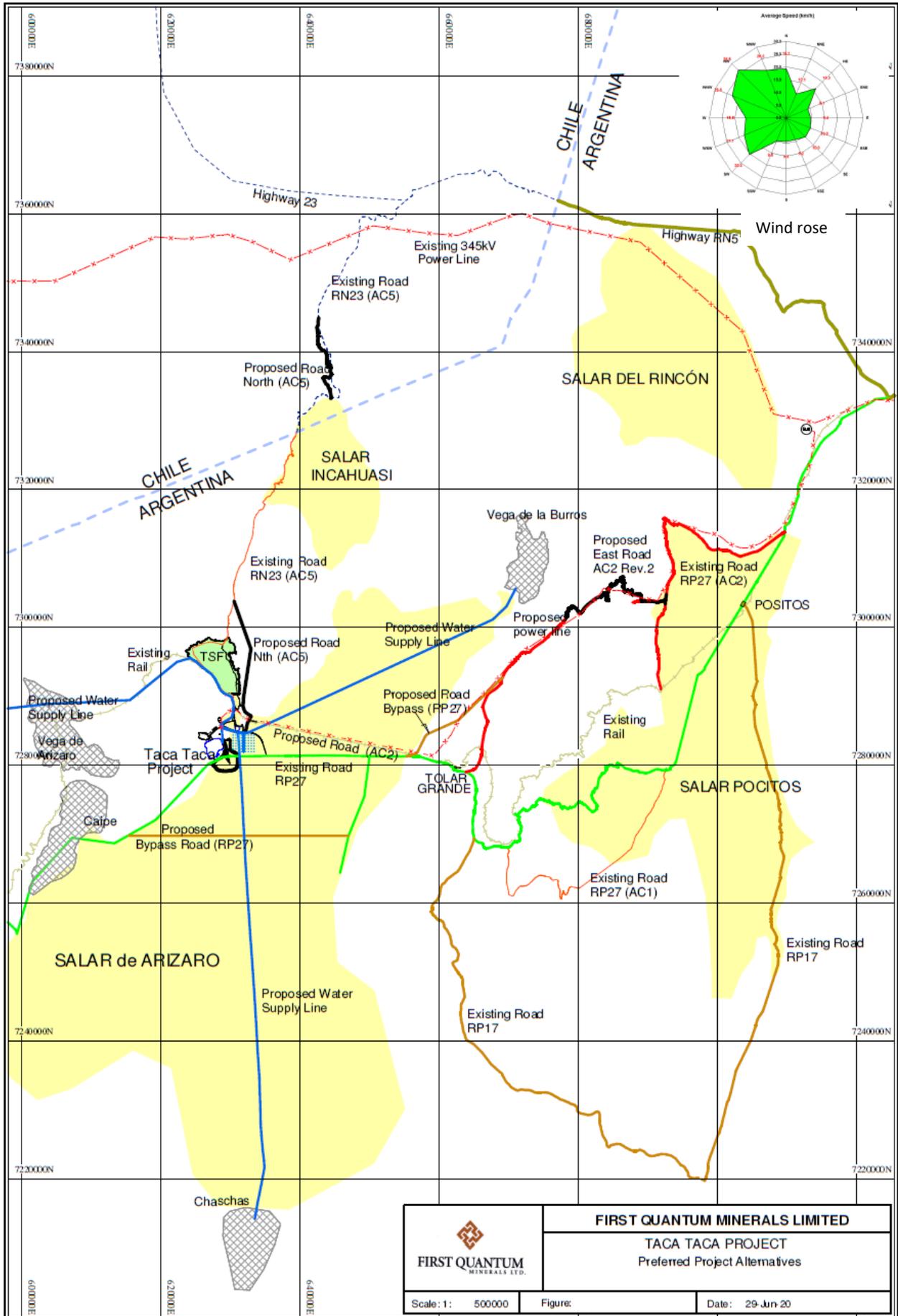
Figure 19-17 is a collation of the infrastructure layout preferences for the Project mining, processing and alternatives. A possible secondary (AC 5) access route shown on this figure will be considered further during the engineering phase.

19.15 Preliminary development timeframe

The current pre-development phase of the Project will see completion of the water supply investigations and confirmation of supply source sustainability. Logistical planning, discussions and negotiations will continue through this phase, whilst the Project and infrastructure ESIA's will also continue through the essential government review and approvals process.

The Project construction phase is expected to proceed over a notional 38 month period, overlapping with the mine pre-stripping and development works.

Figure 19-17 Project infrastructure siting



ITEM 20 MARKET STUDIES AND CONTRACTS

20.1 Product marketability

A marketing study for Taca Taca was completed for Lumina by H&H Metals Corp in 2013 (H&H Metals Corp, 2013). Using contacts with smelters and then recent data from other projects, H&H analysed the following marketing aspects in its report:

- supply and demand for both copper and molybdenum concentrates, including anticipated production from new projects in the construction or planning stages
- smelter capacities, utilisation, and the impacts on TCs, RCs, and payables; ocean freight costs; and future metals pricing

Indications from the H&H marketing study were that, based on Project metallurgical information at the time, the copper product appeared to be a clean standard grade copper concentrate which could be used for blending in all smelter processes. It was noted that there was significant variability in TCs, RCs, freight charges, and metal payable rates across the industry at the time, however it was thought by H&H that Taca Taca may attract premium terms for its clean concentrate.

The Company assumes that the copper concentrate produced at the Project would be marketed to international smelters globally, and exported to those markets by seaborne trade via a port at Mejillones Bay, Chile. No deleterious elements have been identified or considered at this time. Molybdenum concentrate is considered to be potentially saleable to customers within Chile.

The Company has undertaken a review of H&H's forecasts for treatment charges, refining charges, metal payables, product transport and freight costs, and other metal price factors. The Company has adapted these forecasts based on up-to-date market information and its experience at other projects, and these updated forecasts have been incorporated into the Mineral Reserve cashflow model.

20.2 Agreements for sale of concentrate

The plans for the Taca Taca Project are to produce separate copper and molybdenum concentrates. As is the case with products from the Company's other operations, all products will be sold through the Company's internal marketing division, Metal Corp Trading AG (MCT). There are as yet, no contracts in place for the sale of products from the Project.

20.3 Other contracts and agreements

Copper and molybdenum concentrates from the Project will be transported by railway to Chile. No contractual arrangements for concentrate transport, port usage, shipping, smelting, or refining exist at this time.

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 21 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

21.1 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
- 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.

21.2 Status of environmental approvals

21.2.1 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

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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.

21.2.2 Environmental studies and submissions

Detailed environmental baseline data collection at Taca Taca began in 2016 and Project Alternatives Analysis and Project Description documents were completed in 2018 to complement a Project ESIA which was submitted to the authorities in February 2019. Observations to the EIA were received from the Secretariat of Mining at the end of Q3 2019, including requests for clarification and more information on some environmental aspects. The response document to the observation was submitted in Q1 2020, and remains under review by the authority. Final approval of the ESIA is expected in 2021.

Three additional material permits will be required for the Project, including two ESIA's; for the connection of the 345 kV transmission line to the national electrical grid and for the proposed bypass road construction. The two ESIA's are under preparation and are anticipated to be filed with the relevant authorities in 2021.

The third application will be for approval of the development of borefields for water supply to the Project. This will be completed following the completion of Phase III water supply borehole pump testing and aquifer modelling.

Regulations require the Project ESIA to be updated every two years, whereas the transmission line and road access ESIA's will have indefinite validity.

21.3 Summary of environmental impacts and management requirements

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 include optimisation of heavy equipment, use of equipment with emission control technology, preventive and corrective maintenance of the vehicle fleet, installation of water sprinklers, spraying of roads to control fugitive dust.
2. Exploitation of groundwater:
 - Mitigation measures include optimisation of the use of the resource, respecting the sustainability of the water sources, and prospecting for new water supply sites.

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3. Changes in the chemical constitution of the aquifers with respect to Salar de Arizaro and Salar de Taca Taca:
 - Mitigation measures include the preparation of a Management Plan for the control of contact and non-contact waters (specifically involving routine water quality monitoring).
4. Protection of fauna:
 - Mitigation measures include compliance with traffic regulations, the installation of traffic signs to define fauna crossings and potential interaction locations.

21.4 Summary of social and community impacts and management requirements

The people of Tolar Grande mostly belong to the Kolla Aboriginal Community. An indigenous Prior Consultation and Public Hearing will be led by the Ministry of Indigenous Affairs or the Mining Secretariat. Once the transmission line and road access ESIA's have been submitted, the authority will define the place and date of the public hearing.

As part of the Project ESIA process, workshops, open house discussions and focus group meetings have already been held in the towns of Tolar Grande, Salar de Pocitos, Olacapato and San Antonio de los Cobres, between 2016 and 2018.

Project social and community impacts, and management requirements may be summarised as follows:

1. Immigration for jobs and greater demand for public services in Tolar Grande:
 - Mitigation measures include technical assistance (urban growth planning) and tourist development support (e.g. training in lodging and food services provision).
2. Generation of temporary Project employment:
 - Mitigation measures include sponsoring the best students of the Tolar Grande school for technical/higher studies and the timely communication of policies and procedures for hiring workers.
3. Expectations for engaging with local suppliers:
 - Mitigation measures include strengthening communal enterprises.

21.5 Resettlement

There are no known resettlement requirements for the Project.

21.6 Project closure provisions

The Project closure plan and cost provisioning has considered stages of progressive closure (during the operation stage), final closure and post-closure (maintenance and post-closure monitoring). The estimate is provided in Item 21.4.

ITEM 22 CAPITAL AND OPERATING COSTS

22.1 Cost estimation basis

22.1.1 Capital costs

The Project capital costs produced by Ausenco (2013) for the PEA report are set out in a detailed itemised format which was reviewed by the Company and, where possible, benchmarked against comparable estimates drawn from other Company development projects. Some of these itemised costs have been retained for the Technical Report estimate, whilst others have been replaced with new estimates derived from more recent information supplied by specialist consultants and/or vendors.

Where the Ausenco (2013) capital cost estimates have been retained, these are flagged as such in new and updated tabulations. The original basis of these costs were reported by Ausenco (2013) as follows:

- estimates expressed in first quarter 2013 US dollar terms
- estimates exclude:
 - foreign currency exchange fluctuations
 - escalation
 - all owner's taxes, such as financial transaction tax, withholding tax, or value-added tax (VAT)
 - financing costs
 - reclamation costs
- initial (capitalised) mining costs derived from vendor quotations, and based on the PEA mining pre-strip profile then applicable
- processing plant and infrastructure direct costs derived from material take-off (MTO) quantities relevant to the then applicable layouts and flow sheet
- labour rates and productivity factors benchmarked against then contemporary South American projects
- estimates for the railway line upgrade and rolling stock provided by TFP
- estimates for the power transmission line and associated infrastructure provided by Hugo Gil
- costs for water supply and treatment provided by Schlumberger

For this Technical Report, the basis for updated capital cost estimates is as follows:

- where Ausenco estimates have been retained, these have had contingency factors applied which are typically 15%, and 20% in some instances
- based on experience during the construction of the Cobre Panamá project, several plant and infrastructure cost items have been adjusted and/or added
- an estimate for the TSF, as proposed, was provided by Hillerton Consultants (2019)
- an estimate for the road access diversion (via the preferred AC 2 route) was available from the Ausenco alternatives analysis (2018), but was increased following discussions with a road construction contractor currently involved with upgrading regional roads near to the Project site
- estimates for the railway line upgrade, rolling stock and load-out facility were provided by *Auraxis SA*, with contingency factors up to 30% subsequently reduced by the Company to 15% for the sake of consistency
- estimates for the power transmission line (following the preferred route defined in the Project alternatives analysis) were provided by *Tranelsa*, whilst the associated infrastructure cost estimate (i.e., substations) was adopted from the 2013 Hugo Gil report

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- updates for the water supply estimates were produced internally, based on pipeline length and bore equipping costs, and also referencing new information from M&A on the number of bores required

22.1.2 Operating costs

General and administration costs, and also processing costs, as described in the PEA report (Ausenco 2013) were reviewed by FQM and initially benchmarked against Company projects and operations data. New estimates have since been derived from first principles.

Mine operating costs were estimated from first principles, using the Project mining plan and schedule, with ore and waste haulage routes and lengths identified, and thereby enabling the calculation of loading and hauling cycle times, primary equipment productivity and required fleet numbers.

In both the mining and process operating cost estimates, a diesel fuel price of \$0.75/litre was assumed. From information provided by CAMMESA, the assumed power supply cost was \$0.06/kWh.

22.1.3 Metal costs

Estimated concentrate (rail) transport charges were provided by *Auraxis* (2018), whilst port handling and sea freight charges are estimates based on preliminary enquiries by the Company. Information on applicable treatment and refining charges was provided by the Company's internal metals marketing group.

The *Auraxis* estimates included variable and fixed charges for concentrate transport, reflecting:

- fuel and lubricants for the locomotives and rolling stock
- track and related infrastructure maintenance
- train crew remuneration
- fuel price of \$1.00/litre
- 20% contingency on the variable costs and 30% contingency on the fixed costs

The basis for these estimates was retained for cashflow modelling so as to cover the costs for a separate rail operating entity. For consistency, the fuel price was reduced to \$0.75/litre and the contingency factors were reduced to 15%.

The *Auraxis* estimates also included an amortisation charge for the rail line upgrade capital. This was ignored on the basis that the cost of the rail upgrade is currently carried in the Company's capital cost ledger.

Whilst a capital cost provision is also carried in the Company's ledger for a port upgrade, a concentrate handling charge is nevertheless included with the metal costs to cover a charge for handling concentrate between trains and ships.

22.1.4 Estimate status and accuracy

The contingency amount placed on the Ausenco (2013) capital costs was approximately 15% on average. Ausenco considered the estimate to be developed to a level sufficient to assess/evaluate the Project concept, various development options and the overall viability of the Project. After inclusion of the contingency amount, Ausenco considered the Project capital costs to have a level of accuracy in the range of minus 25% to plus 35%.

The *Auraxis* (2018) estimates for railway capital and operating costs were originally provided with a 30% contingency on the infrastructure items and from 10% to 25% contingency on all other items. For consistency with other infrastructure capital cost estimates, the contingency was adjusted to 15%.

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The order of accuracy of the updated capital cost estimates is considered to be now in the range of minus 15% to plus 15%. Substantial items totalling approximately 85% of the itemised capital costs have benefited from either first principles estimates and material take-offs, or are based on actual costs incurred in the construction of the Company's Cobre Panamá project. Contingency provisions on the itemised costs vary from 0% to 20%, with an overall average of 11%.

22.2 Capital costs

Table 22-1 lists the summary capital costs estimated comprehensively by Ausenco (2013), and broken down into direct and indirect costs. Whilst these are not in the same groupings as listed by Ausenco, the FQM groupings as shown, amount to the same total cost.

During 2018 and again in 2020, the Ausenco estimates in relation to the initial (capitalised) mining costs and indirect costs (engineering, procurement, construction and management (EPCM), and contingency) were reviewed, benchmarked against comparable costs for the Company's Cobre Panamá development project, and updated as listed in Table 22-2.

Subsequently, a number of consultant and vendor cost estimates became available and these are also inserted into Table 22-2 where applicable.

Attention is specifically drawn to the processing plant and infrastructure cost updates, details for which are discussed in the following commentary. Some of these revised capital cost estimates could be assigned to initial Project expenditure and to subsequent expansion. This is particularly so for the processing plant costs, the water supply infrastructure, and for related indirect costs. A split between initial and expansion expenditure is addressed in Item 22.

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Table 22-1 Ausenco (2013) Project capital cost estimate

CAPITAL COST ESTIMATES	120 ktpd (\$M)	60 ktpd (\$M)	180 ktpd (\$M)	CAPITAL COST ESTIMATES	120 ktpd (\$M)	60 ktpd (\$M)	180 ktpd (\$M)
DIRECT COSTS				INDIRECT COSTS			
Mining				Owners costs			
direct costs	\$108.7		\$108.7	preproduction employment & training	\$27.0		\$27.0
primary mining equipment	\$441.5		\$441.5	project & construction management	\$10.6	\$6.1	\$16.7
mining ancillaries and technical services				operations catering	\$10.4		\$10.4
preproduction stripping	\$416.1		\$416.1	camp power	\$5.2		\$5.2
Subtotal	\$966.3	\$0.0	\$966.3	ROW, land acquisition, legal, permits, fees	\$6.9		\$6.9
Processing				insurance	\$14.9	\$2.2	\$17.1
crushing, conveying & storage	\$71.2	\$4.9	\$76.1	corporate travel & services	\$1.3		\$1.3
secondary crushing circuit				environmental	\$1.4		\$1.4
grinding & concentrator	\$504.5	\$252.9	\$757.4	medical, security, communication	\$1.3		\$1.3
concentrate thickening, filtration, storage, handling	\$47.2		\$47.2	community development	\$1.4		\$1.4
reagents/consumables storage/distribution/handling	\$17.9		\$17.9	geotechnical facilities			
Subtotal	\$640.9	\$257.8	\$898.7	third party inspections/testing			
Tailings Management				vendor representatives/commissioning assistance			
TSF construction	\$106.7		\$106.7	spare parts/consumables/initial fills			
Subtotal	\$106.7	\$0.0	\$106.7	Subtotal	\$80.3	\$8.3	\$88.5
Infrastructure				Construction			
access road through Argentina	\$1.8	\$0.0	\$1.8	contractor indirects	\$35.3	\$0.0	\$35.3
access road through Chile				construction temporary facilities	\$33.6	\$10.3	\$43.9
railway, maintenance facilities and airstrip:		rolled-up as below		construction equipment	\$1.8	\$1.3	\$3.1
railway upgrade				construction camp	\$60.0	\$24.4	\$84.4
railway rolling stock and related equipment	\$55.6		\$55.6	Subtotal	\$130.7	\$36.0	\$166.7
rail load-out				Contractor			
port upgrade/expansion	\$0.0	\$0.0	\$0.0	EPCM services	\$151.3	\$38.8	\$190.1
water supply borefield				geotechnical facilities	\$7.6	\$0.0	\$7.6
water supply pumps and pipeline	\$32.7	\$0.1	\$32.8	third party inspections/testing	\$5.2	\$1.6	\$6.8
borefield power supply				vendor representatives/commissioning assistance	\$20.5	\$8.2	\$28.7
process water treatment, storage and distribution	\$4.9	\$0.0	\$4.9	spare parts/consumables/initial fills	\$63.6	\$2.6	\$66.2
power transmission line	\$118.3	\$0.3	\$118.6	Subtotal	\$248.3	\$51.1	\$299.4
power line substations	\$33.8	\$0.1	\$33.9	Other Costs			
internal power distribution	\$25.9	\$0.1	\$26.0	freight, duties & taxes	\$30.8	\$12.9	\$43.7
site earthworks	\$60.5	\$0.1	\$60.6	ROW, land acquisition, legal, permits, fees			
other infrastructure:		Itemised as below		insurance			
camp	\$11.5		\$11.5	contingency	\$386.5	\$64.6	\$451.1
administration building	\$1.7		\$1.7	Subtotal	\$417.3	\$77.5	\$494.9
office/engineering equipment, software, furniture	\$2.1		\$2.1	Total Costs			
laboratory incl. equipment and met lab	\$1.5		\$1.5	Subtotal Indirect Costs	\$876.6	\$172.9	\$1,049.5
mess/kitchen, warehouse, workshops, sewage etc	\$11.8		\$11.8	TOTAL PROJECT CAPITAL			
plant & mine warehouse/truck shop equipment	\$1.0		\$1.0	Total Costs	\$3,005.5	\$431.2	\$3,436.7
medical, safety, security, communication	\$2.0		\$2.0				
site security and fencing	\$0.2	\$0.0	\$0.2				
unspecified site and off-site facilities	\$2.3	\$0.0	\$2.3				
Subtotal	\$367.6	\$0.6	\$368.2				
Other Costs							
ancillary, plant mobile equipment, light vehicles	\$47.4		\$47.4				
Subtotal	\$47.4	\$0.0	\$47.4				
Total Costs							
Subtotal Direct Costs	\$2,128.9	\$258.4	\$2,387.2				

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Table 22-2 Updated Project capital cost estimates, Q3 2020

CAPITAL COST ESTIMATES	Cost (\$M)	Contingency (%)	Cost plus cont. (\$M)	CAPITAL COST ESTIMATES	Cost (\$M)	Contingency (%)	Cost plus cont. (\$M)
DIRECT COSTS				INDIRECT COSTS			
Mining				Owners costs			
primary mining equipment	\$331.9	0%	\$331.9	preproduction employment & training	\$27.0	15%	\$31.0
mining ancillaries and technical services	\$88	0%	\$87.7	project & construction management	\$16.7	15%	\$19.2
preproduction stripping	\$311	0%	\$311.1	operations catering	\$10.4	15%	\$12.0
Subtotal	\$730.7	0%	\$730.7	camp power	\$5.2	15%	\$5.9
Processing				ROW, land acquisition, legal, permits, fees insurance			included in Other Costs
crushing, conveying & storage	\$76.1	15%	\$87.5	corporate travel & services	\$1.3	15%	\$1.5
secondary crushing circuit	\$55.0	15%	\$63.3	environmental	\$1.4	15%	\$1.6
grinding & concentrator	\$757.4	15%	\$871.0	medical, security, communication	\$1.3	15%	\$1.4
concentrate thickening, filtration, storage, handling	\$47.2	15%	\$54.3	community development	\$1.4	15%	\$1.6
reagents/consumables storage/distribution/handling	\$17.9	15%	\$20.6	geotechnical facilities	\$7.6	15%	\$8.8
Subtotal	\$953.7	15%	\$1,096.8	third party inspections/testing	\$6.8	15%	\$7.8
Tailings Management				vendor representatives/commissioning assistance	\$62.3	0%	\$62.3
TSF construction	\$59.6	15%	\$68.5	spare parts/consumables/initial fills	\$66.2	15%	\$76.1
Subtotal	\$59.6	15%	\$68.5	Subtotal	\$207.4	10%	\$229.2
Infrastructure				Construction			
access road through Argentina	\$25.0	15%	\$28.8	contractor indirects	\$35.3	15%	\$40.6
access road through Chile	\$0.0		\$0.0	construction temporary facilities	\$43.9	15%	\$50.5
railway:		Itemised as below		construction equipment	\$3.1	15%	\$3.6
railway upgrade	\$156.4	15%	\$179.9	construction camp	\$98.2	15%	\$112.9
railway rolling stock and related equipment	\$97.6	15%	\$112.2	Subtotal	\$180.5	15%	\$207.5
rail load-out	\$18.4	15%	\$21.1	Contractor			
port upgrade/expansion	\$66.7	20%	\$80.0	EPCM services	\$117.0	10%	\$128.7
water supply borefield	\$59.4	15%	\$68.4	geotechnical facilities			Included in Owners Costs
water supply pumps and pipeline	\$35.3	15%	\$40.6	third party inspections/testing			
borefield power supply	\$56.3	15%	\$64.7	vendor representatives/commissioning assistance			
process water treatment, storage and distribution	\$3.2	20%	\$3.9	spare parts/consumables/initial fills			
power transmission line	\$96.8	10%	\$106.4	Subtotal	\$117.0	10%	\$128.7
power line substations	\$33.8	10%	\$37.2	Other Costs			
internal power distribution	\$17.2	20%	\$20.6	freight, duties & taxes	\$114.6	15%	\$131.8
site earthworks	\$60.6	20%	\$72.8	ROW, land acquisition, legal, permits, fees insurance	\$6.9	15%	\$7.9
other infrastructure:		rolled-up as below			\$14.9	15%	\$17.1
camp				Subtotal	\$136.4	15%	\$156.9
administration building				Total Costs			
office/engineering equipment, software, furniture				Subtotal Indirect Costs	\$641.3	13%	\$722.3
laboratory incl. equipment and met lab				TOTAL PROJECT CAPITAL			
mess/kitchen, warehouse, workshops, sewage etc	\$113.2	15%	\$130.2	Total Costs	\$3,272.5	11%	\$3,639.4
plant & mine warehouse/truck shop equipment							
medical, safety, security, communication							
site security and fencing							
unspecified site and off-site facilities							
Subtotal	\$839.9	15%	\$966.7				
Other Costs							
ancillary, plant mobile equipment, light vehicles	\$47.4	15%	\$54.5				
Subtotal	\$47.4	15%	\$54.5				
Total Costs							
Subtotal Direct Costs	\$2,631.2	11%	\$2,917.1				

22.2.1 Mining equipment and pre-production stripping

The total pre-production mining volumes are 92.3 Mbcm of waste and 6.7 Mbcm of ore, mined onto a stockpile. This equates to a total mined tonnage of 257.5 Mt mined over three years. The total cost of these mining volumes is \$311.1 M, derived from a unit mining cost of \$1.21/t mined for the initial mining horizons.

In terms of the initial purchase of the primary mining and auxiliary fleets listed in (Table 17-7), the total cost is \$419.6 M. This expenditure is summarised below:

- primary mining equipment: electric rope shovels, ultra-class haul trucks, large FEL, blasthole drills, ancillary plant: \$331.9 M
- support mining and maintenance equipment: \$46.0 M
- major equipment spares inventory (8.5% of initial capital purchase costs): \$28.2 M
- technical equipment (communications, office buildings, software, survey equipment etc.): \$6.0 M
- initial mine access roads and bench development: \$7.5 M

22.2.2 Processing plant and TSF

The comprehensive Ausenco (2013) cost estimate, without contingency and for a 120 ktpd flotation plant was \$640.9 M, plus a subsequent \$257.8 M for expansion to 180 ktpd capacity. The Company's benchmarking review considered two comparable sized projects (one of which was Cobre Panamá) and concluded that this cost was a reasonable estimate at the current level of accuracy.

However, a secondary crushing circuit will be required as an additional capital expense not considered by Ausenco. This circuit will be required to enable an increase from initial 40 Mtpa processing to eventual 60 Mtpa processing. As such, an additional expansion cost estimate has been made based on actual costs incurred at the Cobre Panamá project.

Expenditure on several items of the plant design could be deferred until 60 Mtpa processing is required. These deferred items include one primary crusher, one rougher flotation bank, the ultimate cleaner circuit and of one of the three required tailings thickeners.

With the inclusion of a 15% contingency amount, the all-up cost of the 60 Mtpa processing plant for inclusion in the cashflow modelling for this Technical Report, is \$1,096.8 M.

The Ausenco (2013) PEA estimate for the TSF has been replaced by a (provisional) estimate of \$59.6 M plus a 15% contingency. This estimate was produced by Hillerton (May, 2019), and is itemised in Table 22-3.

Table 22-3 Updated (and provisional) cost estimate for the TSF

Item	Description	Units	Unit rate (\$)	Starter facility		Staged expenditure (Yr 15 - 32)	
				Quantity	Amount (\$)	Quantity	Amount (\$)
1.0	Confining embankment						
1.1	Foundation preparation. Excavate unsuitable materials and haul to designated area within 1 km	bcm	\$6.00	25,000	\$150,000	130,000	\$780,000
1.2	Embankment construction. Load, haul, spread to level, moisture condition, mix and compact embankment fill in layers not exceeding 1 m thickness. Haul distance within 2 km (note 2)	bcm	\$15.00	75,000	\$1,125,000	0	\$0
1.3	Embankment construction. Load, haul, spread to level, moisture condition, mix and compact embankment fill in layers not exceeding 1 m thickness. Haul distance within 8 km (note 3)	bcm	\$15.00	0	\$0	2,220,000	\$33,300,000
1.4	Install monitoring bores, downstream of embankment	no.	\$5,000	10	\$50,000	10	\$50,000
2.0	Tailings discharge berms						
2.1	Berm construction. Load, haul, spread to level, moisture condition, mix and compact embankment fill in layers not exceeding 1 m thickness. Haul distance within 2 km	bcm	\$15.00	350,000	\$5,250,000	1,000,000	\$15,000,000
3.0	Access road from plant site to TSF embankment and peripheral road on west side of facility						
3.1	Excavation. Excavate road profile and cart to spoil within 2 km	bcm	\$10.00	125,000	\$1,250,000	0	\$0
3.2	Road construction. Load, haul, spread to level, moisture condition, mix and compact sheeting material	bcm	\$35.00	60,000	\$2,100,000	0	\$0
3.3	Construct road drainage	LS	\$500,000	1	\$500,000	0	\$0
4.0	Tailings delivery pipeline						
4.1	Supply and install complete tailings delivery pipeline. Pipeline to comprise dual steel / HDPE pipelines, 910 mm diameter. Static head = 20 m				by others		
				TOTAL	\$10,425,000		\$49,130,000

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The staged expenditure amount, with the addition of a 15% contingency, has been apportioned as an annual sustaining charge in the cashflow model, applicable from Year 15 of the Project.

22.2.3 New road access to the Project site

In their 2018 alternatives analysis, the findings from which were incorporated into the Project Description document (FQM, 2018), Ausenco completed a cost estimate comparison for alternative Project road access routes. The preferred access route, from the Argentine side, is the modified AC 2 route which by-passes the RP N°27 route in the vicinity of Los Colorados (Item 5.5). The Ausenco (2018) estimate for this by-pass road is \$10.6M, a figure that the Company believes to be potentially underestimated.

Based on the Company's communications with a road construction contractor currently involved with upgrading regional roads near to the Project site, a base estimate of \$25M plus a 15% contingency has been adopted, subject to detailed estimation as the Project engineering study proceeds.

In the ledger of updated capital costs in Table 22-2, no allowance is made at this time for the secondary access road AC 5, from the Chilean side of the border.

22.2.4 Railway upgrade, rolling stock and load-out facility

Auraxis (2018) produced a preliminary capital cost estimate, based on proposals and information provided by rail industry specialists and from their own experience. For the 16 tonne/axle upgrade scenario, *Auraxis* estimated a total base cost of \$338.5 M. Subsequently, it was understood that the Mejillones to Augusta Victoria length of rail track was to be upgraded by the rail owner, and hence the corresponding rail infrastructure cost was deleted from Table 22-4 to yield an amended total of \$272.4 M. The *Auraxis* estimate contingency factors were reduced to 15% to be consistent with other infrastructure estimates.

Table 22-4 Summary capital cost estimate for 16 tonne/axle rail upgrade scenario (after Auraxis, 2018)

Project Component	Entity	Cost Estimate (\$)	Contingency		Total Cost (\$)
			(%)	(\$)	
Rail infrastructure					
Mejillones - Augusta Victoria (223 km)	FCAB				
Augusta Victoria - Socompa (181 km)	FERRONOR	\$83,104,990	15%	\$12,465,749	\$95,570,739
Socompa - Taca Taca (134 km)	ADIFSE	\$73,301,956	15%	\$10,995,293	\$84,297,250
Subtotal		\$156,406,946	15%	\$23,461,042	\$179,867,988
Other items					
Rolling stock	Taca Rail Co.	\$67,865,800	15%	\$10,179,870	\$78,045,670
Other mobile equipment	Taca Rail Co.	\$10,589,663	15%	\$1,588,449	\$12,178,112
Workshops and buildings	Taca Rail Co.	\$12,554,000	15%	\$1,883,100	\$14,437,100
Technology	Taca Rail Co.	\$6,564,720	15%	\$984,708	\$7,549,428
Subtotal		\$97,574,183	15%	\$14,636,127	\$112,210,310
Rail loadout facility					
Subtotal	Taca Rail Co.	\$18,383,619	15%	\$2,757,543	\$21,141,161
TOTAL		\$272,364,748	15%	\$40,854,712	\$313,219,460

Railway upgrade

In the ledger of updated capital costs in Table 22-2, the 16 tonne/ axle rail infrastructure upgrade cost is assumed to be borne by the Company rather than by each of the rail network owners in Argentina and Chile.

Table 22-4 lists a subtotal cost estimate of \$156.4 M (before contingency) for upgrade of respective lengths of track owned by each rail operator. The *Auraxis* (2018) estimate behind these subtotals is detailed to the extent of itemised physicals and unit costs for earthworks, drainage, culverts, ballast, rail track and fasteners, sleepers, turnouts, track works (tamping), track welding, compaction, labour and project management.

Rolling stock and other equipment

In relation to rolling stock and other equipment, Table 22-4 lists a subtotal cost estimate of \$97.6 M (before contingency) for the 16 tonne/axle upgrade scenario. An itemised cost breakdown is provided in Table 22-5.

Table 22-5 Summary rolling stock and other equipment estimate for 16 tonne/axle rail upgrade scenario (Auraxis, 2018)

Item	No. of	Unit price EXW (\$)	Transport & Insurance (\$)	Unit price CIF (\$)	Total Cost (\$)
Rolling stock					
Main line locomotives	13	\$3,250,000	\$125,000	\$3,375,000	\$43,875,000
Shunting locomotives	2	\$3,250,000	\$125,000	\$3,375,000	\$6,750,000
Flat wagons	172	\$75,000	\$10,000	\$85,000	\$14,620,000
Containers	336	\$6,300	\$1,500	\$7,800	\$2,620,800
subtotal	523	\$121,925	\$7,837	\$129,763	\$67,865,800
Other mobile equipment					
Maintenance equipment	7	\$597,643	\$30,714	\$628,357	\$4,398,500
Breakdown equipment	3	\$1,701,888	\$77,667	\$1,779,554	\$5,338,663
Logistics equipment	8	\$105,000	\$1,563	\$106,563	\$852,500
subtotal	18	\$562,731	\$25,583	\$588,315	\$10,589,663
Workshops and buildings					
Various	12	\$993,333	\$28,333	\$1,021,667	\$12,554,000
subtotal	12	\$993,333	\$28,333	\$1,021,667	\$12,554,000
Technology					
Various	302	\$21,161	\$576	\$21,737	\$6,564,720
subtotal	302	\$21,161	\$576	\$21,737	\$6,564,720
Total	855	\$100,370	\$5,730	\$106,100	\$97,574,183

Load-out facility

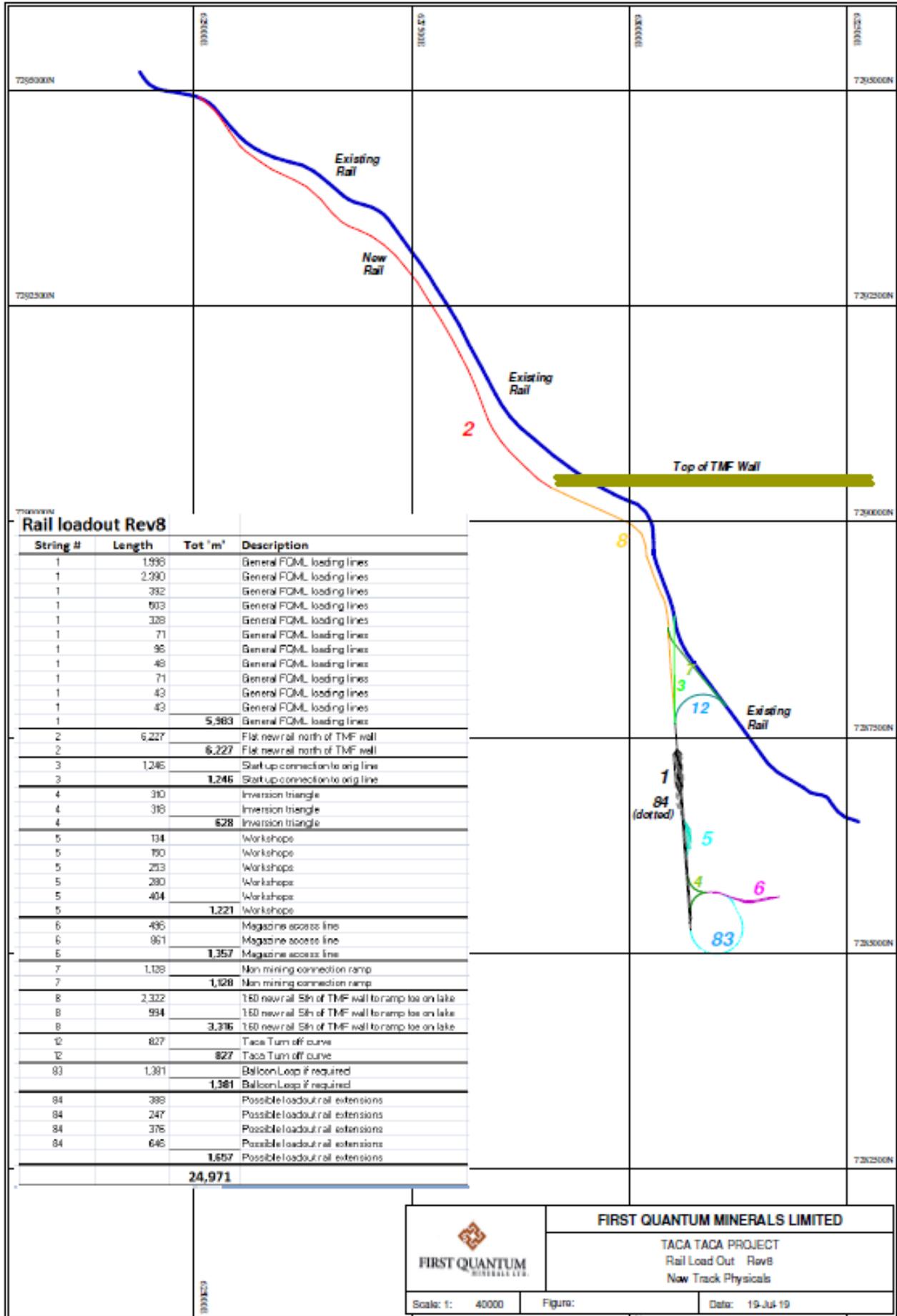
In Table 22-4, \$18.4 M is listed as a subtotal cost estimate (before contingency) for the rail load-out facility.

The Auraxis (2018) estimate behind this subtotal is detailed to the extent of itemised physicals and unit costs for earthworks, drainage, culverts, ballast, rail track and fasteners, sleepers, turnouts, track works (tamping), track welding, compaction, labour and project management. The summary itemisation is provided in Table 22-6. Figure 22-1 shows the reference rail track lengths for this cost estimate.

Table 22-6 Summary rail load-out cost estimate (after Auraxis, 2018)

	Total Cost (\$)
Project management	\$1,446,838
Earthworks	\$6,242,349
Drainage	\$218,558
Ballast	\$1,117,950
Rail track	\$1,212,819
Rail fastenings	\$2,631,291
Rail turnouts	\$2,183,818
Formation works	\$701,888
Track welding	\$995,713
Labour	\$686,323
Site transport	\$672,925
subtotal	\$18,110,469
Shipping & freight insurance	\$273,149
Contingencies	\$2,757,543
Total	\$21,141,161

Figure 22-1 Rail load-out track lengths



22.2.5 Port upgrade/expansion

In relation to the potential Chilean ports identified in Item 18.7, Table 22-2 includes a provisional estimated allowance of \$80 M for upgrade or expansion of a suitable port.

This situation lends itself to the Company holding a competitive bidding process to place concentrate tonnage with one of these ports with minimal capital contribution, if any. There will be a need to balance this against expected port usage/handling fees and tonnage commitments, all aspects of which are to be further explored during the on-going engineering phase.

22.2.6 Power supply

In assessing the alternative power transmission line routes, *Tranelsa* (2017) estimated the engineering and construction costs for the Taca Taca power supply. Considering the preferred route and line distances, the associated capital cost estimate is as shown in Table 22-7.

In addition to the power line itself, the Table 22-2 ledger includes a provision of \$37.2 M (inclusive of 10% contingency) for substations. This figure was assumed from the Ausenco (2013) estimates. A further \$20.6 M is provided for internal power distribution.

Table 22-7 Power line capital cost estimate (*Tranelsa*, 2017)

Item	Cost (\$)
Engineering costs	
general	\$2,370,942
Subtotal	\$2,370,942
Direct and indirect costs	
materials	\$36,438,327
civil works	\$15,596,624
construction	\$26,996,432
Subtotal	\$79,031,383
Contingencies	
materials	\$728,767
civil works	\$1,559,662
construction	\$2,699,643
Subtotal	\$4,988,072
Other costs	
general	\$10,366,848
financial	\$0
Subtotal	\$10,366,848
Contractor margin	
10%	\$9,675,725
Total costs	
Total	\$106,432,970
Total per metre	\$866

22.2.7 Water supply

Referring to the number of estimated bores for the Project (Table 19-3), an estimate was made for the total cost of these, assuming that each of them is 200 m deep and the unit costs for drilling and equipping would be similar to the actual costs incurred for Project water bore drilling in 2018 (Table 22-8). The total cost of \$59.4 M (plus 15% contingency) shown in the Table 22-2 ledger could be staged to suit the ramp-up profile to full scale processing.

In addition to the bores there is also the cost of the pipelines and pumping, an estimate summary of which is provided in Table 22-9. Assumptions and unit costs adopted for this estimate are as follows:

- fresh water borefield pipeline distances and pumping head details are as listed in Table 18-4

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- brine water borefield (Salar de Arizaro) assumed to be on the eastern side of the pit, within 5 km of the plant
- DN200 pipe unit cost is \$36.00/m for a pipe specification of heat treated steel, or \$2.00/m without this specification
- borefield pipeline installation cost is \$5.00/m ,whereas overland pipeline installation cost is \$50.00/m
- electrics costs are \$20.00/m of pipeline distance
- power line transmission costs are \$300,000/km for the distant borefields and \$175,000/km for the brine water supply bores (Table 22-10)
- Valle de Arizaro bore pumps:
 - 2 x 500 kW APP55-250 pumps (together)
- Valle de las Burras bore pumps:
 - 1 x 600 kW APP55-250 pumps and 1 x 300 kW pump on a flat area
- Valle de Chaschas bore pumps:
 - 1 x 600 kW APP55-250 pumps and 1 x 300 kW pump on a flat area
- Socompa bore pumps:
 - 7 x 600 kW APP55-250 pumps and a high pressure pipeline
- Salar de Arizaro bore pumps:
 - 2 x 500 kW APP55-250 pumps (together)
- Assumed \$50/m installation cost of pipelines from Valle de Arizaro, Valle de las Burras and Valle de Chaschas
- Assumed \$500/m installation cost for a high-pressure mountain pipeline from Socompa

Table 22-8 Borefield and associated capital cost estimate

Source/location	No. of production bores	Total bore depth (m)	Bore drilling & equipping (\$)	DN200 pipe cost (\$)	Pipe line installation (\$)	Pumps and tanks (\$)	Electrical reticulation (\$)	Total borefield (\$)
Fresh Water Supply			@ \$1,500/m		@ \$5/m			
Valle de Arizaro	9	1,800	\$2,700,000	\$45,000	\$112,500	\$720,000	\$450,000	\$4,027,500
Valle de las Burras	11	2,200	\$3,300,000	\$55,000	\$137,500	\$880,000	\$550,000	\$4,922,500
Valle de Chaschas	10	2,000	\$3,000,000	\$50,000	\$125,000	\$800,000	\$500,000	\$4,475,000
Socompa	8	1,600	\$2,400,000	\$720,000	\$100,000	\$640,000	\$400,000	\$4,260,000
Subtotal	38		\$11,400,000	\$870,000	\$475,000	\$3,040,000	\$1,900,000	\$17,685,000
Brine Supply								
Pit slope depressurisation/drains	4	800	\$1,200,000	\$20,000	\$50,000	\$320,000	\$200,000	\$1,790,000
Salar de Arizaro	115	23,000	\$23,000,000	\$575,000	\$1,437,500	\$9,200,000	\$5,750,000	\$39,962,500
Subtotal	119		\$24,200,000	\$595,000	\$1,487,500	\$9,520,000	\$5,950,000	\$41,752,500

Table 22-9 Borefield pipeline and associated capital cost estimate

Source/location	Overland pipeline dist. (km)	Pumping lift (m)	Pipe cost (\$)	Heat tracing (\$)	Pipe line installation (\$)	Total pipe line (\$)
Fresh Water Supply						
Valle de Arizaro	25.0	140	\$250,000	\$0	\$1,250,000	\$1,500,000
Valle de las Burras	49.0	0	\$490,000	\$0	\$2,450,000	\$2,940,000
Valle de Chaschas	72.0	0	\$720,000	\$0	\$3,600,000	\$4,320,000
Socompa	61.0	965	\$7,200,000	\$360,000	\$18,000,000	\$25,560,000
Subtotal (Soc pipeline joins VdA)	207.0		\$8,660,000	\$360,000	\$25,300,000	\$34,320,000
Brine Supply						
Pit slope depressurisation/drains	3.0	10	\$150,000	\$0	\$150,000	\$300,000
Salar de Arizaro	6.7	10	\$335,000	\$0	\$335,000	\$670,000
Subtotal	9.7		\$485,000	\$0	\$485,000	\$970,000

Table 22-10 Borefield power transmission and associated capital cost estimate

Source/location	Transmission length (km)	Installed cost (\$)
Fresh Water Supply		
Valle de Arizaro	25.0	\$7,500,000
Valle de las Burras	49.0	\$14,700,000
Valle de Chaschas	72.0	\$21,600,000
Socompa	36.0	\$10,800,000
Subtotal	182.0	\$54,600,000
Brine Supply		
Pit slope depressurisation/drains	3.0	\$525,000
Salar de Arizaro	6.7	\$1,172,500
Subtotal	9.7	\$1,697,500

Table 22-11 summarises the total cost estimates (excluding contingency) that are included in the Table 22-2 ledger.

Table 22-11 Total water supply capital cost estimate

Source/location	Total borefield (\$)	Total pipeline (\$)	Total power (\$)	Grand total (\$)
Fresh Water Supply				
Valle de Arizaro	\$4,027,500	\$1,500,000	\$7,500,000	\$13,027,500
Valle de las Burras	\$4,922,500	\$2,940,000	\$14,700,000	\$22,562,500
Valle de Chaschas	\$4,475,000	\$4,320,000	\$21,600,000	\$30,395,000
Socompa	\$4,260,000	\$25,560,000	\$10,800,000	\$40,620,000
Subtotal	\$17,685,000	\$34,320,000	\$54,600,000	\$106,605,000
Brine Supply				
Pit slope depressurisation/drains	\$1,790,000	\$300,000	\$525,000	\$2,615,000
Salar de Arizaro	\$39,962,500	\$670,000	\$1,172,500	\$41,805,000
Subtotal	\$41,752,500	\$970,000	\$1,697,500	\$44,420,000
Total	\$59,437,500	\$35,290,000	\$56,297,500	\$151,025,000

22.2.8 Other infrastructure

Other infrastructure includes the camp, kitchen, administration buildings, warehouses, site offices and laboratories. The Ausenco estimate was replaced with a provision that referenced the actual construction costs at the Cobre Panamá project. The list of buildings at Cobre Panamá is extensive, so a pro-rated single overall cost provision of \$130.2 M, inclusive of 15% contingency, has been included. This provision also included construction indirect cost allowances as follows:

- vendor representatives and specialist consultants
- 1.8 million man-days for construction of a 4,300 man camp
- EPCM services, assumed to be 65% of Cobre Panamá actual costs
- freight, duties and taxes, assumed to be 65% of Cobre Panamá actual costs

22.3 Sustaining capital costs

The life of mine sustaining cost estimated by Ausenco (2013) was \$1,375.46 M (excluding plant expansion expenditure).

New estimates for mine sustaining capital were based on the following:

- primary equipment replacement costs, spread over the life of mine and amounting to \$688.6 M
- ancillary equipment replacement costs, spread over the life of mine and amounting to \$91.5 M
- support mining and maintenance equipment = \$3.70 M/annum (average)

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- trolley assist electrical infrastructure for pit and waste dump ramps, equivalent to 19.0 km @ \$3.0 M/km and allocated as a proportional annual charge
- dewatering system infrastructure (pumps, pipelines, electrical reticulation) = \$1.0 M/annum
- miscellaneous technical equipment capital = \$0.3 M/annum
- additional spares inventory and technical equipment \$11.0 M (Year 1 only)

On the above basis, the estimated total mine sustaining capital expenditure (excluding the pre-production period) over the life of mine timeframe is approximately \$875.1 M or \$0.19/t mined.

On top of this, an additional annual allowance of 4.5% of the processing and G&A operating costs was estimated as a sustaining provision for the processing plant and for site infrastructure. The total Project life provision amounts to \$372.5 M.

The staged TSF expenditure amount shown in Table 22-3 as \$49.1 M, for embankment lifts etc., (and with the inclusion of 15% contingency) was assumed to be applicable as a further sustaining capital charge.

22.4 Closure costs

GT Ingeniería (2020) produced a conceptual closure plan and cost estimate as listed in Table 22-12. This is a comprehensive estimate, itemised into eleven closure components, plus post-closure activities such as on-going monitoring at specifically identified sites across the Project area.

The estimate components account for dismantling and demolition of infrastructure, longer term surface water management and erosion control, and rehabilitation of the landscape.

Table 22-12 Project closure cost estimate

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%
	CM7: Fuel bay		0.0%
	CM7: Scrap metal, workshops		0.0%
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%
	CM11: Airfield		0.0%
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	22,296,956	
	Supervision (7%)	1,560,787	
	Owners costs (3%)	668,909	
	Total Cost	24,526,652	
	Allowance for complementary studies e.g., ARD, salt formaiton, hydrology, geotechnics (0.1%)	24,527	
	Engineering cost (0.1%)	24,527	
	Contingency (40%)	9,830,282	
	Grand Total Cost	34,405,987	

22.5 Operating costs

22.5.1 General and administration costs

The PEA report (Ausenco, 2013) tabled an overall general and administration (G&A) cost of \$0.57/t of plant feed for an operation scaled at 180,000 tpd. This figure was reviewed and benchmarked against comparable cost centres at the Company operations, inclusive of the following functions:

- general management
- commercial and finance
- support
- environmental and foundation
- safety and security

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Specific reference was made to the Sentinel operations, from which a December 2019 business review report provides the following G&A costs:

On this basis, and considering that the Taca Taca Project would be of a similar scale to Sentinel (i.e. 60 Mtpa processing), the applicable overall unit G&A cost would be \$1.05/tonne processed.

22.5.2 Mining costs

Mining costs comprising drill, blast, load and haul costs, on a bench by bench basis, were derived in Q3 2020. These derivations were estimated from first principles using productivity parameters for the proposed equipment fleet, haul profiles related to preliminary mine designs and a production schedule, and corresponding ore/waste haulage destinations. The estimated costs also took account of faster cycle times arising from trolley assisted haulage and included the following specific components:

- fixed mining costs, comprised of site management and overheads, supervision, operating and maintenance labour
- a variable drill and blast cost depending on material type (ore/waste) and drill type, and based on blasting all mined material
- a variable load and haul cost comprised of the following:
 - truck cycle times estimated from first principles for scheduled haul source and destinations
 - truck diesel fuel usage and electrical power usage (for trolley assist sections)
 - calculation of truck productivity for each haul route
 - calculation of loading unit productivity based on unlimited trucks allocated to the scheduled loading unit
- a stockpile rehandle cost applied to ore material, based on load and haul profiles estimated from first principles
- major equipment life costs based on estimates prepared in 2018 for similar equipment operating at Cobre Panamá
- major component costs for the large haul truck fleet were scheduled according to a planned maintenance profile based on individual truck operational hours
- a variable ancillary plant support cost for the mining operations
- variable costs associated with site support equipment

Key mining cost estimate inputs were as follows:

- diesel fuel = \$0.75/litre
- bulk explosives = \$800/t
- electrical power = \$0.06/kWh

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The commentary below provides information on the estimate of specific fixed and variable mining costs, respectively. An overall summary of the estimated average unit mining operating costs, for ore and waste separately, is given in Table 22-13.

Table 22-13 Estimated average unit mining costs

Unit Cost Analysis	Ore (\$/t)	Waste (\$/t)	Total (\$/t)
Mining overheads (management and services)	\$0.06	\$0.06	\$0.06
Drilling	\$0.08	\$0.07	\$0.07
Blasting	\$0.35	\$0.25	\$0.29
Loading	\$0.12	\$0.12	\$0.12
Hauling	\$0.93	\$0.80	\$0.85
Ancillary Plant	\$0.20	\$0.18	\$0.19
Mining Support	\$0.09	\$0.09	\$0.09
Total cost	\$1.84	\$1.58	\$1.69

Ignoring the capitalised three year pre-strip, the average ore mining cost is \$1.84/t, whilst the average waste mining cost is \$1.58/t. The cost of reclaiming ore from stockpiles is estimated at \$0.74/t rehandled.

Fixed mining costs

The fixed mining overhead cost estimates were based on the Cobre Panamá life of mine plan estimates and scaled where appropriate for the different size of the proposed Taca Taca operation. The total costs were split into the following major components:

- site management and supervision; covering the management, administrative, planning and supervision requirements for the mining operation (including maintenance planning and supervision)
- operating labour; covering the operating labour for the mine (primarily wages labour) for major equipment operation, blast crew activities, grade control sampling, and other support functions (but excludes maintenance labour)
- technical services; including mine planning, mine geology, geotechnical engineering, hydrogeology, surveying (i.e., grade control was assumed to be performed by RC drilling, as at the Cobre Panamá operations)

Table 22-14 shows the effective labour rates per machine hour used in the calculation of the overall wages labour cost. The stated cost includes an allowance for non-productive time while allocated to a machine, and therefore the paid labour rate is lower than the stated values. The fixed cost estimate for mine site management and services is itemised in Table 22-15.

Table 22-14 Labour rates, mining cost estimates

Labour category	\$/machine hour
Blasthole driller	\$14.11
Truck operator	\$12.26
Loading unit operator	\$14.43
Ancillary plant operator	\$16.71

Table 22-15 Fixed mining costs estimate, site management and services

Category	LOM Overhead	Basis of Estimate
Management		
Personnel costs	\$28,170,626	Expatriate 10%, local staff 75 %, local wages 15%
Contract services costs	\$2,817,063	10% of personnel costs
Other operating costs	\$2,271,500	\$0.50/kt mined
Subtotal	\$33,259,189	
Mining Operations		
Personnel costs	\$76,976,253	Expatriate 5%, local staff 30 %, local wages 65%
Other operating costs	\$15,900,500	\$3.50/kt mined
Subtotal	\$92,876,753	
Maintenance		
Personnel costs	\$20,445,597	Expatriate 30%, local staff 35 %, local wages 35%
Contract services costs	\$2,044,560	10% of personnel cost
Equipment, supplies, materials costs	\$2,271,500	\$0.50/kt mined
Other operating costs	\$3,861,550	\$0.85/kt mined
Subtotal	\$28,623,207	
Geology		
Personnel costs	\$18,895,687	Expatriate 45%, local staff 30%, local wages 25%
Contract services costs	\$1,889,569	10% of personnel cost
Equipment, supplies, materials costs	\$2,271,500	\$0.50/kt mined
Assay/laboratory allocation costs	\$0	
RC grade control drilling & assay	\$88,100,000	\$0.05/ore tonne mined (per CP 2021 budget)
Other operating costs	\$1,135,750	\$0.25/kt mined
Subtotal	\$93,396,819	
Technical Services		
Personnel costs	\$29,133,142	Expatriate 45%, local staff 50%, local wages 5%
Contract services costs	\$9,086,000	\$2.00/kt mined
Equipment, supplies, materials costs	\$4,543,000	\$1.00/kt mined
Other operating costs	\$0	
Subtotal	\$42,762,142	
Operating Costs		
Ore	\$117,857,048	
Waste	\$185,854,356	
Total	\$303,711,404	
Unit Cost		
ore (\$/t)	\$0.07	
waste (\$/t)	\$0.07	
total (\$/t)	\$0.07	

Variable mining costs – mining equipment cost parameters

Operating cost estimates for the major and ancillary mining equipment were based on Cobre Panamá cost estimates prepared in November 2018 using a combination of internal and external resources.

Key assumptions used in the calculation of the equipment costs are listed in Table 22-16. In this table the effective maintenance labour hour/rostered hour ratio is a measure of the proportion of time that maintenance activities are performed during a rostered shift.

Table 22-16 Variable mining costs estimate, assumptions used for estimate of major equipment costs

Key Assumptions	Units	Value
Diesel fuel cost	\$/L	\$0.75
Electrical power cost	\$/kwh	\$0.06
Maintenance labour cost	\$/hr	\$15.00
Maintenance labour effective hrs/rostered hr	%	75%
Maintenance labour cost/machine hr	\$/hr	\$20.00

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Table 22-17 lists the key productivity parameters and rates adopted for estimation of the drill and blast equipment costs.

Table 22-17 Variable mining costs estimate, drill and blast equipment parameters and rates

Drill and blast parameters	Units	Fresh Ore		Fresh Waste		Trim	
SG	t/m ³	2.65	2.65	2.65	2.65	2.65	2.65
Drilling Equipment							
Indicative model		PV351	PV271	PV351	PV271	PV271	D65
Power source		electric	electric	electric	electric	electric	diesel
Purchase price	\$M	\$4.4	\$3.2	\$4.4	\$3.2	\$3.2	\$1.1
Service life	Hrs	65,000	65,000	65,000	65,000	65,000	40,000
Depreciation	\$/engine hr	\$67.69	\$49.23	\$67.69	\$49.23	\$49.23	\$26.50
Fuel/electrical power	\$/engine hr	\$74.22	\$40.50	\$74.22	\$40.50	\$40.50	\$22.50
Fluids & grease	\$/engine hr	\$3.70	\$2.16	\$3.70	\$2.16	\$2.16	\$2.07
Drill maintenance and repairs							
Services	\$/engine hr	\$12.11	\$2.64	\$12.11	\$2.64	\$2.64	\$4.25
Minor components & general repairs	\$/engine hr	\$68.74	\$46.73	\$68.74	\$46.73	\$46.73	\$42.10
Major component replacement	\$/engine hr	\$62.01	\$19.34	\$62.01	\$19.34	\$19.34	\$28.38
Bucket/ blade / GET	\$/engine hr						
Dump body	\$/engine hr						
Tyres/tracks	\$/engine hr	\$14.51	\$5.35	\$14.51	\$5.35	\$5.35	\$6.50
Maintenance labour	\$/engine hr	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00	\$20.00
Drill operation rate	\$/engine hr	\$255.29	\$136.71	\$255.29	\$136.71	\$136.71	\$125.80
Driller labour rate	\$/engine hr	\$14.11	\$14.11	\$14.11	\$14.11	\$14.11	\$14.11
Drilling - other parameters							
Maintenance labour hrs/engine hr		1.00	1.00	1.00	1.00	1.00	1.00
Fuel usage	L/hr						30
Electrical power usage	kW/h	1,237	600	1,237	600	600	N/A
Drilling specifications							
Hole diameter	mm	311	270	311	270	251	165
Bench height	m	15.0	15.0	15.0	15.0	15.0	15.0
Subdrill	m	3.0	2.5	3.0	2.5	0.0	1.5
Burden	m	7.1	6.3	9.0	8.0	7.0	5.0
Spacing	m	8.2	7.2	10.4	9.2	8.1	5.8
Drill LM/hole	LM/hr	18.0	17.5	18.0	17.5	15.0	16.5
BCM/hole	BCM	873.3	680.4	1404.0	1104.0	850.5	435.0
Penetration rate	LM/hr	30	30	30	30	30	25
Redrill/extra holes %	%	5%	5%	5%	5%	5%	8%
Drill time/hole	Hr	0.63	0.61	0.63	0.61	0.53	0.71
Production blasting							
Explosive density	g/cm ³	1.20	1.20	1.20	1.20	1.20	1.20
Weight strength	%	1.10	1.10	1.10	1.10	1.10	1.10
Stemming length	m	6.40	5.50	6.40	5.50	5.50	4.00
Stemming volume	m ³	0.97	0.63	0.97	0.63	0.54	0.17
Charge length	m	11.60	12.00	11.60	12.00	9.50	12.50
Charge weight	kg	1057.43	824.48	1057.43	824.48	564.08	320.74
PF (kg/BCM)	kg/BCM	1.21	1.21	0.75	0.75	0.66	0.74
Bulk explosive cost	\$/hole	\$528.71	\$412.24	\$528.71	\$412.24	\$282.04	\$160.37
Detonators (electronic)	\$/hole	\$25.00	\$25.00	\$25.00	\$25.00	\$25.00	\$25.00
Primer	\$/hole	\$4.50	\$4.50	\$4.50	\$4.50	\$4.50	\$4.50
Blasting accessories	\$/hole	\$12.50	\$12.50	\$12.50	\$12.50	\$12.50	\$12.50
Stemming	\$/hole	\$29.17	\$18.89	\$29.17	\$18.89	\$16.33	\$5.13
Blast crew labour	\$/BCM	\$0.10	\$0.10	\$0.10	\$0.10	\$0.10	\$0.10
Explosives contractor labour	\$/BCM	\$0.10	\$0.10	\$0.10	\$0.10	\$0.10	\$0.10

Table 22-18 lists the key productivity parameters and rates adopted for estimation of the load and haul equipment costs. The fuel cost for the haul trucks is calculated on an individual haul route basis, and is therefore zero in Table 22-18.

Table 22-19 lists the key productivity parameters and rates adopted for estimation of the ancillary equipment costs.

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Table 22-18 Variable mining costs estimate, load and haul equipment parameters and rates

Load and haul parameters	Units	Fresh Ore	Fresh Waste	Fresh Ore	Fresh Waste	Fresh Ore	Fresh Waste
SG	t/m ³	2.65	2.65	2.65	2.65	2.65	2.65
Loading and hauling		Loading		Loading		Hauling	
Indicative model		P&H 4100	P&H 4100	R9800	R9800	T284	T284
Power source		electric	electric	diesel	diesel	diesel	diesel
Purchase price	\$M	\$25.0	\$25.0	\$13.8	\$25.0	\$5.5	\$5.5
Service life	Hrs	104,000	104,000	78,000	104,000	72,000	72,000
Depreciation	\$/engine hr	\$240.38	\$240.38	\$176.92	\$240.38	\$76.39	\$76.39
Fuel/electrical power	\$/engine hr	\$92.00	\$92.00	\$375.00	\$92.00	\$0.00	\$0.00
Fluids & grease	\$/engine hr	\$49.64	\$49.64	\$47.20	\$49.64	\$4.79	\$4.79
Maintenance and repairs							
Services	\$/engine hr	\$9.28	\$9.28	\$15.95	\$9.28	\$4.51	\$4.51
Minor components & general repairs	\$/engine hr	\$83.78	\$83.78	\$55.91	\$83.78	\$58.07	\$58.07
Major component replacement	\$/engine hr	\$324.90	\$324.90	\$126.17	\$324.90	\$137.33	\$137.33
Bucket/ blade / GET	\$/engine hr	\$68.24	\$68.24	\$50.00	\$68.24		
Dump body	\$/engine hr					\$13.13	\$13.13
Tyres/tracks	\$/engine hr	\$118.21	\$118.21	\$104.49	\$118.21	\$50.00	\$50.00
Maintenance labour	\$/engine hr	\$25.00	\$25.00	\$25.00	\$25.00	\$25.00	\$25.00
Other parameters							
Maintenance labour hrs/engine hr		1.25	1.25	1.25	1.25	1.25	1.25
Fuel usage	l/hr			500.00		0.0	0.0
Electrical power usage	Kw/h	1,150	1,150	1,150	1,150		
Tyre life	hrs	N/A	N/A	N/A	N/A	6,500	6,500
Price/tyre	\$					\$54,167	\$54,167
Unit cost	\$/hr/tyre					\$8.33	\$8.33
Loading specifications							
Bucket payload	BCM	35.8	35.8	26.9	26.9		
Bucket payload	t	93.0	93.0	70.0	70.0		
Bucket fill factor	%	90%	90%	90%	90%		
Effective bucket load	t	83.7	83.7	63	63		
Bucket cycle time	(min)	0.6	0.6	0.6	0.6		
First bucket drop time	(min)	0.2	0.2	0.1	0.1		
Trucking specifications							
Payload	BCM					137.0	137.0
Payload	t					363.0	363.0
Number of buckets/truck load	No.					4.3	4.3
Truck queue time at loading unit	(min)					0.00	0.00
Truck spot & depart time at loading unit	(min)					0.75	0.75
Queue at dump	(min)					0.50	0.00
Truck spot & dump time at dump	(min)					1.00	1.00
Truck load time	(min)					2.10	2.10
Total truck fixed time/cycle	(min)					4.35	3.85
Truck dump time	(min)					1.50	1.00
Truck loading area time	(min)					2.85	2.85
Productivity							
Loading unit productive time	mins/hr	52.0	52.0	50.0	50.0		
Fully trucked fleet productivity	BCM/OP Hr	2,448	2,448	1,878	1,878		
Fully trucked fleet productivity	TONNES/OP Hr	6,366	6,366	4,884	4,884		
Productive hours/yr	Hrs/annum	6,000	6,000	6,000	6,000		
Production capacity	Mtpa	38.2	38.2	29.3	29.3		

Table 22-19 Variable mining costs estimate, ancillary equipment parameters and rates

Equipment Type	Unit	Large FEL	Dozer	Dozer	Grader	Water cart
Ancillaries						
Indicative model		L2350	D11	D10	16M	785 WDT
Power source		diesel	diesel	diesel	diesel	diesel
Purchase price	\$M	\$9.50	\$1.80	\$1.00	\$0.85	\$1.75
Service life	Hrs	72,000	65,000	65,000	50,000	50,000
Depreciation	\$/engine hr	\$131.94	\$27.69	\$15.38	\$17.00	\$35.00
Fuel/electrical power	\$/engine hr	\$135.00	\$74.25	\$52.50	\$21.00	\$71.25
Fluids & grease	\$/engine hr	\$3.52	\$3.05	\$2.03	\$0.98	\$3.47
Maintenance and repairs						
Services	\$/engine hr	\$6.32	\$2.96	\$2.98	\$2.57	\$5.17
Minor components & general repairs	\$/engine hr	\$81.55	\$34.85	\$28.14	\$21.37	\$28.41
Major component replacement	\$/engine hr	\$219.21	\$38.84	\$32.92	\$5.54	\$55.21
Bucket/ blade / GET	\$/engine hr	\$27.14	\$17.38	\$14.85	\$25.22	
Dump body	\$/engine hr					
Tyres/tracks	\$/engine hr	\$44.56	\$30.92	\$22.03	\$3.92	\$19.11
Maintenance labour	\$/engine hr	\$25.00	\$20.00	\$20.00	\$20.00	\$20.00
Other Parameters						
Maintenance labour hrs/engine hr		1.25	1.00	1.00	1.00	1.00
Fuel usage	l/hr	180	99	70	28	95
Electrical power usage	Kw/h					
Tyre life	hrs	10,000	N/A	N/A	3,000	4,000
Price/tyre	\$	\$111,405			\$2,400	\$22,910
Unit cost	\$/hr/tyre	\$11.14			\$0.80	\$5.73

Drill and blast costs

Drill and blast costs were estimated using the following assumptions (and are summarised in Table 22-20):

- drill patterns, allocated drill type and performance were developed referencing Cobre Panamá experience and internal benchmarking
- bulk emulsion explosives supply and distribution costs (including appropriate on-site manufacturing and storage facilities) referencing November 2018 budget quotes from an explosives manufacturer located in Argentina
- drill operator labour rate based on Cobre Panamá unit rates
- key drill and blast consumable costs based on recent budget quotes for Cobre Panamá and referring to operating experience:
 - a single electronic detonator per hole at \$25/detonator
 - a single primer per hole at \$4.50 each
 - other blast consumables (detonating cord, etc.) allowed at \$12.50/hole
 - crushed stemming produced from an onsite crusher supplied at \$30/lcm
- blast crew labour and explosives contractor labour, separately, were allowed at \$0.10/bcm blasted

Furthermore:

- an allowance of between 5% and 8%, depending on pattern type, was made for additional re-drilling of blastholes that may collapse, and/or for additional holes required on a blast pattern to allow blasting of irregular blast faces or back break
- an allowance for pre-splitting was included on the basis of pre-splitting a proportion of each bench in each phase, primarily below and adjacent to haulage ramps; the total pre-split drill metres over the life of mine are estimated to be 1.9 M metres.
- a small amount of development waste blasting (i.e. <1 % of the total waste volume) was assumed to be required in the upper benches of each mining phase, where working space will be restricted

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- trim blasting volumes were estimated by assuming an average 30 m width around the perimeter of all mining benches and pit phases

Table 22-20 Variable mining costs estimate, drill and blast

Drill and Blast Costs	Units	Fresh Ore		Fresh Waste		Trim	
		PV351	PV271	PV351	PV271	PV271	D65
Drilling							
Drill cost per hole	\$/hole	\$221.51	\$142.47	\$221.51	\$142.47	\$122.12	\$148.16
Drill cost per linear metre	\$/LM	\$11.72	\$7.75	\$11.72	\$7.75	\$7.75	\$8.31
Drill cost (incl drill consumables)	\$/BCM	\$0.25	\$0.21	\$0.25	\$0.21	\$0.14	\$0.34
Drill cost (incl drill consumables)	\$/t	\$0.10	\$0.08	\$0.10	\$0.08	\$0.05	\$0.13
Production blasting							
Subtotal blasting cost	\$/hole	\$604.80	\$477.45	\$606.12	\$478.65	\$345.23	\$210.98
Total blasting cost	\$/BCM	\$0.89	\$0.90	\$0.63	\$0.63	\$0.61	\$0.69
Total blasting cost	\$/t	\$0.34	\$0.34	\$0.24	\$0.24	\$0.23	\$0.26
Total Costs							
Total drill and blast cost	\$/BCM	\$826.32	\$619.93	\$827.64	\$621.13	\$467.35	\$359.14
Total drill and blast cost	\$/BCM	\$1.15	\$1.11	\$0.79	\$0.76	\$0.75	\$1.03
Total drill and blast cost	\$/t	\$0.43	\$0.42	\$0.30	\$0.29	\$0.28	\$0.39

Load and haul costs

Utilising the Project pit designs and production schedules, individual haul profiles were generated for each phase of mining at 15 m vertical intervals. These profiles covered the following destinations:

- ore to crusher
- ore to low grade stockpile
- waste to available waste dump areas

The resulting truck cycle times were generated for an indicative Liebherr T284 electric drive truck using the travel speeds on each segment type as listed in Table 22-21. Additional allowances were made for the impact of switchbacks and traffic controls (e.g. stop signs) within appropriate haul routes.

Table 22-21 Variable mining costs estimate, T284 truck cycle times

Segment Type	Units	No Trolley		With Trolley	
		Loaded	Empty	Loaded	Empty
On bench	km/h	20	25	20	25
Uphill (10% gradient)	km/h	12	25	22	25
Flat < 500 m distance	km/h	25	30	25	30
Flat > 500 m distance	km/h	40	45	40	45
Downhill (10% gradient)	km/h	15	25	15	25

Where appropriate (e.g. final inpit phase ramps and waste dump ramps of greater than 300 m length), cycle times based on the use of trolley assist were also generated. For information, 75% of inpit (ore and waste) material was determined to be hauled with some component of trolley assist on the inpit ramp system, and 35% of waste was determined to be hauled using a component of trolley assist on the waste dump ramps.

Truck fuel burn rates for the indicative Liebherr T284 haul truck on each haul route were estimated using the fuel burn rates for each segment type as listed in Table 22-22.

Table 22-22 Variable mining costs estimate, T284 truck fuel burn rates

Segment Type	Units	Diesel Fuel Burn Rate
Loaded; up ramp; no trolley	l/hr	656
Loaded; up-ramp; trolley	l/hr	0
Loaded; down ramp	l/hr	200
Loaded; flat haul	l/hr	400
Empty; flat haul	l/hr	145
Truck spotting and loading	l/hr	40
Truck spotting and dumping	l/hr	40

In order to check the validity of the assumptions, a representative range of twenty four 'no trolley' haul profiles (twelve each of ore and waste) and a similar number of hauls with trolley assist were simulated by Liebherr in October 2018. The travel time and fuel burn values simulated by Liebherr were generally within 10% to 15% of the values originally calculated.

In order to generate the numbers of trucks required on a particular haul, the following methodology was adopted:

- The scheduled shovel productivity was assumed to be maximised at all times by fully trucking the shovel to meet the scheduled material movement requirements.
- The 'exact match' number of trucks was calculated to achieve maximum shovel productivity from ore or waste, respectively. As the scheduled tonnages on each haul are large (averaging > 5 Mt), this provides a reasonable approximation of the average number of trucks to be allocated to a haul route.
- If the 'exact match' number of trucks on a particular haul route was greater than 10, the allocated trucks were limited to eight units, in accordance with good mining practice to minimise congestion, etc.
 - this has the result of lowering shovel productivity (particularly on the longer hauls at the base of each stage and in the later stages of the mine life)
 - the loading unit productivity of approximately 35% of material mined was affected by this restriction

The resulting estimated average load and haul costs are summarised in Table 22-23.

Table 22-23 Variable mining costs estimate, load and haul

Load and Haul Cost	Units	Primary Shovel	Primary Excavator	Haul Truck
Basic cost		P&H 4100	R9800	T284
Basic cost	\$/engine hr	\$771.05	\$799.72	\$439.08
Operating cost				
ore	\$M	\$179.7	\$27.1	\$1,625.4
waste	\$M	\$308.2	\$24.0	\$2,159.4
total	\$M	\$488.0	\$51.1	\$3,784.9
Unit cost				
ore	\$/t	\$0.10	\$0.02	\$0.92
waste	\$/t	\$0.11	\$0.01	\$0.78
total	\$/t	\$0.11	\$0.01	\$0.83

Stockpile rehandle costs

Stockpile rehandle costs were estimated on the following basis:

- Stockpile rehandle during the mine life is envisaged to be performed with a large FEL (L2350) and L284 haul trucks as part of normal operations. This fleet would only be used where this activity was not the only source of mill feed, as the feed rate would be insufficient to meet the target plant feed rates.
- At the end of the mine life, the long term stockpiles would be rehandled using the P&H 4100 shovels and L284 trucks to meet the plant feed requirements.
- The rehandle stockpiles are envisaged to be located within 1 to 2 km of the processing plant, so as to minimise haulage requirements.
- Haul profiles were generated for all hauls from the active and long term stockpile locations, which when combined with typical equipment costs and productivities, resulted in a direct average cost of \$0.81/t for each rehandled ore tonne (excluding capital, mine overheads and mine support).

Ancillary equipment costs

The major mobile ancillary plant fleet provides support to the mining operation in the following activities:

- surface preparation for mining and dumping operations
- blasthole drill surface preparation
- levelling of post blast surfaces as required
- construction and maintenance of temporary access roads inside and outside the pit area
- construction and maintenance of permanent and temporary haul roads
- mining bench floor preparation and loading unit clean-up
- minor stockpile rehandle activities and replacement for shovels when moving for blasts
- waste dump tip head management for haulage operations
- waste dump battering to final slope and associated rehabilitation

The ancillary fleet requirements relate to the assumptions listed in Table 21-24. These assumptions were based on internal and external benchmarking of similar operations. It should be noted that the level of watercart usage is based on implementing an effective surface road stabilisation strategy (which is likely to include road cementing products and the like) to minimise water usage on main haul roads.

The resulting estimated average load and haul costs are summarised in Table 21-25.

Table 22-24 Variable mining costs estimate assumptions for ancillary equipment

Ancillary fleet	Typical unit	Usage rate
Dozer	Cat D11/D10	1.50 hours/shovel hour
Grader	Cat 16M	0.30 hours/haul truck hour
Watercart	Cat 785	0.20 hours/haul truck hour
Large FEL	LeTourneau L2350	0.20 hours/shovel hour

Table 22-25 Variable mining costs estimate, ancillary equipment

Equipment Type	Units	Large FEL	Dozer	Dozer	Grader	Water cart
Basic cost		L2350	D11	D10	16M	785 WDT
Basic cost	\$/engine hr	\$542.30	\$222.27	\$175.45	\$100.60	\$202.62
Operating cost						
ore	\$M	\$33.0	\$91.2		\$88.1	\$141.9
waste	\$M	\$51.6	\$146.0		\$118.9	\$191.5
total	\$M	\$84.6	\$237.2		\$206.9	\$333.4
Unit cost						
ore	\$/t	\$0.02	\$0.05		\$0.05	\$0.08
waste	\$/t	\$0.02	\$0.05		\$0.04	\$0.07
total	\$/t	\$0.02	\$0.05		\$0.05	\$0.07

Site support equipment and services costs

Site support equipment and services costs typically relate to:

- Pit dewatering; maintenance and operation of the expit borehole dewatering system located, in this instance, between the pit and the salar, horizontal drain hole installation and operation, as well as in-pit sump pumping as required, and as listed in Table 21-26.
- Maintenance and operations support equipment and other items as listed in Table 21-27; these items have been costed on an annual basis (excluding operating labour) and are treated as fixed costs as their usage is relatively independent of the rate of mining.

Table 22-26 Variable mining costs estimate, pit dewatering cost

Category	LOM Overhead	Basis of Estimate
Dewatering labour cost	\$8,773,554	18 operators per year x \$16,000/annum per person
Horizontal drainage	\$23,373,000	Based on \$360/LM drilled and equipped
Electric power cost	\$75,422,805	Power @\$0.08/kwh, variable pumping requirement
Dewatering materials	\$13,415,380	Based on 4.5 ML/day pumped production
Operating cost (incl fuel/power)		
ore	\$44,263,889	
waste	\$76,720,849	
total	\$120,984,738	
Unit cost		
ore (\$/t)	\$0.03	
waste (\$/t)	\$0.03	
total (\$/t)	\$0.03	

Table 22-27 Variable mining costs estimate, maintenance and operations support equipment

Category	Description	No of Units	Service Life (Yrs.)	Capital Cost	Op. Cost/Yr.	LOM OP. Cost
Mining Support	350 t hydraulic excavator	1	10	\$3,500,000	\$850,000	\$28,050,000
Mining Support	120 t hydraulic excavator	1	10	\$1,425,123	\$402,063	\$13,268,095
Mining Support	85 t payload haul truck	4	10	\$1,500,000	\$475,000	\$62,700,000
Mining Support	Hydraulic hammer/impactor	1	12	\$144,010	\$24,036	\$793,178
Mining Support	45 t backhoe/loader	1	8	\$129,384	\$56,065	\$1,850,159
Mining Support	Prime mover for equipment float	1	20	\$5,633,363	\$495,531	\$16,352,533
Mining Support	Cable reeler	1	7	\$240,360	\$30,147	\$994,849
Mining Support	Stemming/roadbase crushing plant	1	15	\$5,971,185	\$1,032,135	\$34,060,446
Mining Support	Lighting plant	30	5	\$8,776	\$11,069	\$10,958,511
Mining Support	Vibratory compactor	5	10	\$183,387	\$54,034	\$8,915,610
Maintenance	200 t capacity diesel crawler crane	2	15	\$1,586,356	\$60,659	\$4,003,483
Maintenance	80 t rough-terrain crane	1	15	\$753,800	\$36,601	\$1,207,827
Maintenance	5 t all-terrain forklift	2	10	\$61,879	\$9,766	\$644,528
Maintenance	16 t all-terrain forklift	3	10	\$255,123	\$31,930	\$3,161,031
Maintenance	30 t tyre handling forklift	1	10	\$321,102	\$35,229	\$1,162,542
Maintenance	Off-highway fuel/lube truck	3	15	\$1,624,142	\$539,690	\$53,429,310
Maintenance	On-highway fuel/lube truck	3	5	\$230,923	\$49,049	\$4,855,859
Maintenance	Light truck for field mechanics/welders	4	5	\$306,639	\$56,621	\$7,473,929
Maintenance	Aerial boom truck	3	10	\$313,443	\$32,977	\$3,264,686
Maintenance	Utility truck with crane	3	5	\$136,134	\$35,244	\$3,489,160
Maintenance	Tyre handler	1	15	\$225,000	\$0	\$0
Vehicles	4x4 diesel pickup - single cab	12	4	\$28,937	\$24,402	\$9,663,192
Vehicles	4x4 diesel pickup - dual cab	20	4	\$30,141	\$24,552	\$16,204,320
Vehicles	Crew bus, diesel (30 seat)	20	10	\$207,499	\$25,113	\$16,574,580
Miscellaneous	Truck dispatch system	1	10	\$2,796,423	\$190,000	\$6,270,000
Miscellaneous	Ambulance	1	15	\$81,275	\$5,978	\$197,273
Miscellaneous	Mine & geology software	1	15	\$545,900	\$132,113	\$4,359,729
Operating Cost						
ore						\$114,846,291
waste						\$199,058,538
total						\$313,904,830
Unit Cost						
ore (\$/t)						\$0.07
waste (\$/t)						\$0.07
total (\$/t)						\$0.07

22.5.3 Processing costs

The PEA report (Ausenco, 2013) tabled an overall process operating cost of \$4.26/t of plant feed, to which was added \$0.19/t for rail operations and \$0.06/t for rail infrastructure (load-out) maintenance; yielding a total operating cost of \$4.51/t of plant feed.

The estimate has been updated by the Company on a first principles basis, as outlined in the following commentary.

Estimate information sources and assumptions

Mill liner and ball consumption rates were provided by Orway Mineral Consultants (OMC), based on the ten composite samples tested in 2019 (refer to Item 13). Consumption rates were benchmarked against actual operating data from Sentinel and Cobre Panamá.

Reagent costs were calculated from the Process Design Criteria (Item 17) and by reference to the results obtained from flotation testwork at ALS in Canada using brine solutions. Reagent consumption rates for molybdenum flotation were obtained from the Cobre Panamá design criteria, whilst reagent costs were provided from a combination of Cobre Panamá and Sentinel current cost data.

Energy consumption estimates were taken from the PEA report (Ausenco, 2013), and modified according to changes to the flowsheet, equipment sizes (particularly for the mills and secondary crushers) and throughput rates. A detailed equipment list has not been derived for the process plant at this point in the Project engineering. A cost for energy of \$60/MWh was adopted.

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Maintenance costs were calculated as 5% of the total direct mechanical, electrical and mobile equipment capital costs, together with an allowance for piping and valves.

Labour costs were developed assuming labour levels similar to those at Sentinel and Cobre Panamá. Salary plus benefit costs were derived from a combination of costs for the same job categories at Cobre Panamá, from certain costs listed by Ausenco in the PEA, and salary levels as set out by the Argentine construction union (UECARA, September 2018).

Consumables costs

Crusher liner consumption rates and costs were supplied from the Cobre Panamá operations.

As noted above, mill liner and steel ball consumptions were derived from comminution testwork on ten composite samples of the various ore types at Taca Taca, as 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 consumptions.

Mill liner consumptions were calculated using original liner weights, with an assumption that 30% would be scrap at the end of the liner life. Wear rates were calculated in kg/t milled, as estimated by OMC. However, these calculations gave unrealistic liner life (typically three months) relative to industry norms. Based on operating practice at Sentinel, a full liner set per mill, per annum, has been allowed for at Taca Taca. This is felt 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.

The process consumable costs (predominantly reagents) were obtained from consumption rates detailed in the design criteria and derived from the recent testwork at ALS.

The recent work focused on flotation in brine solutions obtained from the Salar de Arizaro adjacent to the mine site. Reagent consumptions from the locked cycle flotation tests in brine solutions were used in this cost analysis. Reagent costs were a combination of recent costs at Sentinel and Cobre Panamá.

Table 22-28 lists the itemised breakdown of the consumables cost estimate, as at October 2020.

Costs described under 'General' in Table 21-28 have been taken from Cobre Panamá operating cost estimates, with the exception of:

- Laboratory reagents and consumables were adjusted upwards to \$300,000 per annum based on advice from Sentinel. This includes the cost of running the metallurgical laboratory.
- In the 2013 PEA, Ausenco allowed a figure of \$250,000 per annum for off-site testwork and for consultants. This figure has been retained for this cost estimate.

These numbers compare with the Ausenco 2013 PEA:

- \$0.17/t for liners at 180,000 tpd
- \$1.04/t for grinding media which comprised Cr-Mo balls instead of forged steel balls
- \$0.81/t for reagents, with the main difference being a higher lime dosage used by Ausenco, but much lower frother and NaHS addition rates

Costs at Sentinel for 2019 were \$0.82/t for grinding media and \$0.48/t for reagents. Reagent costs at Taca Taca will be higher because of the high consumption of frother, indicated by the testwork, and the inclusion of reagents (e.g. NaHS) for molybdenum recovery.

Table 22-28 Consumables cost estimate breakdown, October 2020

Item		Cost to Site US\$/Unit	Unit	Consumption Rate	Annual Consumption	TOTAL	
						US\$/a	US\$/t
Crusher Liners							
Primary Crusher	Concave	\$192,930	set		8 set(s)	\$1,543,440	\$0.03
Primary Crusher	Mantle	\$158,050	set		8 set(s)	\$1,264,400	\$0.02
Secondary & Pebble Crusher Liners							
(3 x Metso MP 2500, 2 x MP 1250)	Allowance					\$4,000,000	\$0.07
Mill Liners							
SAG Mill	Steel	\$2,435,400	set	1 set per mill pa	2 set(s)	\$4,870,800	\$0.08
Ball Mill	Steel	\$1,063,800	set	1 set per mill pa	4 set(s)	\$4,255,200	\$0.07
Regrind Ball Mill	Steel	\$501,400	set		1 set(s)	\$501,400	\$0.01
Lime Ball Mill Liners	Steel	\$50,000	set		1 set(s)	\$50,000	\$0.00
Subtotal Liners						\$16,485,240	\$0.27
Grinding Media							
SAG Mill Balls	140 mm	\$1,150	t	0.28 kg/t ore	16,800 t	\$19,320,000	\$0.32
Ball Mill Balls	65 mm	\$1,000	t	0.45 kg/t ore	27,000 t	\$27,000,000	\$0.45
Regrind Mill Balls	Ceramic Beads	\$5,100	t	0.02 kg/t ore	1,140 t	\$5,814,000	\$0.10
Lime Ball Mill Media	50 mm	\$1,000	t	0.80 kg/t lime	38.4 t	\$38,400	\$0.00
Subtotal Grinding Media						\$52,172,400	\$0.87
Reagents							
Flocculant	Rougher Tails	\$4,100	t	25 g/t feed	1,478 t	\$6,061,534	\$0.10
Flocculant	Regrind	\$4,100	t	25 g/t regrind feed	225 t	\$922,500	\$0.02
Flocculant	Cln Scav Tails	\$4,100	t	25 g/t cln scav tails	203 t	\$834,034	\$0.01
Flocculant	Concentrate	\$4,100	t	25 g/t conc	22 t	\$88,466	\$0.00
Collector	SEX	\$2,000	t	50 g/t ore	3,000 t	\$6,000,000	\$0.10
Mo Promoter		\$2,000	t	10 g/t ore	600 t	\$1,200,000	\$0.02
Mo Collector	Pine Oil	\$1,000	t	20 g/t ore	1,200 t	\$1,200,000	\$0.02
Sodium Silicate		\$425	t	0.13 kg/t conc	112 t	\$47,685	\$0.00
Sodium Hydroxide		\$540	t	0.33 kg/t conc	285 t	\$153,801	\$0.00
Frother	MIBC	\$3,000	t	100 g/t ore	6,000 t	\$18,000,000	\$0.30
Lime (pH Modifier)	Quicklime	\$300	t	800 g/t ore	48,000 t	\$14,400,000	\$0.24
Xanthate Remover (for Mo)	NaHS	\$650	t	15 kg/t conc	12,515 t	\$8,134,529	\$0.14
Subtotal Reagents						\$57,042,549	\$0.95
Concentrate Filtration							
Filter Cloth		\$9,809	set		16 set(s)	\$156,948	\$0.00
Water Treatment							
Potable Water Treatment	Allowance	\$0.10	kL	500 kL/day	182,500 kL	\$18,250	\$0.00
Fuel							
Diesel Process Plant		\$750	kL		657 kL	\$492,750	\$0.01
General							
Mill Lubricants	Allowance	\$70,000	lot			\$70,000	\$0.00
Plant Operating Tools & Equipment	Allowance	\$50,000	lot			\$50,000	\$0.00
General Supplies	Allowance	\$15,000	lot			\$15,000	\$0.00
Operator Supplies	Allowance	\$50,000	lot			\$50,000	\$0.00
Lab Reagents and Consumables	Allowance	\$300,000	lot			\$300,000	\$0.01
Consultants & Testwork	Allowance	\$250,000	lot			\$250,000	\$0.00
Subtotal Miscellaneous						\$1,152,948	\$0.02
TOTAL						\$126,853,137	\$2.11

Energy costs

Energy will be supplied to the processing plant from the local power grid, via 122.5 km of new transmission line installed for the Project. On the basis of recent discussions with CAMMESA, unit costs of \$60/MWh have been used in this estimate, which is a reduction from the \$85/MWh as used by Ausenco.

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Energy consumption was calculated from the electrical load list generated as part of the mechanical equipment developed by Ausenco in the 2013 PEA. Although the equipment list has not been updated for this Technical Report, adjustments to the energy consumption estimate for the 180,000 tpd plant have been made on the basis of:

- changes in the number and size of primary crushers (from two to three operating, four installed)
- addition of three secondary crushers to the circuit
- increased mill sizes

The milling circuit will account for approximately 80% of the energy consumption of the processing plant, and it was thus assumed that any changes in the energy consumption of the remaining areas of the plant would be minimal, and have not been captured. A more detailed and up to date equipment list and power study will be performed during the engineering phase of the Project.

The updated energy cost estimate is listed in Table 22-29.

Table 22-29 Energy (power) cost estimate breakdown

Plant Area	MWh pa	\$pa	\$/t
Crushing	75,389	\$4,523,350	\$0.075
Milling	1,329,219	\$79,753,147	\$1.329
Flotation	120,803	\$7,248,205	\$0.121
Cons Thickening	14,721	\$883,281	\$0.015
Tails	9,128	\$547,682	\$0.009
Reagents	994	\$59,649	\$0.001
Utilities	22,745	\$1,364,687	\$0.023
Total	1,573,000	\$94,380,000	\$1.573

Maintenance costs

Annual maintenance consumables costs are based on 5% of the total direct capital cost for the plant. The estimated costs are shown in Table 22-30.

Table 22-30 Maintenance cost estimate

Description	Unit	Value
Direct Capital Cost	\$	\$1,096,761,300
Maintenance Cost	%	5%
	\$ pa	\$51,673,065
	\$/t ore treated	\$0.861

The estimate in Table 22-30 compares with Sentinel costs of \$41,415,273 for maintenance to the end of September 2019 (i.e., total engineering costs less labour and contractors) on 36,472,642 tonnes milled, which equates to \$1.135/tonne milled. Sentinel engineering costs, however, included crusher and mill liners under engineering spares; a figure estimated as \$0.28/t. For the Taca Taca cost estimate, liners have been included under mill consumables.

After excluding liners from the engineering and maintenance costs, the Sentinel costs would be \$0.855/t, which is very similar to those estimated above.

Process labour costs

Management, supervisor, operator and maintenance personnel requirements for the processing facilities have been estimated from the personnel numbers at similar sized FQM projects, i.e. Sentinel and Cobre Panamá.

The total cost estimate for expatriate personnel has been based on the net salary for each position, as paid at Cobre Panamá, and multiplied by a factor of 1.9 to cover all taxes, social security, medical, and miscellaneous government taxes and fees. Local labour rates have been supplied by the Administration and Finance Superintendent for FQM in Salta in May 2020 (Table 22-31).

Table 22-31 Argentine salary rates

Position	Gross Salary, AR\$		Social Security and Taxes	Total Annual Cost	
	Monthly	Annual		AR\$	US\$
Supervisor	\$116,897	\$1,519,655	\$395,110	\$1,914,765	\$30,393
Operator	\$41,215	\$535,790	\$139,305	\$675,095	\$10,716
Attendant	\$36,868	\$479,281	\$124,613	\$603,894	\$9,586
Labourer	\$26,664	\$346,632	\$90,124	\$436,756	\$6,933

These costs include a thirteenth cheque, loadings for working at altitude, plus medical insurance with the current coverage used for the field workers. Rosters are likely to be 14 days on, 14 days off, with shifts of 12 hours duration. Four people would be required to cover each shift position.

Leave allowances in Argentina depend on the length of employment in the Company:

- from 6 months up to 5 years: 14 days per year
- from 5 years up to 10 years: 21 days per year
- from 10 years up to 20 years: 28 days per year
- more than 20 years: 35 days per year

The salaries presented in Table 22-31 are significantly different from those reported by Ausenco in the 2013 PEA, where operators and labourers were costed at US\$29,963 per annum. This difference can be ascribed to the fact that local labour will be paid in Argentine Pesos, whereas the costs reported in this Technical Report are in US dollars.

Whilst salaries have increased substantially in Argentina since 2013, the exchange rate for the peso has changed from AR\$5.50 per dollar in 2013 to AR\$63.00 in 2019.

In order to allocate the labour costs, the plant was divided into five main operating areas, namely crushing, grinding, flotation, services (reagents), and tailings management. Each area would be managed by an area superintendent on day shift, with a number of 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.

Table 22-32 summarises the processing labour complement and the associated operating cost estimate.

Maintenance labour costs

The maintenance labour estimate has been built-up in a similar manner to the operating complement estimate, with maintenance supervisors on dayshift in charge of the maintenance for individual sections of the plant.

Table 22-32 Processing labour complement and operating cost estimate

Processing	Day Shift	Per Shift	Total	\$pa (each)	\$pa total
Plant Manager (expat)	1		1	\$399,000	\$399,000
Assist Plant Manager (expat)	1		1	\$351,500	\$351,500
Admin Assistant	1		1	\$11,000	\$11,000
Office Cleaners	2		2	\$7,000	\$14,000
Process Trainer	1		1	\$40,000	\$40,000
Training Assistants	3		3	\$15,000	\$45,000
Technical Superintendent (expat)	1		1	\$256,500	\$256,500
Senior Metallurgists (expat)	3		3	\$228,000	\$684,000
Metallurgists	6		6	\$30,000	\$180,000
Met Assistants/Junior Mets	4		4	\$22,500	\$90,000
Process Control Superintendent (expat)	1		1	\$256,500	\$256,500
Process Control Engineer	2		2	\$218,500	\$437,000
Control Room Operators		2	8	\$15,000	\$120,000
Chief Chemist (expat)	1		1	\$250,800	\$250,800
Senior Chemists	2	1	6	\$40,000	\$240,000
Laboratory Technicians	4	2	12	\$15,000	\$180,000
Lab Cleaners		2	8	\$7,000	\$56,000
Shift Supervisor		1	4	\$30,000	\$120,000
TMF Supervisor	1		1	\$30,000	\$30,000
Area Superintendents	5		5	\$40,000	\$200,000
Operators		10	40	\$15,000	\$600,000
Attendants		13	52	\$10,000	\$520,000
Labourers	10	23	102	\$7,000	\$714,000
Totals	49	54	265		\$5,795,300

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.

Table 21-33 summarises the maintenance labour complement and the associated operating cost estimate.

The total process and maintenance costs are similar to those estimated by Ausenco in 2013. The unit costs for local labour are significantly lower than those used by Ausenco, as described above, but these are offset by the higher complement of 362 (i.e., 265 for process, plus 97 for maintenance) compared with the 206 personnel detailed by Ausenco.

By comparison, the Sentinel labour complement was 365 in 2019 (including 43 positions filled by contractors).

Table 22-33 Maintenance labour complement and operating cost estimate

Maintenance	Day Shift	Per Shift	Total	\$pa (each)	\$pa total
Engineering Manager (expat)	1		1	\$360,000	\$360,000
Electrical Superintendent (expat)	1		1	\$294,500	\$294,500
Mechanical Superintendent (expat)	1		1	\$294,500	\$294,500
Instrumentation Superintendent (expat)	1		1	\$294,500	\$294,500
Admin Assistant	1		1	\$11,000	\$11,000
Maintenance Planner	1		1	\$45,000	\$45,000
Crusher Supervisor	1		1	\$30,000	\$30,000
Mill Supervisor	1		1	\$30,000	\$30,000
Flotation Supervisor	1		1	\$30,000	\$30,000
Service Supervisor	1		1	\$30,000	\$30,000
Mechanics, Boilermakers	40	1	44	\$15,000	\$660,000
Electricians	15	1	19	\$15,000	\$285,000
Control & Instrumentation	20	1	24	\$15,000	\$360,000
Totals	85	3	97		\$2,724,500

Estimate summary

Table 22-34 provides a summary of the process operating costs for a plant throughput of 60 Mtpa and compares these costs with estimates provided by Ausenco in the 2013 PEA Report. Differences between the current operating cost estimates and the earlier estimates are discussed above.

Table 22-34 Process operating cost summary

Throughput	FQM, 2020 60 Mtpa, 7500 tph		Ausenco, 2013 180,000 tpd
	\$pa	\$/t	\$/t
Mill & Crusher Liners	\$16,485,240	\$0.275	\$0.17
Grinding Media	\$52,172,400	\$0.870	\$1.04
Reagents	\$57,042,549	\$0.951	\$0.81
Misc	\$1,152,948	\$0.019	\$0.02
Sub Total - Consumables	\$126,853,137	\$2.114	\$2.04
Energy	\$94,380,000	\$1.573	\$1.84
Maintenance	\$51,673,065	\$0.861	\$0.18
Process Labour	\$5,795,300	\$0.097	\$0.15
Engineering Labour	\$2,724,500	\$0.045	incl in above
Concentrate transportation			\$0.01
Total unit processing cost	\$281,426,002	\$4.69	\$4.22

The process costs are distributed between fixed and variable costs in Table 22-35, below. Fixed costs are defined as 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.

An initial estimate was completed preparatory to the pit optimisation described in Item 15.3, yielding an overall cost of \$4.82/t. The \$4.69/t listed in Table 22-34 results from a revised energy cost estimate as described above.

Table 22-35 Variable and fixed processing costs estimate

Variable costs	\$/t
Consumables	\$2.11
Energy	\$1.57
Maintenance (assume 50% variable)	\$0.43
Subtotal	\$4.12
Fixed costs	\$/t
Maintenance (50%)	\$0.43
Labour	\$0.14
Subtotal	\$0.57
Total	\$4.69

22.6 Metal costs

In addition to royalties, metal costs for each of the copper and molybdenum concentrates comprise:

- concentrate transport charges
- port handling charges
- concentrate treatment and refining charges

22.6.1 Railway operations (concentrate transport costs)

Auraxis (2018) produced a preliminary concentrate transport cost estimate of approximately \$60.00/t, based on proposals and information provided by rail industry specialists and from their own experience (Table 22-36). Whilst this total cost was used for the pit optimisation, the figure was reduced for the subsequent cashflow model by deleting the amortisation component and adjusting the *Auraxis* fuel price and contingency assumptions.

Table 22-36 Operating cost estimate for rail transport of copper concentrate (*Auraxis*, 2018), showing charges adopted for pit optimisation input and cashflow modelling

Rail Operating Costs	Units	16 tonne/axle scenario	
		Optimisation	Cashflow model
Production figures			
Concentrate tonnage (including moisture)	tonnes	1,000,000	
Production of net-tonne-kilometres	tkm	538,000,000	
Rail operating costs			
Variable	\$	\$25,949,060	\$22,591,789
Fixed	\$	\$19,457,919	\$16,903,906
Amortisation	\$	\$14,652,499	
Subtotal costs	\$	\$60,059,479	\$54,148,193
Unit costs			
Variable	\$/tkm	\$0.048	\$0.042
Fixed	\$/tkm	\$0.036	\$0.031
Amortisation	\$/tkm	\$0.027	
Subtotal unit metal costs	\$/tkm	\$0.112	\$0.073
Variable	\$/t con	\$25.95	\$22.59
Fixed	\$/t con	\$19.46	\$16.90
Amortisation	\$/t con	\$14.65	
Subtotal unit metal costs	\$/t con	\$60.06	\$39.50

The Company's metals marketing division advised an approximate cost of \$48.00/t for molybdenum concentrate transport assuming that the product would be freighted in two tonne bulk bags.

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The unit transport costs in the cashflow model were converted to a dry metric tonnes basis, assuming a 10% moisture content in the concentrate.

22.6.2 Port and shipping costs

From preliminary discussions that the Company has had with port operators at Mejillones Bay, the indicative price for port logistics and concentrate handling charges is in the order of \$7.00/wmt (\$7.78/dmt) of concentrate.

Initial discussions with these 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 \$37.00/t of concentrate was adopted for the pit optimisations. Based on subsequent advice from the Company's metals marketing group, this was increased to \$48.50/wmt (\$53.89/dmt).

22.6.3 Export levy

As at October 2020, it is understood that the Argentine government has not extended a minerals concentrate export levy beyond the end of 2021. On this basis, no such levy is included with royalties in the estimation of Project metal costs.

22.6.4 Summary metals cost estimate

Table 22-37 provides a comparison summary of the metal costs adopted for mine planning and cashflow modelling. In comparing the metal costs carried in the PEA (Ausenco, 2103) cashflow model against those adopted for the subsequent FQM optimisations and cashflow modelling, the following comments can be made:

- the updated copper transport and freight charges are about 32% higher than those estimated for the PEA
- the updated copper treatment and refining charges are also higher, with a net effect of an overall copper metal cost increase of 57% relative to the PEA estimated costs
- the updated molybdenum transport and freight charges are significantly higher than those estimated for the PEA
- the updated gold refining charge is lower and hence the gold metal cost is about 40% lower than the PEA estimate
- a difference between the metal costs adopted for mine planning (pit optimisation) and subsequently for cashflow modelling, arises from an increase in the gold price from \$1,200/oz to \$1,500/oz
- despite the differences, the updated total net return (i.e., from recovered copper, molybdenum and gold) is within 5% of that in the PEA, and the updated marginal cut-off grade is slightly higher than would apply for the PEA

The royalty charges shown in Table 21-37 are simply a gross royalty. Net of the charges described in Item 4.6, the average levied royalties would be:

- copper royalty, net of charges = \$0.09/lb Cu
- molybdenum royalty, net of charges = \$0.038/lb Mo
- gold royalty, net of charges = \$51.52/oz

Further work is required during the Project engineering phase to review and revise all product treatment and freight charges. The capital costs in the Mineral Reserve cashflow model include the cost of rail upgrades between the site and the Chilean port, and also the indicative cost of a port upgrade. It may transpire that

the rail upgrade and port capital costs could be deleted and replaced with additional concentrate transport and handling charges.

Table 22-37 Comparison of metal costs used in pit optimisation and cashflow modelling

Metal costs (TCRCs)	Units	PEA c/flow	Optimisation		Cashflow model	
			Primary	Non-primary	Primary	Non-primary
Metal prices						
Copper price	\$/lb	\$2.75	\$3.00	\$3.00	\$3.00	\$3.00
Molybdenum price	\$/lb	\$12.00	\$12.00	\$12.00	\$12.00	\$12.00
Gold price	\$/oz	\$1,200	\$1,200	\$1,200	\$1,500	\$1,500
Royalty rates						
Salta provincial royalty	%	3.0%	3.0%	3.0%	3.0%	3.0%
Third party royalty	%	1.5%	1.5%	1.5%	1.5%	1.5%
Withholding tax on imports	%					
Total - all metals	%	4.5%	4.5%	4.5%	4.5%	4.5%
Process recovery						
Copper recovery	%	90.0%	86.0%	83.8%	86.0%	83.8%
Molybdenum recovery	%	56.5%	40.0%	40.0%	40.0%	40.0%
Gold recovery	%	63.7%	60.0%	60.0%	60.0%	60.0%
Copper concentrate charges						
Copper payable rate	%	96.5%	96.2%	96.2%	96.2%	96.2%
Cu concentrate grade	%	31.7%	25.0%	25.3%	25.0%	25.3%
Transport and freight charges:						
Concentrate rail transport	\$/dmt		\$60.00	\$60.00	\$43.88	\$43.88
Port charges	\$/dmt		\$13.50	\$13.50	\$7.78	\$7.78
Sea freight charges	\$/dmt		\$37.00	\$37.00	\$53.89	\$53.89
subtotal	\$/dmt	\$79.67	\$110.50	\$110.50	\$105.55	\$105.55
Copper treatment charge	\$/dmt	\$70.00	\$90.00	\$90.00	\$90.00	\$90.00
Cu refining (on payable)	\$/lb	\$0.07	\$0.09	\$0.09	\$0.09	\$0.09
Copper metal cost	\$/lb	\$0.29	\$0.47	\$0.46	\$0.46	\$0.45
Molybdenum concentrate charges:						
Molybdenum payable rate	%	86.0%	86.0%	86.0%	86.0%	86.0%
Molybdenum concentrate grade	%	50.4%	47.0%	47.0%	47.0%	47.0%
Transport and freight charges:						
Concentrate rail transport	\$/dmt		\$48.00	\$48.00	\$53.33	\$53.33
Port charges	\$/dmt		\$13.50	\$13.50	\$7.78	\$7.78
Sea freight charges	\$/dmt		\$37.00	\$37.00	\$53.89	\$53.89
subtotal	\$/dmt	\$15.67	\$98.50	\$98.50	\$115.00	\$115.00
Molybdenum treatment charge	\$/dmt	\$68.19	\$68.19	\$68.19	\$68.19	\$68.19
Molybdenum refining (on payable)	\$/lb	\$0.00	\$0.00	\$0.00	\$0.00	\$0.00
Molybdenum metal cost	\$/lb	\$0.09	\$0.19	\$0.19	\$0.21	\$0.21
Gold charges						
Gold payable rate	%	100.0%	90.0%	90.0%	90.0%	90.0%
Gold refining (on payable)	\$/oz	\$7.50	\$5.10	\$5.10	\$5.10	\$5.10
Gold metal cost	\$/oz	\$7.50	\$4.59	\$4.59	\$4.59	\$4.59
Metal costs, including royalties						
	Units	PEA c/flow	Optimisation		Cashflow model	
			Primary	Non-primary	Primary	Non-primary
Royalty charges						
Copper	\$/lb	\$0.12	\$0.14	\$0.14	\$0.14	\$0.14
Molybdenum	\$/lb	\$0.54	\$0.54	\$0.54	\$0.54	\$0.54
Gold	\$/oz	\$54.00	\$54.00	\$54.00	\$67.50	\$67.50
Total metal costs						
Total copper metal cost	\$/lb	\$0.42	\$0.60	\$0.60	\$0.59	\$0.59
Total molybdenum metal cost	\$/lb	\$0.63	\$0.73	\$0.73	\$0.75	\$0.75
Total gold metal cost	\$/oz	\$61.50	\$58.59	\$58.59	\$72.09	\$72.09
Total metal costs as %age of price						
Copper metal cost	%	15.1%	20.1%	19.9%	19.8%	19.6%
Molybdenum metal cost	%	5.2%	6.1%	6.1%	6.2%	6.2%
Gold metal cost	%	5.1%	4.9%	4.9%	4.8%	4.8%
Net return						
Copper metal cost	\$/lb	\$0.42	\$0.60	\$0.60	\$0.59	\$0.59
Copper net return	\$/lb	\$2.33	\$2.40	\$2.40	\$2.41	\$2.41
Copper net return (recovered)	\$/lb	\$2.10	\$2.06	\$2.01	\$2.07	\$2.02
Molybdenum metal cost	\$/lb	\$0.63	\$0.73	\$0.73	\$0.75	\$0.75
Molybdenum net return	\$/lb	\$11.37	\$11.27	\$11.27	\$11.25	\$11.25
Cu_{eq} net return (recovered)	\$/lb	\$0.12	\$0.06	\$0.06	\$0.06	\$0.06
Gold metal cost	\$/oz	\$61.50	\$58.59	\$58.59	\$72.09	\$72.09
Gold net return	\$/oz	\$1,139	\$1,141	\$1,141	\$1,428	\$1,428
Cu_{eq} net return (recovered)	\$/lb	\$0.16	\$0.14	\$0.14	\$0.18	\$0.18
Total Net Return (recovered)	\$/lb	\$2.38	\$2.26	\$2.21	\$2.31	\$2.26
Total Net Return (recovered)	\$/10kg	\$52.49	\$49.75	\$48.81	\$50.82	\$49.90
Marginal cut-off grade impact						
Cu_{eq} COG	%	0.10	0.13	0.13	0.12	0.12

ITEM 23 ECONOMIC ANALYSIS

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 in order to demonstrate a positive cashflow for mining and processing. The development and expansion capital costs are included in the analysis for completeness. The model is provided pre-tax and post-tax.

23.1 Methodology and key assumptions

The basic methodology adopted for the economic analysis was to tabulate the detailed production schedule (ore and waste mined, ore processed, stockpiles reclaimed) and the recovered metal profiles for the schedule described in Item 16. To reflect the gradual attainment of the designed plant recovery during plant ramp-up in the first year 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. Operating costs, initial and expansion capital costs, and also sustaining capital costs were deducted from the net revenue to finally arrive at an annual cashflow.

Key assumptions for the base case economic analysis are as follows:

- the plant feed tonnes and grade account for mining dilution and mining recovery factors
- the initial development capital is expended in Years -3 to -1
 - the assumed spending intensity is 21% in Year -3, 29% in Year -2 and 50% in Year -3
- initial production starts in Year 1 at 30 Mt, ramping up to 40 Mtpa for Years 2 to 6, then to 50 Mt in Year, 7 and finally to 60 Mtpa processed from Year 8 onwards
- the expanded production at a rate of 60 Mtpa continues until Year 31
- the final year, Year 32, completes the process feed inventory by reclaiming a final 42.7 Mt from long term stockpiles
- costs associated with pre-strip mining volumes in Years -3 to -1 were assumed to be capitalised
- the first three years of processing feature down-rated processing recoveries
- the annual operating costs (i.e., processing and G&A unit costs) were not profiled for each year
- the metal costs were not profiled against varying concentrate grade, payability or other factors
- the cashflows consider royalties, notional capital depreciation schedules and the applicable corporate tax rate; they exclude an export levy

23.1.1 Metal pricing

The annual revenues in the Mineral Reserve cashflow model were calculated referencing late Q3 2020 average consensus pricing information for copper, from a number of banks and financial service institutions, as listed in Table 23-1. As such, a long term price of \$3.00/lb copper was adopted for modelling, the same as that adopted for the pit optimisation (Item 15.3.3).

A similar projection of gold pricing information was also referenced, as listed in Table 23-2. A conservative long term price of \$1,200/oz gold was adopted for the pit optimisation (Item 15.3.3), whereas \$1,500/oz gold was adopted for the cashflow model. A long term price of \$12.00/lb for molybdenum was adopted.

Table 23-1 Consensus copper pricing information referenced for cashflow modelling, Q3 2020

Date	2020E (\$/lb)	2021E (\$/lb)	2022E (\$/lb)	2023E (\$/lb)	2024E (\$/lb)	LT (\$/lb)
19 Oct '20	\$2.74	\$2.85	\$2.76	\$2.71	\$2.65	\$2.75
23 Oct '20	\$2.77	\$2.82	\$2.60	\$2.68	\$2.91	\$3.25
11 Sep '20	\$2.69	\$2.86	n/a	n/a	n/a	\$2.95
15 Oct '20	\$2.74	\$3.00	\$3.00	\$3.00	\$3.00	\$3.00
18 Oct '20	\$2.70	\$2.88	\$3.00	\$3.10	\$3.20	\$3.55
01 Jun '20	\$2.40	\$2.68	n/a	n/a	n/a	n/a
19 Oct '20	\$2.75	\$3.00	\$3.00	\$3.00	\$3.00	\$3.00
19 Oct '20	\$2.69	\$2.80	n/a	n/a	n/a	\$3.00
25 Oct '20	\$2.72	\$2.95	\$2.72	n/a	n/a	\$2.90
14 Oct '20	\$2.65	\$2.95	\$3.00	\$3.25	\$3.25	\$3.25
15 Oct '20	\$2.72	\$2.75	\$2.65	\$2.65	\$2.75	\$3.00
06 Oct '20	\$2.76	\$3.15	\$3.25	\$3.50	\$3.50	\$3.00
15 Oct '20	\$2.74	\$3.03	\$3.03	\$3.11	n/a	\$3.00
15 Oct '20	\$2.74	\$3.36	\$2.83	\$2.53	\$2.69	\$3.17
18 Oct '20	\$2.72	\$2.95	\$2.90	\$2.90	n/a	\$3.00
11 Sep '20	\$2.69	\$3.00	n/a	n/a	n/a	n/a
13 Oct '20	\$2.74	\$3.00	\$3.10	\$3.15	\$3.25	\$3.00
29 Sep '20	\$2.74	\$2.88	\$2.81	\$2.90	\$2.99	n/a
27 Oct '20	\$2.68	\$2.75	\$2.50	\$2.50	n/a	\$3.00
09 Oct '20	\$2.74	\$3.15	\$2.95	\$3.00	\$3.05	\$3.10
01 Oct '20	\$2.74	\$3.00	\$3.20	\$3.30	\$3.30	\$3.00
Consensus Average	\$2.71	\$2.94	\$2.90	\$2.96	\$3.04	\$3.05
Maximum	\$2.77	\$3.36	\$3.25	\$3.50	\$3.50	\$3.55
Median	\$2.74	\$2.95	\$2.95	\$3.00	\$3.00	\$3.00
Minimum	\$2.40	\$2.68	\$2.50	\$2.50	\$2.65	\$2.75

Table 23-2 Consensus gold pricing information referenced for cashflow modelling, Q3 2020

Date	2020E (\$/oz)	2021E (\$/oz)	2022E (\$/oz)	2023E (\$/oz)	2024E (\$/oz)	LT (\$/oz)
19 Oct '20	\$1,776	\$1,850	\$1,575	\$1,400	\$1,350	\$1,350
23 Oct '20	\$1,783	\$1,915	\$1,855	\$1,800	\$1,675	\$1,400
15 Oct '20	\$1,778	\$1,916	\$1,943	\$1,965	\$1,984	\$1,984
18 Oct '20	\$1,800	\$2,300	\$2,200	\$2,100	\$2,000	\$1,650
01 Jun '20	\$1,680	\$1,925	n/a	n/a	n/a	n/a
21 Sep '20	\$1,800	\$2,125	n/a	n/a	n/a	n/a
19 Oct '20	\$1,777	\$1,900	\$1,900	\$1,900	\$1,900	\$1,900
19 Oct '20	\$1,830	\$2,500	n/a	n/a	n/a	\$1,400
25 Oct '20	\$1,777	\$1,963	\$1,622	n/a	n/a	\$1,550
14 Oct '20	\$1,787	\$2,100	\$2,200	\$2,100	\$2,100	\$1,700
13 Oct '20	\$1,524	\$1,700	\$1,650	n/a	n/a	n/a
15 Oct '20	\$1,793	\$1,965	\$1,850	\$1,862	\$1,881	\$1,600
15 Oct '20	\$1,802	\$1,980	\$1,948	\$1,917	n/a	\$1,600
15 Oct '20	\$1,782	\$1,951	\$1,825	\$1,475	\$1,380	\$1,350
18 Oct '20	\$1,787	\$1,935	\$1,820	\$1,705	n/a	\$1,590
27 Oct '20	\$1,786	\$1,893	\$1,800	\$1,800	n/a	\$1,500
19 Oct '20	\$1,775	\$1,850	\$1,850	\$1,700	\$1,500	\$1,500
29 Sep '20	\$1,761	\$1,750	\$1,750	\$1,750	\$1,750	n/a
15 Oct '20	\$1,769	\$1,750	\$1,750	\$1,750	n/a	n/a
20 Oct '20	\$1,789	\$2,100	\$2,000	\$1,800	\$1,500	\$1,500
01 Oct '20	\$1,815	\$2,100	\$2,000	\$1,625	\$1,500	\$1,300
Consensus Average	\$1,770	\$1,975	\$1,863	\$1,791	\$1,710	\$1,555
Maximum	\$1,830	\$2,500	\$2,200	\$2,100	\$2,100	\$1,984
Median	\$1,783	\$1,935	\$1,850	\$1,800	\$1,713	\$1,525
Minimum	\$1,524	\$1,700	\$1,575	\$1,400	\$1,350	\$1,300

23.1.2 Corporate tax

The applicable Argentine corporate tax rate, as advised by the Company's internal taxation advisers, is 25%. In adopting this rate, the taxable income as determined by the Company's advisers, has been calculated on a simple basis and does not reflect any benefits that would usually be expected from the structuring of Project funding to include an appropriate level of debt and equity.

23.1.3 Royalties and other levies

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 addition, there is 3% royalty payable to the Province of Salta, net of smelting/refining, transport, general and administration costs, and also process operating costs. 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.

23.1.4 Carbon tax

Whilst the basic cashflow model reported herein is pre-tax, the following supplementary information is provided in the context of the estimated carbon dioxide (CO₂) emissions that would otherwise be considered, based on the projected annual diesel fuel and electrical power consumption for the Project. Table 23-3 lists the annual diesel fuel and power consumption for the Project, in addition to the estimated emission of CO₂. 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 CO₂ emissions relates to the following coefficients:
 - diesel fuel = 2.69 kg CO₂ per litre of fuel
 - explosives = 0.19 kg CO₂ per litre of fuel
 - electric power = 485.6 g CO₂ 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

Considering the above, the impact of a carbon tax is considered to be immaterial to the economics of the Project.

23.1.5 Payable metal factors

The payable metal factors adopted in the cashflow model were assumed to be:

- recovered copper = 96.2%
- recovered molybdenum = 86.0%
- recovered gold = 90.0%

Table 23-3 Theoretical carbon dioxide emissions

Year	Diesel fuel consumption		Bulk explosives (kt or ML)	Power consumption		Mining emission Diesel coeff.: 2.69 (tonne CO _{2e})	Explosives emission Explosives coeff.: 0.19 (tonne CO _{2e})	Scope 1 Emission Total estimated emission (tonne CO _{2e})	Power supply emission	
	Mining (fleet consumption)	Processing (not combusted)		Mining (fleet consumption)	Processing plant				Base case 100% renewable	Alt case (80% natural gas, 20% renewable)
	(ML)	(ML)		(MWh)	(MWh)				(tonne CO _{2e})	(tonne CO _{2e})
-3	18.4		12.6	3,694		49,515	2,383	51,897	0	1,435
-2	36.6		29.7	17,250		98,297	5,604	103,901	0	6,701
-1	45.5		34.7	20,278		122,248	6,551	128,799	0	7,878
1	88.2	0.01	64.2	40,757	898,000	237,022	12,126	249,148	0	364,689
2	111.2	0.01	77.9	67,977	1,042,000	298,848	14,722	313,570	0	431,204
3	108.3	0.02	77.1	68,623	1,042,000	291,057	14,576	305,634	0	431,455
4	104.8	0.02	77.7	82,242	1,042,000	281,814	14,688	296,502	0	436,746
5	105.5	0.02	78.6	90,973	1,042,000	283,470	14,856	298,325	0	440,137
6	117.9	0.01	80.1	97,884	1,042,000	316,900	15,139	332,039	0	442,822
7	103.9	0.02	78.3	98,522	1,307,000	279,369	14,793	294,162	0	546,017
8	105.7	0.02	78.0	127,552	1,566,000	284,160	14,743	298,903	0	657,911
9	120.4	0.02	80.6	140,694	1,566,000	323,572	15,239	338,811	0	663,016
10	115.0	0.01	80.0	159,563	1,566,000	309,230	15,117	324,347	0	670,347
11	72.0	0.01	60.5	139,780	1,566,000	193,569	11,440	205,009	0	662,661
12	64.0	0.01	55.2	136,031	1,566,000	171,967	10,441	182,408	0	661,205
13	62.7	0.01	52.0	145,788	1,566,000	168,573	9,836	178,409	0	664,995
14	61.5	0.01	51.2	152,540	1,566,000	165,263	9,671	174,934	0	667,618
15	57.8	0.01	51.5	156,147	1,566,000	155,425	9,733	165,159	0	669,020
16	59.3	0.01	51.9	175,224	1,566,000	159,485	9,807	169,292	0	676,431
17	58.8	0.01	48.9	178,580	1,566,000	158,011	9,241	167,252	0	677,735
18	61.5	0.01	49.4	192,732	1,566,000	165,175	9,344	174,519	0	683,232
19	56.7	0.01	43.6	176,779	1,566,000	152,503	8,232	160,734	0	677,035
20	61.8	0.01	43.8	182,595	1,566,000	166,234	8,277	174,510	0	679,294
21	59.9	0.01	41.0	173,714	1,566,000	161,132	7,752	168,885	0	675,844
22	61.2	0.01	42.0	186,180	1,566,000	164,484	7,937	172,421	0	680,687
23	50.4	0.01	33.4	155,071	1,566,000	135,374	6,303	141,677	0	668,602
24	47.2	0.01	31.6	155,745	1,566,000	126,800	5,964	132,764	0	668,864
25	49.0	0.01	32.1	159,979	1,566,000	131,655	6,064	137,719	0	670,508
26	46.5	0.01	28.3	138,408	1,566,000	124,998	5,355	130,354	0	662,128
27	36.0	0.01	12.8	59,128	1,566,000	96,717	2,425	99,142	0	631,330
28	17.1	0.00		12,616	1,566,000	45,833		45,833	0	613,261
29	15.9	0.00		12,650	1,573,000	42,621		42,621	0	615,993
30	15.4	0.00		12,616	1,566,000	41,432		41,432	0	613,261
31	17.1	0.00		12,616	1,566,000	45,998		45,998	0	613,261
32	13.4	0.00		8,995	1,121,000	35,947		35,947	0	438,980
33										
TOTAL	2,226.6	0.39	1,578.6	3,739,923	46,127,000	5,984,697	298,361	6,283,058	0	19,372,302
AVERAGE	63.6	0.01	52.6	106,855	1,441,469	170,991	9,945	179,516	0	553,494

23.1 Cashflow model inputs

23.1.1 Production schedule

The production schedule forming the basis of the Mineral Reserve cashflow model is the same as that listed in Item 16-3. It is reproduced here as Table 23-4.

Table 23-4 Life of mine production schedule

Year	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)
-3					43.5	43.5							
-2	3.6	0.22	85.80	0.10	97.1	100.6							
-1	13.9	0.39	107.20	0.12	99.5	113.4							
1	33.6	0.45	115.90	0.16	171.2	204.7	29.9	0.52	119.80	0.16	97.5	1.4	91.8
2	40.4	0.60	93.18	0.11	205.4	245.8	39.9	0.63	106.56	0.14	193.2	1.7	104.1
3	34.8	0.94	135.26	0.13	210.8	245.6	39.9	0.82	128.22	0.13	275.2	2.0	98.5
4	37.4	0.74	145.01	0.12	208.7	246.2	39.9	0.76	143.66	0.12	262.1	2.3	90.9
5	43.7	0.74	167.10	0.16	202.2	245.8	39.9	0.79	178.34	0.17	274.6	2.8	134.1
6	57.8	0.61	166.46	0.15	185.3	243.0	39.9	0.74	182.47	0.15	259.3	2.9	118.4
7	63.1	0.53	134.67	0.12	174.4	237.5	49.9	0.63	163.15	0.15	271.1	3.3	142.8
8	61.2	0.49	128.28	0.12	174.3	235.5	59.9	0.52	147.62	0.14	268.5	3.5	156.6
9	79.9	0.44	102.43	0.10	156.3	236.2	59.8	0.52	126.57	0.12	270.9	3.0	137.0
10	78.2	0.41	96.23	0.09	158.2	236.4	59.9	0.48	116.80	0.11	246.9	2.8	122.3
11	74.5	0.36	79.27	0.08	96.7	171.2	59.9	0.42	90.02	0.09	214.6	2.2	102.9
12	77.1	0.38	80.36	0.08	74.3	151.5	59.9	0.44	92.14	0.09	225.4	2.2	103.8
13	73.9	0.39	77.01	0.07	67.4	141.3	59.9	0.45	84.64	0.08	227.2	2.0	92.3
14	72.5	0.40	83.89	0.07	67.6	140.1	59.9	0.45	90.85	0.08	231.0	2.2	92.8
15	76.6	0.38	93.84	0.08	63.9	140.5	59.9	0.45	101.85	0.09	231.3	2.4	107.2
16	80.0	0.37	110.48	0.08	59.6	139.7	59.9	0.45	117.28	0.09	230.1	2.8	105.4
17	75.8	0.34	107.37	0.08	56.1	132.0	59.8	0.40	119.66	0.10	203.2	2.9	109.7
18	79.8	0.37	103.36	0.07	50.9	130.8	59.9	0.44	108.84	0.08	227.4	2.6	92.7
19	77.6	0.35	114.73	0.07	36.3	113.9	59.9	0.41	119.37	0.08	208.8	2.9	95.7
20	78.4	0.36	128.68	0.07	34.3	112.7	59.9	0.43	135.04	0.08	219.6	3.2	92.6
21	75.8	0.37	135.20	0.08	27.7	103.5	59.8	0.43	138.43	0.09	221.2	3.3	107.5
22	80.4	0.37	151.16	0.08	23.2	103.6	59.9	0.44	152.79	0.09	230.1	3.7	102.8
23	67.8	0.39	156.85	0.07	14.4	82.2	59.9	0.43	164.26	0.08	221.8	3.9	92.6
24	65.3	0.43	167.85	0.07	10.8	76.1	59.9	0.46	173.23	0.08	238.9	4.1	88.2
25	67.7	0.44	158.83	0.07	8.0	75.8	59.9	0.48	160.99	0.08	248.5	3.9	90.7
26	59.2	0.49	161.29	0.08	5.5	64.8	59.9	0.48	160.91	0.08	251.3	3.9	91.9
27	28.4	0.56	187.44	0.08	0.7	29.1	59.9	0.35	133.30	0.06	181.8	3.2	71.2
28							59.9	0.15	72.89	0.04	72.5	1.7	45.7
29							60.0	0.15	72.89	0.04	72.7	1.7	45.8
30							59.9	0.15	72.89	0.04	72.5	1.7	45.7
31							59.9	0.15	72.89	0.04	72.5	1.7	45.7
32							42.7	0.15	72.89	0.04	51.7	1.2	32.6
33													
Total	1,758.5	0.44	121.40	0.09	2,784.5	4,543.0	1,758.5	0.44	121.40	0.09	6,573.1	85.4	3,052.0

23.1.2 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 85.5% to a concentrate grade of 25.3 %Cu
- molybdenum recovery of 40% to a molybdenum concentrate grade of 47% Mo
- gold recovery to the copper concentrate of 60%, with a grade of approximately 4.5 g/t Au.

To reflect the modelling of initial cashflows for the Cobre Panamá project, copper recovery ramp-up factors were also applied to the Taca Taca model, as follows:

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- Year 1 = 85% of the modelled recovery; Year 2 = 92% of the modelled recovery and Year 3 = 98% of the modelled recovery.

The impact of this factoring is a reduction in the overall average life of mine copper recovery from 85.5% to 85.0%.

23.1.1 Capital costs

The total estimated capital costs for the Project were split into initial (i.e., development) capital expenditure and subsequent plant expansion expenditure, as listed in Table 23-5 and Table 23-6 for direct and indirect costs, respectively.

The initial plant throughput will be 40 Mtpa and this will be ramped up to 60 Mtpa in Year 8. It was assumed that the majority of the process facilities and infrastructure would be built initially, with the exceptions described in Section 21.2.2, deferred until Year 7. Certain items of infrastructure, i.e. the water supply borefields, plus expansion indirect costs were also deferred until Years 6 and 7.

Table 23-5 Updated Project capital cost estimates; direct costs split between initial and expansion capital; TSF expansion is a sustaining cost

CAPITAL COST ESTIMATES	Initial Costs (\$)	Expansion Costs (\$)
DIRECT COSTS		
Mining		
primary mining equipment	\$331,893,782	
mining ancillaries and technical services	\$87,701,832	
preproduction stripping	\$311,099,501	
Subtotal	\$730,695,115	\$0
Processing		
crushing, conveying & storage	\$66,498,900	\$21,000,000
secondary crushing circuit		\$63,300,000
grinding & concentrator	\$771,042,200	\$100,000,000
concentrate thickening, filtration, storage, handling	\$54,286,900	
reagents/consumables storage/distribution/handling	\$20,633,300	
Subtotal	\$912,461,300	\$184,300,000
Tailings Management		
TSF construction	\$11,988,750	a sustaining cost
Subtotal	\$11,988,750	\$0
Infrastructure		
access road through Argentina	\$28,750,000	
access road through Chile	\$0	
railway, maintenance facilities and airstrip:		
railway upgrade	\$179,867,988	
railway rolling stock and related equipment	\$93,354,466	
rail load-out	\$21,141,161	
port upgrade/expansion	\$80,000,000	
water supply borefield	\$60,350,186	\$8,002,939
water supply pumps and pipeline	\$35,831,891	\$4,751,609
borefield power supply	\$57,161,970	\$7,580,155
process water treatment, storage and distribution	\$3,436,503	\$455,709
power transmission line	\$106,432,970	
power line substations	\$37,163,500	
internal power distribution	\$20,602,997	
site earthworks	\$72,763,704	
other infrastructure:	rolled-up:	
camp		
administration building		
office/engineering equipment, software, furniture		
laboratory incl. equipment and met lab		
mess/kitchen, warehouse, workshops, sewage etc	\$130,200,000	
plant & mine warehouse/truck shop equipment		
medical, safety, security, communication		
site security and fencing		
unspecified site and off-site facilities		
Subtotal	\$927,057,335	\$39,646,256
Other Costs		
ancillary, plant mobile equipment, light vehicles	\$54,498,500	
Subtotal	\$54,498,500	\$0
Total Costs		
Subtotal Direct Costs	\$2,636,701,000	\$223,946,256

Table 23-6 Updated Project capital cost estimates; indirect costs split between initial and expansion capital

CAPITAL COST ESTIMATES	Initial Costs (\$)	Expansion Costs (\$)
INDIRECT COSTS		
Owners costs		
preproduction employment & training	\$31,005,150	
project & construction management	\$16,943,235	\$2,246,815
operations catering	\$12,000,250	
camp power	\$5,935,150	
ROW, land acquisition, legal, permits, fees		included in Other Costs
insurance		
corporate travel & services	\$1,457,050	
environmental	\$1,552,500	
medical, security, communication	\$1,437,500	
community development	\$1,610,000	
geotechnical facilities	\$8,791,750	
third party inspections/testing	\$7,762,500	
vendor representatives/commissioning assistance	\$51,831,095	
spare parts/consumables/initial fills	\$63,319,877	
Subtotal	\$203,646,057	\$25,505,143
Construction		
contractor indirects	\$35,827,830	\$4,751,070
construction temporary facilities	\$44,574,101	\$5,910,899
construction equipment	\$3,138,463	\$416,187
construction camp	\$102,630,000	\$10,300,000
Subtotal	\$186,170,395	\$21,378,155
Contractor		
EPCM services	\$115,830,000	\$12,870,000
geotechnical facilities		
third party inspections/testing		Included in Owners Costs
vendor representatives/commissioning assistance		
spare parts/consumables/initial fills		
Subtotal	\$115,830,000	\$12,870,000
Other Costs		
freight, duties & taxes	\$109,394,000	\$22,406,000
ROW, land acquisition, legal, permits, fees	\$7,948,800	
insurance	\$15,138,949	\$2,007,551
Subtotal	\$132,481,749	\$24,413,551
Total Costs		
Subtotal Direct Costs	\$638,128,201	\$84,166,849
TOTAL PROJECT CAPITAL		
Total Costs	\$3,274,829,201	\$308,113,105

23.1.2 Sustaining and closure costs

Item 21.3 describes the explicit annual charges for on-going mining equipment replacement over the life of the Project. These are included in the cashflow model and amount to \$875.1 M, or an average of \$32.4 Mpa over 27 years. An amount of \$56.5 M (inclusive of 15% contingency) is also included in the model for TSF expansion works between Years 15 and 32 (as estimated and recommended by Hillerton (2019)).

A further annual sustaining charge is provided to cover processing plant and infrastructure replacement, on the basis of 4.5% of the annual processing and G&A operating costs.

Table 23-7 lists the annual allocation into the cashflow model of the Project closure cost provisions (i.e., Years 27 to 33), as described in Item 21.4.

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Table 23-7 Project closure (annual) cost allocation

Item	Description	Subtotal \$	Proportion (%)	27 \$	28 \$	29 \$	30 \$	31 \$	32 \$	33 \$	34 \$	35 \$
1	Provisional and preliminary works	\$223,000	1.4%	\$223,000								
2	CM1: Filling Taco Chico Pit	\$872,036	5.3%							\$872,036		
3	CM2: Taca Taca Open Pit	\$3,666,959	22.2%							\$3,666,959		
4	CM5: Tailings storage facility	\$2,533,734	15.3%	\$422,289	\$422,289	\$422,289	\$422,289	\$422,289	\$422,289			
5	CM3: East waste dump	\$5,007,600	30.3%	\$834,600	\$834,600	\$834,600	\$834,600	\$834,600	\$834,600			
6	CM4: West waste dump	\$963,000	5.8%	\$160,500	\$160,500	\$160,500	\$160,500	\$160,500	\$160,500			
7	CM6: Ore stockpile	\$18,523	0.1%						\$18,523			
8	CM8: Solid waste landfill	\$223,904	1.4%						\$223,904			
9	CM7: Process plant	\$2,305,682	14.0%							\$2,305,682		
10	CM7: Primary crushing plants	\$20,000	0.1%							\$20,000		
11	CM7: Camp	\$78,830	0.5%							\$78,830		
12	CM7: Mining services area	\$52,412	0.3%							\$52,412		
	CM7: Fuel bay		0.0%									
	CM7: Scrap metal, workshops		0.0%									
13	CM7: Power lines	\$20,000	0.1%							\$20,000		
14	CM7: Explosives magazine	\$245,824	1.5%							\$245,824		
15	CM9: Roads	\$58,000	0.4%							\$58,000		
16	CM10: Railway	\$20,000	0.1%							\$20,000		
	CM11: Airfield		0.0%									
17	Post closure activities	\$206,760	1.3%							\$206,760		
	Total Direct Costs	\$16,516,264	100.0%	\$1,640,389	\$1,417,389	\$1,417,389	\$1,417,389	\$1,417,389	\$1,659,816	\$7,546,503	\$0	\$0
	General expenses (25%)	\$4,129,066		\$410,097	\$354,347	\$354,347	\$354,347	\$354,347	\$414,954	\$1,886,626		
	Profit (10%)	\$1,651,626		\$164,039	\$141,739	\$141,739	\$141,739	\$141,739	\$165,982	\$754,650		
	Subtotal	\$5,780,692		\$574,136	\$496,086	\$496,086	\$496,086	\$496,086	\$580,936	\$2,641,276	\$0	\$0
	Supervision (7%)	\$1,560,787		\$155,017	\$133,943	\$133,943	\$133,943	\$133,943	\$156,853	\$713,145		
	Owners costs (3%)	\$668,909		\$66,436	\$57,404	\$57,404	\$57,404	\$57,404	\$67,223	\$305,633		
	Total Cost	\$2,229,696		\$221,453	\$191,348	\$191,348	\$191,348	\$191,348	\$224,075	\$1,018,778	\$0	\$0
	Allowance for complementary studies e.g., ARD, salt formaiton, hydrology, geotechnics (0.1%)	\$24,527		\$2,436	\$2,105	\$2,105	\$2,105	\$2,105	\$2,465	\$11,207		
	Engineering cost (0.1%)	\$24,527		\$2,436	\$2,105	\$2,105	\$2,105	\$2,105	\$2,465	\$11,207		
	Contingency (40%)	\$9,830,282		\$976,340	\$843,613	\$843,613	\$843,613	\$843,613	\$987,903	\$4,491,588		
	Total Costs	\$9,879,335		\$981,212	\$847,823	\$847,823	\$847,823	\$847,823	\$992,832	\$4,514,001	\$0	\$0
	Grand Total Cost	\$34,405,987		\$3,417,189	\$2,952,645	\$2,952,645	\$2,952,645	\$2,952,645	\$3,457,659	\$15,720,558	\$0	\$0

23.1.3 Operating and metal costs

The overall average unit operating costs in the Mineral Reserve cashflow model are:

- mining ore and waste = \$1.69/t
 - ore mining, excluding the pre-strip period = \$1.82/t (ranging annually between \$1.33/t and \$2.45/t)
 - waste mining, excluding the pre-strip period = \$1.45/t (ranging annually between \$1.34/t and \$2.53/t)
- stockpile reclaim = \$0.74/t reclaimed
- processing = \$4.69/t processed (not profiled)
- rail load-out infrastructure and water supply tariff = \$0.08/t processed (not profiled)
- G&A = \$1.05/t processed (not profiled)

The overall average metal costs (including treatment charges, refining charges and royalties) are:

- copper = \$0.52/lb (including a net royalty charge equating to \$0.09/lb)
- molybdenum = \$0.56/lb (including a net royalty charge equating to \$0.38/lb)
- gold = \$56.01/oz (including a net royalty charge equating to \$51.42/oz)

23.1.4 Capital depreciation

The Company's internal taxation advisers provided notional 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.

23.2 Cashflow model outcomes

The summary cashflow model for the economic analysis supporting the Mineral Reserve estimate is listed in Table 23-8.

On a pre-tax basis:

- the undiscounted cashflow for the Mineral Reserve production schedule is \$17,306.3 M
- the NPV reflecting an 8% discount rate is \$3,428.8 M
- the internal rate of return is 17.4%
- the Project is cashflow positive from Year 2 and payback on the initial development capital is in Year 9

On a post-tax basis:

- the estimated taxable income is estimated as \$17,325.1 M
- the total tax payable on this amount is \$4,331.3 M, yielding an undiscounted post-tax cashflow for the Mineral Reserve production schedule of \$12,975.0
- the NPV reflecting an 8% discount rate is \$2,361.2 M
- the internal rate of return is 15.3%
- the Project is cashflow positive from Year 2 and payback on the initial development capital is in Year 11

Table 23-8 Mineral Reserve cashflow model summary

PHYSICALS		UNITS	TOTAL	Yr-3	Yr-2	Yr-1	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21	Yr22	Yr23	Yr24	Yr25	Yr26	Yr27	Yr28	Yr29	Yr30	Yr31	Yr32	Yr33	
MINING (AFTER MINING DILUTION & RECOVERY)																																								
Ore mined direct to plant	Mt	1,390.4	0.0	0.0	0.0	23.6	30.9	29.6	37.4	39.9	39.9	49.9	51.2	59.8	59.9	56.0	59.9	59.1	58.3	59.2	59.9	56.7	59.9	59.4	59.9	59.8	59.9	55.7	56.3	59.9	59.9	28.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
Ore mined to stockpile	Mt	368.1	0.0	3.6	13.9	10.0	9.4	5.3	0.0	3.8	17.7	12.0	10.7	20.1	17.6	17.4	17.0	15.2	13.9	16.8	20.4	19.5	21.5	17.7	19.2	16.4	20.9	11.0	8.4	7.7	0.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
Ore reclaimed from stockpile	Mt	368.1	0.0	0.0	0.0	6.4	9.0	10.3	2.5	0.0	0.0	0.0	0.0	8.6	0.0	0.0	3.8	0.0	0.7	1.6	0.7	0.0	3.2	0.0	0.4	0.0	0.0	4.2	3.5	0.0	0.0	31.0	59.9	60.0	59.9	59.9	59.9	42.7		
Waste mined to dump	Mt	2,784.5	43.5	97.1	99.5	171.2	205.4	210.8	208.7	202.2	185.3	174.4	174.3	156.3	158.2	96.7	74.3	67.4	67.6	63.9	59.6	56.1	50.9	36.3	34.3	27.7	23.2	14.4	10.8	8.0	5.5	0.7	0.0	0.0	0.0	0.0	0.0	0.0		
FEED TO PLANT (AFTER MINING DILUTION & RECOVERY)																																								
Total direct feed	Mt	1,390.4	0.0	0.0	0.0	23.6	30.9	29.6	37.4	39.9	39.9	49.9	51.2	59.8	59.9	56.0	59.9	59.1	58.3	59.2	59.9	56.7	59.9	59.4	59.9	59.8	59.9	55.7	56.3	59.9	59.9	28.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
	% Cu	0.50	0.00	0.00	0.00	0.55	0.71	0.81	0.74	0.79	0.74	0.63	0.55	0.52	0.48	0.43	0.44	0.45	0.46	0.45	0.45	0.40	0.44	0.41	0.43	0.43	0.44	0.44	0.47	0.48	0.48	0.56	0.00	0.00	0.00	0.00	0.00	0.00		
	ppm Mo	131.59	0.00	0.00	0.00	118.41	100.49	126.68	145.10	178.34	182.47	163.15	146.78	126.57	116.80	89.95	92.14	84.56	90.84	101.97	117.28	119.52	108.84	119.35	135.04	138.43	152.79	163.26	172.70	160.99	160.91	186.93	0.00	0.00	0.00	0.00	0.00	0.00		
	g/t Au	0.10	0.00	0.00	0.00	0.16	0.13	0.12	0.12	0.17	0.15	0.15	0.13	0.12	0.11	0.09	0.09	0.08	0.08	0.09	0.09	0.09	0.08	0.08	0.08	0.09	0.09	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.00	0.00	0.00	0.00	0.00	
Total reclaim feed	Mt	368.1	0.0	0.0	0.0	6.4	9.0	10.3	2.5	0.0	0.0	0.0	8.6	0.0	0.0	3.8	0.0	0.7	1.6	0.7	0.0	3.2	0.0	0.4	0.0	0.0	4.2	3.5	0.0	0.0	31.0	59.9	60.0	59.9	59.9	59.9	42.7			
	% Cu	0.20	0.00	0.00	0.00	0.38	0.37	0.87	1.03	0.00	0.00	0.00	0.37	0.00	0.00	0.34	0.00	0.34	0.34	0.34	0.00	0.29	0.00	0.28	0.00	0.00	0.00	0.27	0.27	0.00	0.00	0.16	0.15	0.15	0.15	0.15	0.15			
	ppm Mo	82.93	0.00	0.00	0.00	124.98	127.29	132.60	121.76	0.00	0.00	0.00	152.61	0.00	0.00	91.12	0.00	91.32	91.32	91.32	0.00	122.25	0.00	122.75	0.00	0.00	0.00	177.60	181.70	0.00	0.00	83.36	72.89	72.89	72.89	72.89	72.89			
	g/t Au	0.05	0.00	0.00	0.00	0.16	0.16	0.14	0.12	0.00	0.00	0.00	0.15	0.00	0.00	0.11	0.00	0.11	0.11	0.11	0.00	0.11	0.00	0.10	0.00	0.00	0.00	0.07	0.07	0.00	0.00	0.04	0.04	0.04	0.04	0.04				
Total plant feed	Mt	1,758.5	0.0	0.0	0.0	29.9	41.9	41.9	41.9	41.9	51.9	51.9	59.9	59.9	59.1	59.9	59.9	59.8	59.9	59.9	59.9	59.9	59.9	59.9	59.9	59.8	59.9	59.9	59.9	59.8	59.9	59.9	60.0	59.9	59.9	59.9	59.9	21.5		
	% Cu	0.44	0.00	0.00	0.00	0.52	0.54	0.87	0.72	0.69	0.67	0.57	0.62	0.51	0.48	0.44	0.47	0.45	0.44	0.46	0.43	0.40	0.43	0.44	0.45	0.45	0.42	0.37	0.42	0.50	0.48	0.46	0.30	0.15	0.15	0.15	0.15	0.15		
	ppm Mo	121.40	0.00	0.00	0.00	120.47	94.12	135.04	139.89	165.52	179.70	158.85	153.75	123.89	99.58	107.86	74.93	89.38	95.25	105.11	116.44	110.13	120.05	126.04	151.81	137.93	143.28	150.99	170.44	162.60	164.02	117.35	73.28	73.28	73.28	73.28	73.28			
	g/t Au	0.09	0.00	0.00	0.00	0.16	0.11	0.13	0.13	0.15	0.16	0.14	0.14	0.11	0.10	0.09	0.08	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.08	0.08	0.08	0.08	0.08	0.06	0.04	0.04	0.04	0.04				
Cu insitu	kt	7,734.7	0.0	0.0	0.0	154.3	251.9	328.5	302.5	316.3	296.9	313.8	313.0	314.1	287.5	253.6	266.1	267.6	270.9	271.0	268.8	237.4	265.1	243.3	254.9	256.4	266.1	256.9	275.2	285.9	289.0	211.0	88.4	88.6	88.4	88.4	88.4	63.0		
Mo insitu	kt	213.5	0.0	0.0	0.0	3.6	4.3	5.1	5.7	7.1	7.3	8.1	8.8	7.6	7.0	5.4	5.5	5.1	5.4	6.1	7.0	7.2	6.5	7.1	8.1	8.3	9.1	9.8	10.4	9.6	9.6	8.0	4.4	4.4	4.4	4.4	4.4	3.1		
Au insitu	k(oz)	5,086.7	0.0	0.0	0.0	153.1	173.4	164.1	151.5	223.5	197.3	238.0	261.0	228.3	203.8	174.4	173.1	153.8	154.6	178.7	175.7	182.9	154.4	159.4	154.3	179.2	171.3	154.4	146.9	151.2	153.1	118.6	76.2	76.4	76.4	76.4	76.4	54.3		
AVERAGE RECOVERIES																																								
Copper recovery	%	85.5%	0.0%	0.0%	0.0%	74.3%	83.4%	85.5%	86.6%	86.8%	87.4%	86.4%	85.8%	86.2%	85.9%	84.6%	84.7%	84.9%	85.3%	85.3%	85.6%	85.6%	85.8%	85.8%	86.2%	86.3%	86.5%	86.3%	86.8%	86.9%	86.9%	86.1%	82.0%	82.0%	82.0%	82.0%	82.0%	82.0%		
Copper ramp-up factor	%	99.4%	0.0%	0.0%	0.0%	85.0%	92.0%	98.0%	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%	100.0%	100.0%	100.0%		
Adjusted copper recovery	%	85.0%	0.0%	0.0%	0.0%	63.2%	76.7%	83.8%	86.6%	86.8%	87.4%	86.4%	85.8%	86.2%	85.9%	84.6%	84.7%	84.9%	85.3%	85.3%	85.6%	85.6%	85.8%	85.8%	86.2%	86.3%	86.5%	86.3%	86.8%	86.9%	86.9%	86.1%	82.0%	82.0%	82.0%	82.0%	82.0%	82.0%		
Molybdenum recovery	%	40.0%	0.0%	0.0%	0.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%	40.0%			
Gold recovery	%	60.0%	0.0%	0.0%	0.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%				
METAL RECOVERED																																								
Unadjusted Cu recovered	kt	6,612.7	0.0	0.0	0.0	114.7	210.0	280.9	262.1	274.6	259.3	271.1	268.5	270.9	246.9	214.6	225.4	227.2	231.0	231.3	230.1	203.2	227.4	208.8	219.6	221.2	230.1	221.8	238.9	248.5	251.3	181.8	72.5	72.7	72.5	72.5	72.5	51.7		
Adjusted Cu recovered	kt	6,573.1	0.0	0.0	0.0	97.5	193.2	275.2	262.1	274.6	259.3	271.1	268.5	270.9	246.9	214.6	225.4	227.2	231.0	231.3	230.1	203.2	227.4	208.8	219.6	221.2	230.1	221.8	238.9	248.5	251.3	181.8	72.5	72.7	72.5	72.5	72.5	51.7		
Mo recovered	kt	85.4	0.0	0.0	0.0	1.4	1.7	2.0	2.3	2.8	2.9	3.3	3.5	3.0	2.8	2.2	2.2	2.0	2.2	2.4	2.8	2.9	2.6	2.9	3.2	3.3	3.7	3.9	4.1	3.9	3.9	3.2	1.7	1.7	1.7	1.7	1.7	1.2		
Au recovered	k(oz)	3,052.0	0.0	0.0	0.0	91.8	104.1	98.5	90.9	134.1	118.4	142.8	156.6	137.0	122.3	102.9	103.8	92.3	92.8	107.2	105.4	109.7	92.7	95.7	92.6	107.5	102.8	92.6	88.2	90.7										

23.3 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 pre-tax cashflow model summarised in Table 23-8, Table 23-9 shows a list of variables for testing the cashflow model sensitivity over respective indicated ranges.

Figure 22-1 shows a tornado chart of the NPV impact (assuming an 8% discount rate) due to the listed sensitivity variables and ranges thereof. This analysis confirms the high sensitivity associated with copper metal price and processing recovery. Figure 23-2 shows a corresponding IRR tornado chart.

Table 23-9 Discounted cashflow model sensitivity analysis

Sensitivity parameter	Base	Minimum	Maximum
Cu metal price (\$/lb)	\$3.00	92% \$2.75	112% \$3.35
Au metal price (\$/oz)	\$1,500	87% \$1,300	113% \$1,700
Cu recovery (%)	85.0%	90% 76.5%	110% 93.5%
Au recovery (%)	40.0%	90% 36.0%	110% 44.0%
Mo recovery (%)	60.0%	90% 54.0%	110% 66.0%
Ore mining (\$/t mined)	\$1.82	90% \$1.64	110% \$2.01
Waste mining (\$/t mined)	\$1.45	90% \$1.31	110% \$1.60
Ore processing (\$/t processed)	\$4.69	90% \$4.22	110% \$5.16
G&A operating costs (\$/t processed)	\$1.05	90% \$0.95	110% \$1.16
Copper treatment costs (\$/dmt)	\$90.00	90% \$81.00	110% \$99.00
Sea freight costs (\$/wmt)	\$48.50	90% \$43.65	110% \$53.35
Development capital costs (\$M)	\$3,274.8	80% \$2,620	120% \$3,930
Expansion capital costs (\$M)	\$308.1	80% \$246	120% \$370
Sustaining capital costs (\$M)	\$1,304.1	80% \$1,043	120% \$1,565

Figure 23-1 Project sensitivity analysis, NPV₈ tornado chart

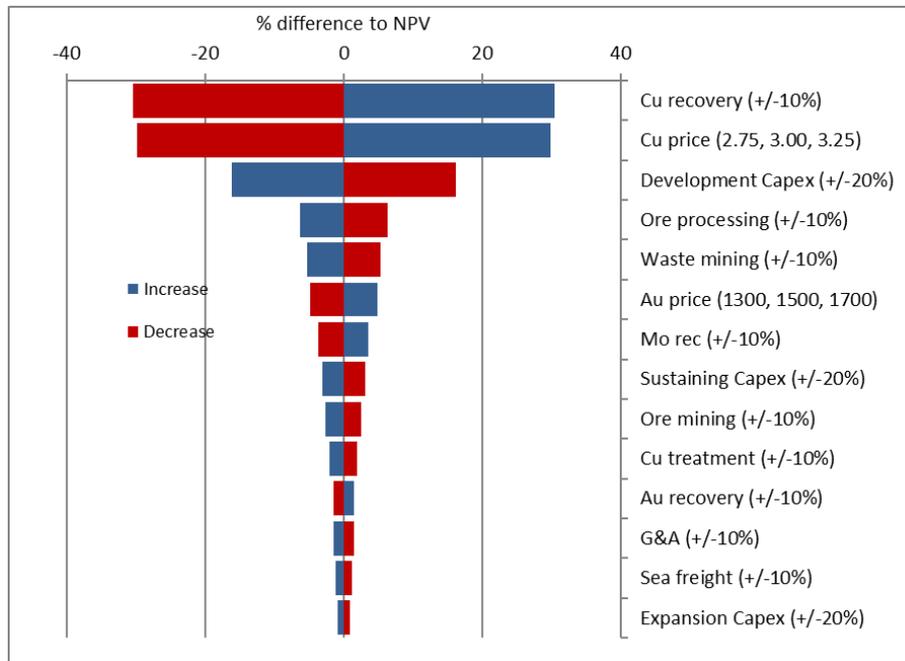
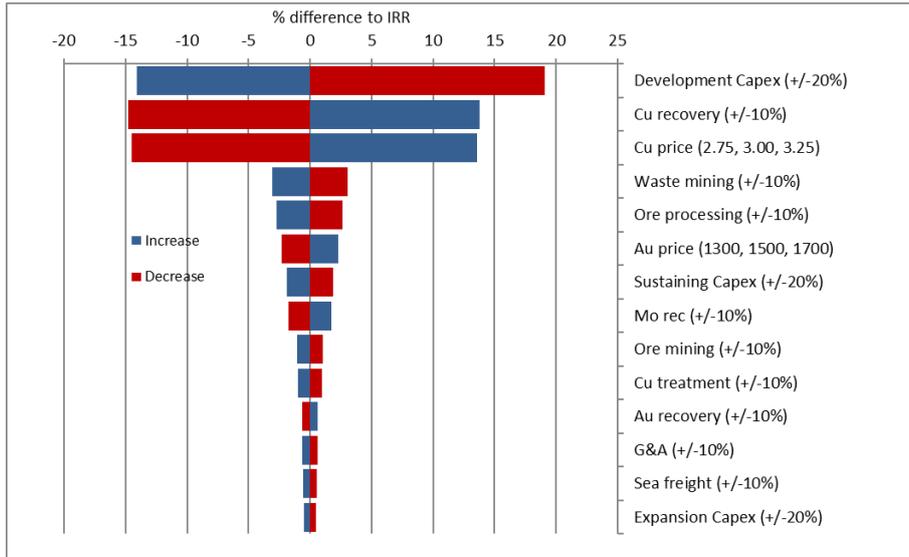
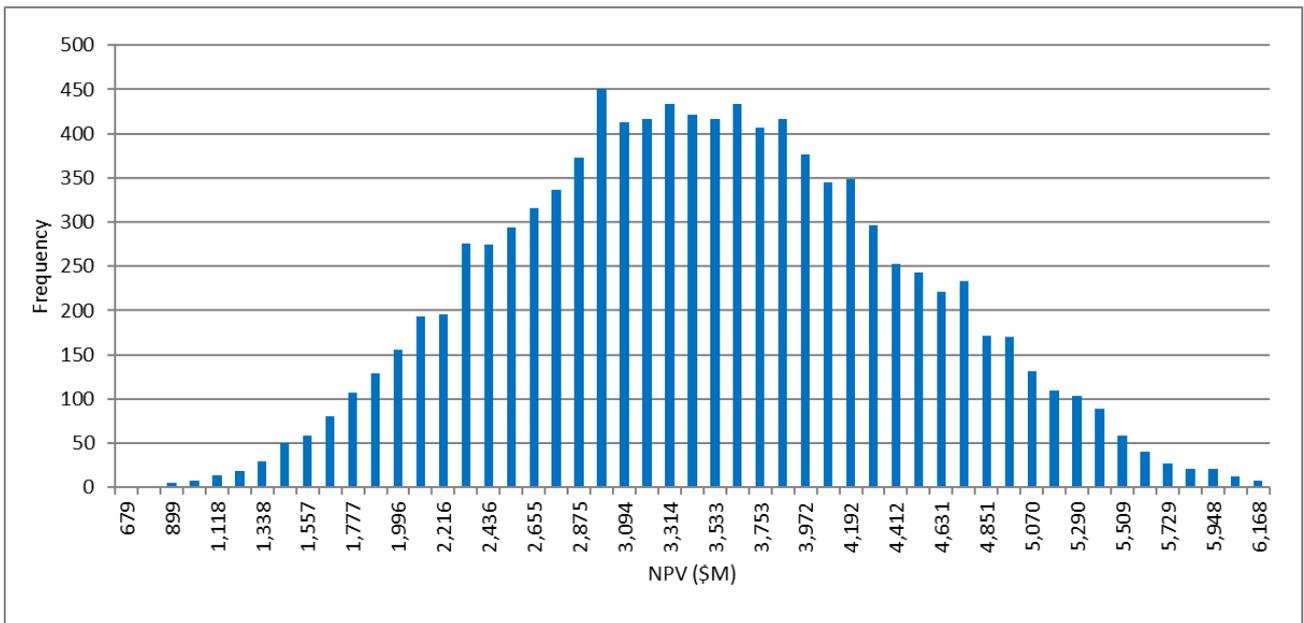


Figure 23-2 Project sensitivity analysis, IRR tornado chart



Figures 22-3 and 22-4 show, respectively, a Monte Carlo simulation distribution of Project NPV and IRR, derived from 10,000 iterations of the cashflow model, through the variances listed in Table 23-9. This simulation indicated a 70% probability of the Project NPV₈ exceeding \$3,000 M and a 72% probability of the IRR exceeding 16% (Figure 23-5), on a pre-tax cashflow basis.

Figure 23-3 Monte Carlo simulation of Project NPV₈



An additional simulation indicated a 70% probability of the Project NPV₈ exceeding \$2,000 M and a 72% probability of the IRR exceeding 14%, on a post-tax cashflow basis.

Figure 23-4 Monte Carlo simulation on Project IRR

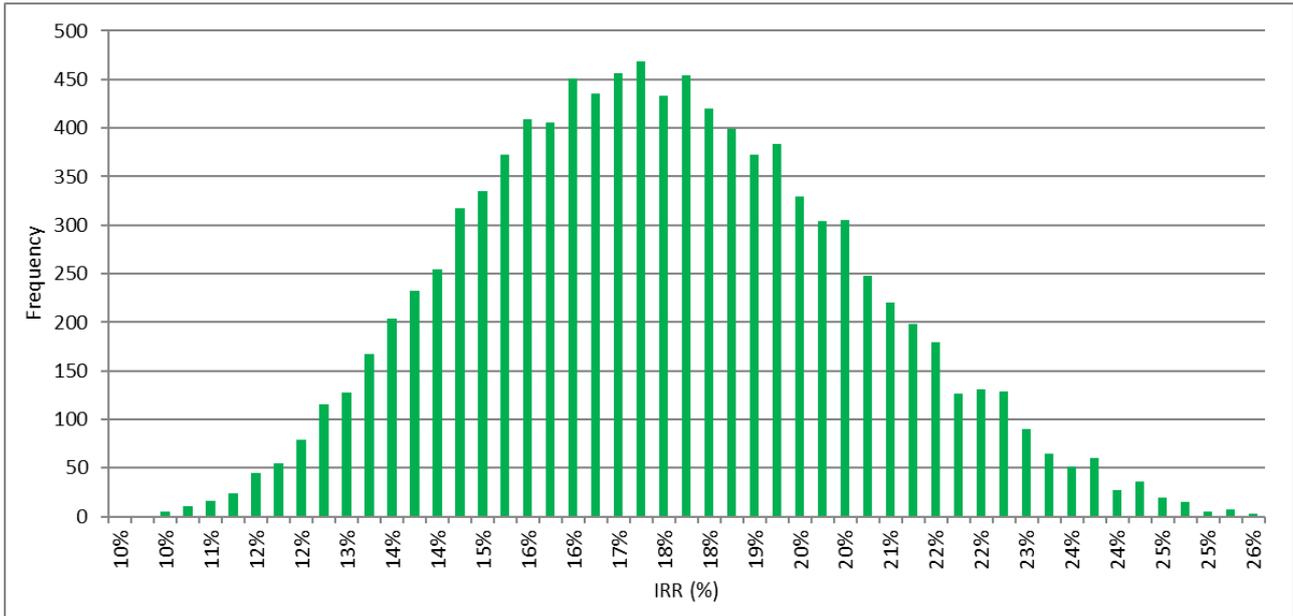
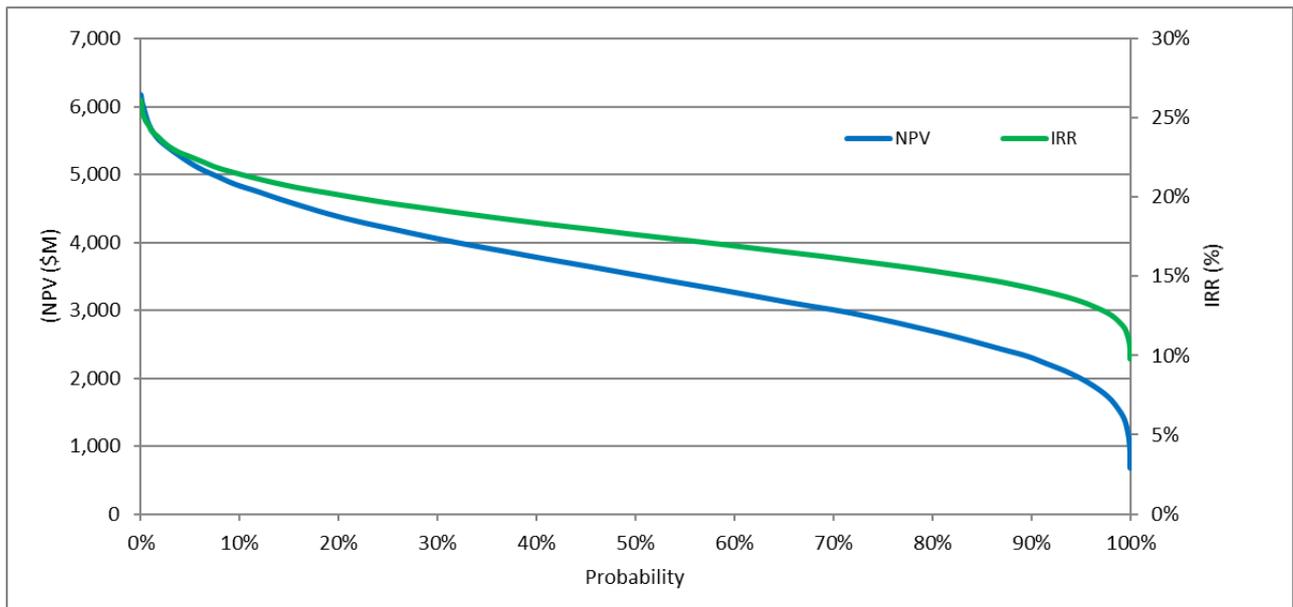


Figure 23-5 NPV8 and IRR probability chart



ITEM 24 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 25 OTHER RELEVANT DATA AND INFORMATION

25.1 Introduction

The supply of water in such an arid environment as the Argentine Altiplano is considered to be a critical engineering aspect for development of the Project. In the absence of a separate Technical Report item under which to report on this important aspect, the following commentary is included here to provide an update on the water supply investigations which remain in progress.

Consistent with the Project infrastructure requirements described in Item 18, there is considered to be an infinite source of brine water for processing; supply would come primarily from a number of bores on the Salar de Arizaro located in close proximity to the processing plant, with further brine likely to be supplied from pit slope depressurisation wells and drains. A substantial volume of fresh water will also be required, for potable and for mining and processing requirements, the sustainable sources for which are the subject of continuing exploration and quantification efforts.

Figure 25-1 shows the location of priority watersheds and potential fresh water supply sources in the Project vicinity, along with the location of other prospective watersheds.

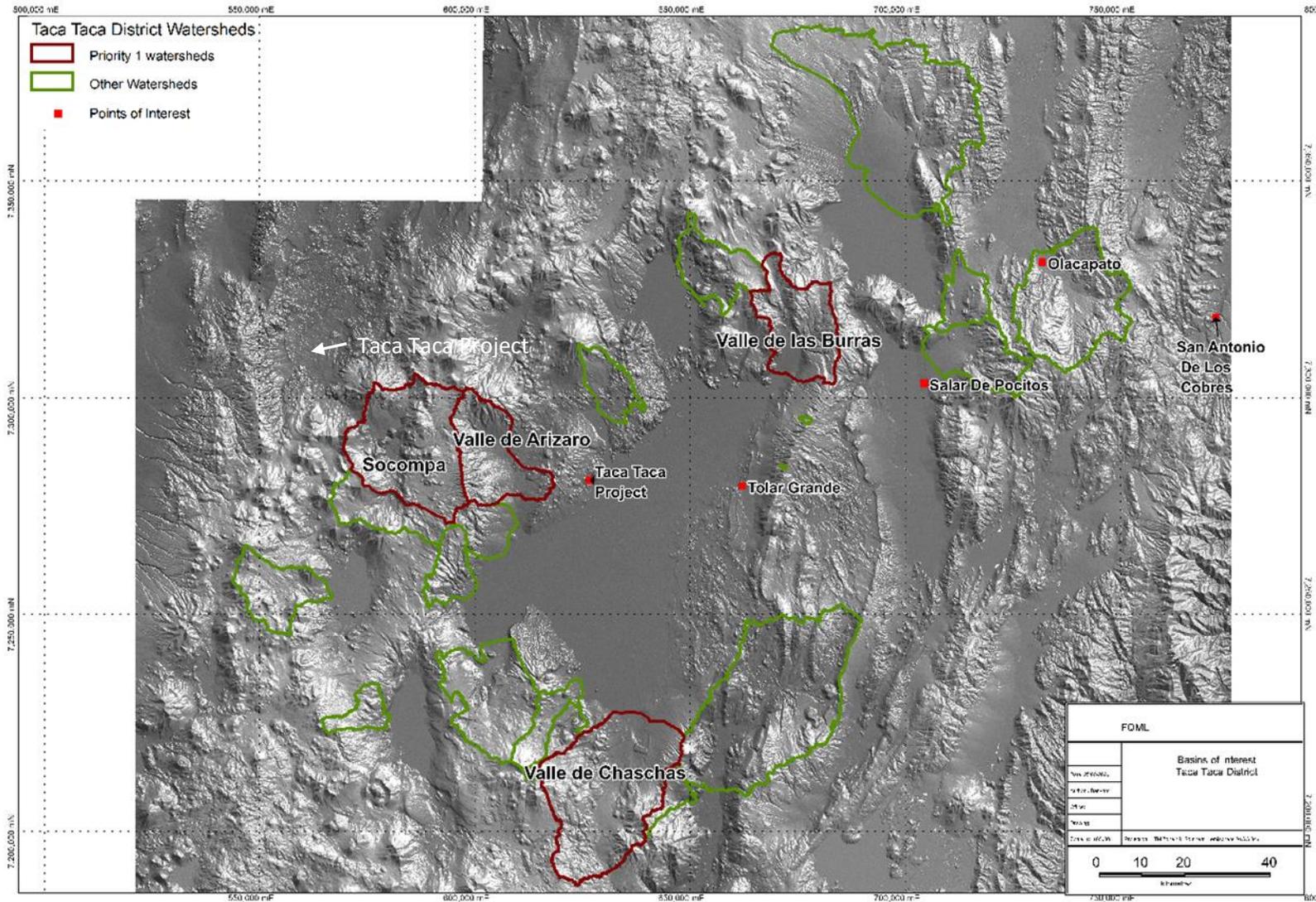
25.2 Water supply requirements

Table 25-1 is reproduced from Item 18 and itemises an average and peak fresh water demand of 431.5 L/sec and 434.9 L/sec, respectively. The average and peak brine water demand is 1,512.6 L/sec and 1,764.7 L/sec, respectively

Table 25-1 Summary water demand for all consumption activities

Water Demand	Average				Peak			
	ML/annum	kl/day	m ³ /h	L/s	ML/annum	kl/day	m ³ /h	L/s
Fresh water demand								
for processing	13,543.0	37,104.0	1,546.0	429.4	13,543.0	37,104.0	1,546.0	429.4
for the camp	52.0	142.5	5.9	1.6	127.0	348.0	14.5	4.0
for mining	5.3	14.5	0.6	0.2	21.9	60.0	2.5	0.7
for site services, rail load-out & construction	8.4	23.0	1.0	0.3	21.9	60.0	2.5	0.7
Subtotal	13,608.7	37,284.0	1,553.5	431.5	13,713.8	37,572.0	1,565.5	434.9
Brine demand								
for processing	39,770.4	108,960.0	4,540.0	1,261.1	39,770.4	108,960.0	4,540.0	1,261.1
for mining	7,899.4	21,642.2	901.8	250.5	15,798.8	43,284.4	1,803.5	501.0
for road maintenance & construction	32.5	89.0	3.7	1.0	81.2	222.5	9.3	2.6
Subtotal	47,702.3	130,691.2	5,445.5	1,512.6	55,650.4	152,466.9	6,352.8	1,764.7
Water in ore processed	2,032.3	5,568.0	232.0	64.4	2,032.3	5,568.0	232.0	64.4
TOTAL	63,343.3	173,543.2	7,231.0	2,008.6	71,396.5	195,606.9	8,150.3	2,264.0
Processing summary								
fresh make-up	13,543.0	37,104.0	1,546.0	429.4	13,543.0	37,104.0	1,546.0	429.4
brine make-up	39,770.4	108,960.0	4,540.0	1,261.1	39,770.4	108,960.0	4,540.0	1,261.1
water in ore processed	2,032.3	5,568.0	232.0	64.4	2,032.3	5,568.0	232.0	64.4
Subtotal	55,345.7	151,632.0	6,318.0	1,755.0	55,345.7	151,632.0	6,318.0	1,755.0
Mining summary								
fresh	5.3	14.5	0.6	0.2	21.9	60.0	2.5	0.7
brine	7,899.4	21,642.2	901.8	250.5	15,798.8	43,284.4	1,803.5	501.0
Subtotal	7,904.7	21,656.7	902.4	250.7	15,820.7	43,344.4	1,806.0	501.7
Camp and other								
fresh	60.4	165.5	6.9	1.9	148.9	408.0	17.0	4.7
brine	32.5	89.0	3.7	1.0	81.2	222.5	9.3	2.6
Subtotal	92.9	254.5	10.6	2.9	230.1	630.5	26.3	7.3
TOTAL	63,343.3	173,543.2	7,231.0	2,008.6	71,396.5	195,606.9	8,150.3	2,264.0

Figure 25-1 Principal watersheds in the vicinity of the Taca Taca Project



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Table 25-2 lists the borefield supply contributions to the overall Project water balance. A highlighted blue line shows a peak brine requirement of 6,157 m³/h, which is equivalent to 1,710 L/s supply. The other highlighted line shows the projected supply capability from the identified fresh water supply borefields as 3,600 m³/h or 1,000 L/s. This represents contingency over the actual Project fresh water requirements of 1,565 m³/h or 435 L/s.

Table 25-2 Summary water balance showing borefield supply contributions

Overall Water Balance	Average m³/h	Peak m³/h
Fresh water In		
regional borefields	1,972.5	3,600.0
Subtotal	1,972.5	3,600.0
To Operations		
process plant	1,546.0	1,546.0
mine operations	0.6	1.8
camp. etc	6.8	17.3
Subtotal	1,553.4	1,565.2
Water consumption		
process plant		
in concentrates	12.0	12.0
in tailings	768.5	768.5
evaporation at TSF	765.5	765.5
mine operations		
to sewage treatment	5.9	14.8
evaporation	1.4	4.3
Subtotal	1,553.4	1,565.2
Brine water In		
pit slope drains/bores	133.0	194.8
Salar borefield	5,312.1	6,157.0
ore moisture content	232.0	232.0
TSF return	0.0	0.0
Subtotal	5,677.1	6,583.8
To Operations		
process plant	4,772.0	4,772.0
mine operations	901.8	1,803.5
road maint. etc	3.3	8.3
Subtotal	5,677.1	6,583.8
Water consumption		
process plant		
in tailings	2,390.5	2,390.5
evaporation at TSF	2,381.5	2,381.5
mine operations		
to the ground	100.0	200.0
evaporation	805.1	1,611.8
Subtotal	5,677.1	6,583.8
Water to TSF		
Brine consumption in plant	4,770.0	4,770.0
Brine to TSF	-4,770.0	-4,770.0
Fresh consumption in plant	1,534.0	1,534.0
Fresh to TSF	-1,534.0	-1,534.0
Balance	0.0	0.0
TSF water balance		
Total to TSF	6,304.0	6,304.0
Total evaporation	-3,146.0	-3,146.0
Total in settled solids	-3,158.0	-3,158.0
Balance	0.0	0.0

25.3 Water sources from fresh water basins

25.3.1 Valle de Arizaro

The Valle de Arizaro (VdA) gravel basin is 18 km from the Project site and is the nearest potential source of large volumes of fresh water (Figure 25-2). The catchment has an aerial extent of 345 km² and contains gravels covering a surface area of 96 km². It is the largest gravel basin by area within 50 km of the Project. Based on drilling and electromagnetic (EM) geophysical modelling completed to date, the volume of saturated gravels is thought to be approximately 6.8 billion m³. This translates to 0.95 billion m³ of extractable water, assuming a 20% specific yield from the gravels.

Preliminary estimates for this catchment suggest that recharge is between 110 – 125 L/s (Montgomery & Associates, 2018). Electrical conductivity measurements in water samples from recent drilling showed results of around 1,000 µS/cm which would be considered as suitable fresh water quality for processing purposes.

Additional field data collection is planned in order to improve the reliability of the yield estimate. This work will focus on measuring evaporation rates over the various land surfaces, such as open water, vega vegetation, wet soil, etc. A weather station has also been installed in the range front of this catchment to collect more reliable rain fall and snow fall data. This data will help improve the snow water equivalent (SWE) data inputs used in the Company's calculations.

25.3.2 Valle de las Burras

The Valle de las Burras (VdB) gravel basin is 50 km north-east of the Project site (Figure 25-3). It consists of a 373 km² basin, with gravels covering an aerial extent of 52 km². It has an estimated saturated gravel volume of approximately 5.9 billion m³, which translates to 0.82 billion m³ of extractable water, assuming 20% specific yield from the gravels. The gravels are therefore around 15% volumetrically smaller than those located in Valle de Arizaro. Preliminary estimates for this catchment suggest that recharge is between 135 and 195 L/s (Montgomery & Associates, 2018).

Borehole TW- 10 at VdB was pump tested in 2018 and confirmed good aquifer transmissivity, with pumping rates of up to 50 L/sec possible. Water quality analysis from samples collected at TW-10 indicated that it was suitable as brackish water for processing. Electrical conductivity measurements from hole T-28 samples showed results of around 2,000 to 3,500 µS/cm which would be considered as fresh water for processing purposes. A similarly suitable result of 1,570 to 1,830 µS/cm was obtained for samples from hole T-27.

Additional field data collection is planned in order to improve the reliability of the yield estimate. As for Valle de Arizaro, this work will focus on measuring evaporation rates over the various land surfaces, such as open water, vega vegetation, wet soil, etc.

25.3.3 Valle de Chaschas

Desktop studies of previous pump testing work and geophysical prospecting has shown that the Valle de Chaschas basin (Figure 25-4) has the potential to provide a sizeable fresh water source for the Project.

Valle de Chaschas (VdC) is located about 60 km southeast of the Project site and is adjacent to the Lindero gold project (Fortuna Silver Mines Inc. (Fortuna)). The Lindero project has permitted water rights for two holes, providing a total flow of up to 47.2 L/s. The Company, however, estimates that there is significant gravel storage and recharge in the basin. The broader water resource in the basin is the subject of proposed further pump testing and investigation.

Figure 25-2 Valle de Arizaro catchment (maroon outline) and gravel basin (rose)



Figure 25-3 Valle de Las Burras catchment (maroon outline) and gravel basin (rose)

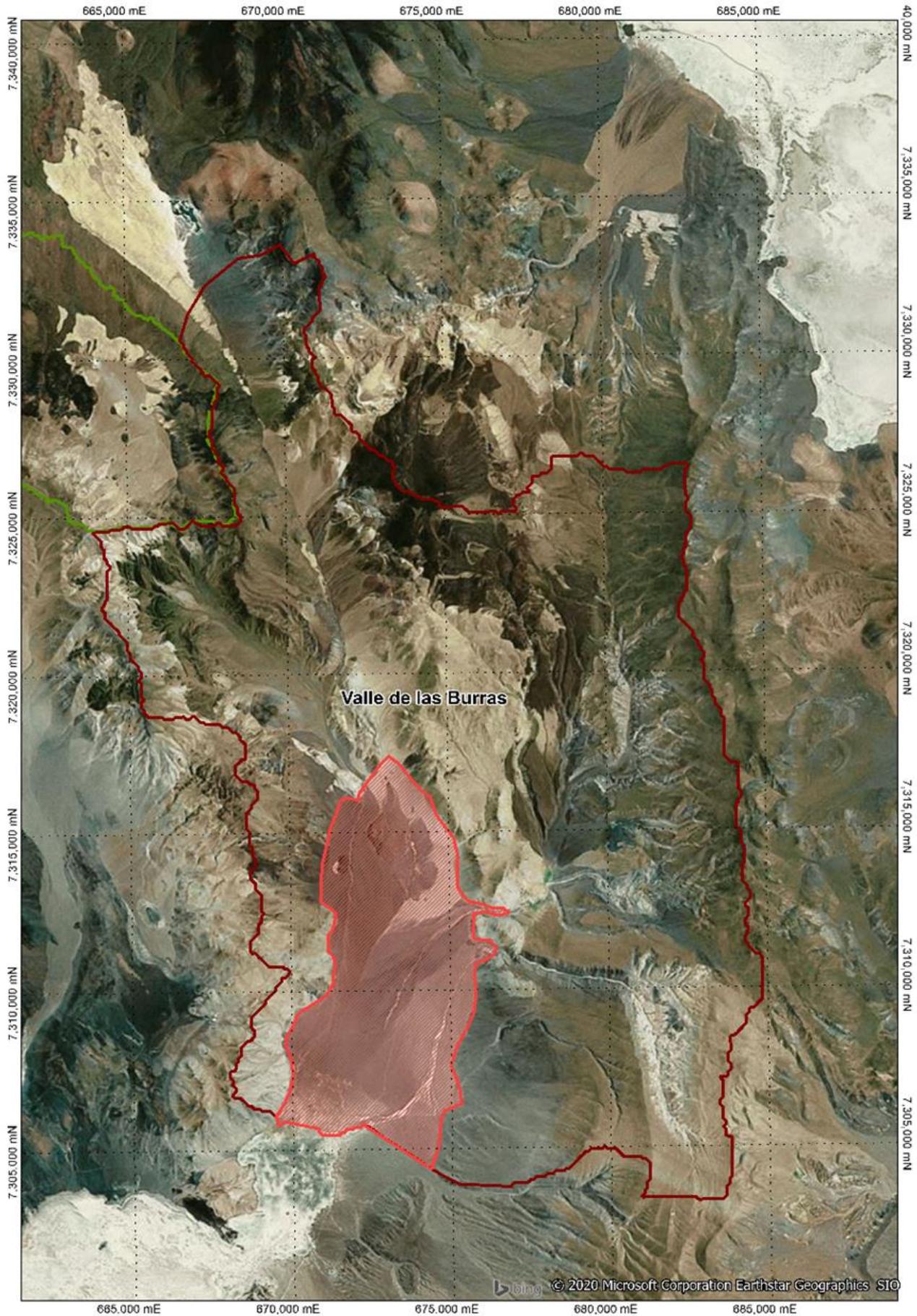
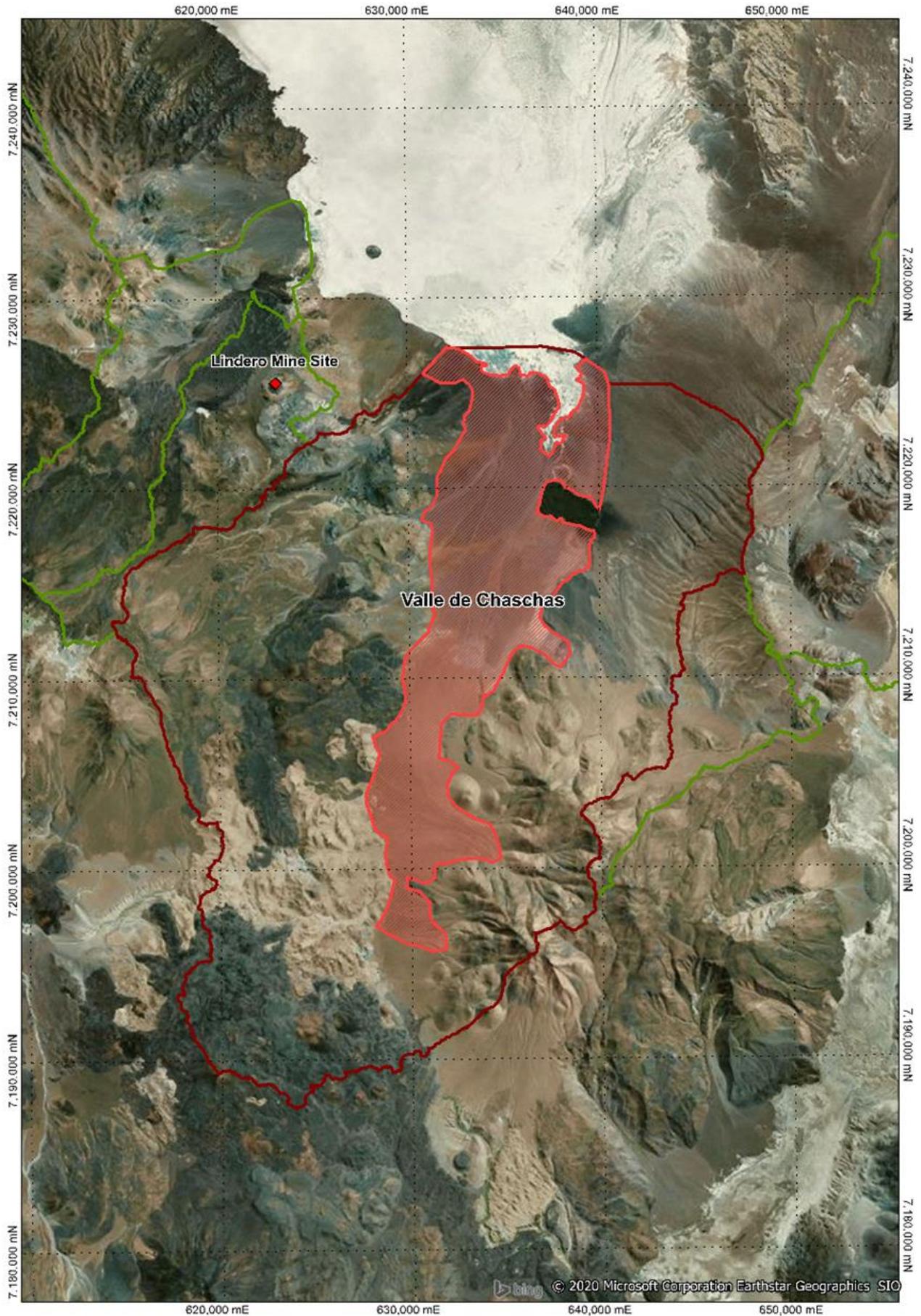


Figure 25-4 Valle de Chaschas catchment (maroon outline) and gravel basin (rose)



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The watershed has an area of 840 km², with gravels covering 168 km². This equates to potentially 7.7 billion m³ of saturated gravels, yielding 1.07 billion m³ of extractable water, assuming 20% specific yield from the gravels. This makes it the largest basin currently under consideration. Preliminary estimates for this catchment suggest that recharge is between 135 L/s and 250 L/s (Montgomery & Associates, 2019). This has a modelled precipitation of 80 mm per annum, whereas the CMORPH data shows an average of 96 mm per annum, so there is potential for even greater recharge.

Chemical testing of the water shows that water in the Valle de Chaschas basin is not suitable for human consumption due to high quantities of boron and iron.

Additional field data collection is planned in order to improve the reliability of the yield estimate. This work will focus on measuring evaporation rates over the various land surfaces, such as open water, vega vegetation, wet soil, etc.

25.3.4 Socompa

The Socompa gravel basin is 50 km from the Project site and is fed by a catchment with an aerial extent of 672 km² (Figure 25-5). It is surrounded by high mountains and with no surficial outflow zones, all run-off water flows into a gravel basin where a lake of surface water of 3 km² to 5 km² in area is maintained throughout the year. The catchment actually contains two gravel basins (Socompa and Socompa North) separated by a bedrock high, but nevertheless hydraulically connected. The combined gravel area is about 22 km², with an estimated saturated gravel volume of 1.7 billion m³, based on geophysical prospecting and drilling data interpretations.

An updated 2018 water balance estimate indicates a recharge of 270 L/s to 280 L/s for the basin (Montgomery & Associates, 2018), with precipitation estimated at 62 mm (1,325 L/s) per annum, based on weather stations in the area, and on altitude modelling. The CMORPH and MODIS satellite precipitation estimates for the Socompa catchment, however, suggest a significantly higher average precipitation of 346 mm per annum, the majority of which would be snow. This suggests that, if the CMORPH and MODIS data is correct, annual recharge rates at this basin could support further increased rates of abstraction.

Previous drilling shows high transmissivity gravels, although these are considered to be of a limited thickness, and translate to 240 million m³ of extractable water, assuming 20% specific yield from the gravels.

Socompa Laguna (lake), near to the basin, is environmentally sensitive for two reasons:

- The lake forms a resting point on Flamingo migration routes, with flamingos present at various times of the year.
- Researchers at Tucuman University [Farias et al. (2013) and Toneatti et al (2017)] have shown that the lake is known to host stromatolites.

The impact of drawdown, or a change in water chemistry, is unknown, however maintaining water levels in the lake should be considered as a priority in order to preserve this occurrence.

Additional field data collection is also planned for this basin in order to improve the reliability of the yield estimate. As with the other basins, this work will focus on measuring evaporation rates over the various land surfaces, such as open water, vega vegetation, wet soil, etc.

Figure 25-5 Socompa catchments (maroon outline) and gravel basins (rose)



25.3.5 Caipe

The Caipe gravel basin is 22 km from the Project site and along with Valle de Arizaro was thought to be a close potential source of large volumes of fresh water. The catchment contains gravels covering a surface area of 83 km² but it is open to the Salar de Arizaro and with only a small, relatively low elevation range front feeding the catchment.

Whilst geophysical profiling suggested that the gravels at Caipe do not contain much water, one historic bore hole at Caipe did contain fresh water. Confirmatory drilling in 2018, however, has now discounted this basin as a significant supply source of fresh processing water.

25.3.6 Other basins

Albeit further afield, there are several other possible water source basins, as described below:

Antofallita

- This is a basin located about 70 km southeast of Taca Taca.
- It has a potentially large catchment area of approximately 1,092 km².
- Interpretation of the electrical geophysical prospecting indicates that there is unlikely to be water saturated sand/gravel zones in the top 150 m (the extent to which the vertical sounding technique can penetrate) of this valley.
 - A target therefore would be to identify a deeper zone of water saturated sediments sitting above the bedrock.

Chuqulaqui

- The Chuqulaqui basin is directly south west of the Caipe alluvial fan.
- Hole T-34 was drilled to a depth of 50 m to test for fresh water storage in the gravels bordering Caipe and Salar de Arizaro.
- Whilst geological logging showed that the hole intersected coarse sand and gravels interbedded with fine clays, electro-profiling showed an absence of water saturation.
- On this basis, it is concluded that Chuqulaqui would not be a suitable source for large volumes of extractable fresh water.

Pocitos

- This is a very large fan structure, with a high likelihood of large water volumes, located approximately 80 km northwest of Taca Taca.
- A borefield pipeline from Pocitos to Taca Taca would involve about 700 m of pumping lift to cross the Cerro Maçon.

Olacapato

- This is a 573 km² catchment with a large outflow of vegas, and a year-round surface stream, located approximately 110 km northeast of Taca Taca.
- A borefield pipeline from Olacapato to Taca Taca would involve about 700 m of pumping lift to cross the Cerro Maçon.

Cauchari

- This is a 130 km² catchment with outflow Vega zones, located approximately 95 km northeast of Taca Taca.
- Previous pump testing by other operators has suggested good transmissivity and a shallow depth to groundwater.
- A borefield pipeline from Cauchari to Taca Taca would involve about 700 m of pumping lift to cross the Cerro Maçon.

Valle de Cori

- This is a basin located about 50 km southwest of Taca Taca.
- It has a potentially large catchment area of approximately 400 km².
- However, it was thought that there may be the likelihood of adequate gravel storage due to the host rock being composed of fine grained sediments (silt, clay).
- A confirmatory RC exploration hole (T-35) to a depth of 150 m was drilled to test the profile at Valle de Cori:
 - This drilling showed that the basin is not prospective for fresh water storage.
 - However, the clay rich nature of the soil and rock profile showed that the substrate may be suitable to form the internal lining of a dam for capturing rainfall run off.
 - Previous geophysical prospecting has shown that the gravels in this basin appear to be saturated from a depth of about 140 m below surface, and have a thickness of 100 m to 140 m over a strike distance of about 14 km.

Rincon

- This is a very large catchment of 1,134 km², with a 180 km² alluvial fan at its south-eastern extent.
- This area likely receives significant recharge, due to its large area, and the gravels represent a potential large volume of fresh water in storage.
- Located 90 km north-northeast of Taca Taca, any pipeline would require a vertical lift of 700m, to an elevation of approximately 4,450 m.
- The fan is located 45km NE of the proposed borefield in Valle de Burras, and could therefore share the infrastructure required to cross the salar.

25.4 Water sources from the Salar de Arizaro

The process plant will be the largest water consumer for the Project. More than half of the plant requirements could be brine sourced from the Salar de Arizaro and to a lesser extent, from depressurisation of the pit slopes on the eastern side adjacent to the salar. It has been assumed that the volume of saline brine available is unlimited, and that brine at up to 300,000 mg/L TDS could be used for milling and rougher flotation.

25.5 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. Phase III investigations were suspended in early 2020 due to the COVID-19 pandemic, and the objectives of that investigation remain incomplete.

25.5.1 Initial investigations, 2011 to 2012

During 2011 and 2012, Ausenco hydrogeologists conducted an investigation to identify and characterise potential water supply sources for the Project, and to quantify potential pit dewatering requirements (Ausenco, 2012). In the first instance, Ausenco researched and analysed published data on the hydrology and hydrogeology of the region, focussing on sedimentary and volcanic stratigraphy, geophysical exploration, and prospects for surface water and groundwater. From this published information, Ausenco was able to list and rank a number of prospective water abstraction sites, based on criteria which included distance from the Project site, water quantity and quality, and favourable hydrological features identified from satellite imagery.

After identifying these sites, Ausenco went on to supervise the installation of 18 x 2-inch diameter standpipe piezometers, using RC and mud rotary drilling methods. Slug tests were conducted at six piezometers. A number of these sites have since been discounted by the Company and investigations refocused into other areas. Furthermore, the projections that Ausenco completed at the time, in terms of water supply quantity and quality, have been superseded by more recent work by the Company.

In addition to the water supply investigation piezometers, Ausenco were involved with the drilling and installation of two piezometers and two test wells in the vicinity of the proposed open pit. After developing a groundwater contour map of static water levels in the pit area, Ausenco carried out groundwater modelling to simulate pit wall depressurisation by means of vertical wells and horizontal drains.

25.5.2 Phase I investigations, 2015

FQM subsequently completed its own Phase I fresh water supply investigations and determined that the gravel basin at Valle de Arizaro held the best potential for a large aquifer within close proximity to the Project. Three wells were drilled in 2015 at Valle de Arizaro, and all three intersected significant water flows at approximately 30 m depth, thereby confirming that the gravels in this particular basin would be at least partially saturated with fresh water. One of the wells (T-22) was converted to a pump testing bore with 10-inch casing to 70 m depth and 8 inch casing to 114 m depth.

Pump testing of T-22 (supervised by SRK Consulting) and TW-12 suggested a pumping rate of at least 50 L/s per bore could be sustainable from the Valle de Arizaro aquifer¹⁷. The Valle de Arizaro basin was interpreted to be sufficiently large that at least four water bores could be utilised to obtain up to 200 L/s from this aquifer.

Water quality testing showed that water obtained during this pump test would be considered as fresh water for processing purposes.

25.5.3 Phase II investigations, 2018 to 2019

Table 25-3 lists the locations of various test bores and production bores completed during Phase II water investigations in 2018 to 2019.

¹⁷ This was scheduled for further testing as part of the Phase III water investigations, and will now be completed at a later date.

Table 25-3 Phase II drilling programme details, 2018 to 2019

Location	Hole ID	Northing (m)	Easting (m)	RL (m)	Hole depth (m)	Depth to water (m)
Valle de las Burras	T-24	7,305,848	670,873	3,478	100.5	4.1
Valle de las Burras	T-25	7,296,386	666,422	3,475	82.5	0.0
Valle de las Burras	T-26	7,308,191	670,991	3,548	285.0	51.2
Valle de las Burras	T-27	7,308,647	672,086	3,570	152.0	76.8
Valle de las Burras	T-28	7,309,771	673,388	3,626	374.0	104.6
Caipe	T-29	7,266,256	604,860	3,598	150.0	0.0
Valle de Arizaro	T-30	7,287,727	605,221	3,576	330.0	56.4
Valle de Arizaro	T-31	7,288,964	604,052	3,632	300.0	111.0
Valle de Arizaro	T-32	7,284,449	606,546	3,573	291.0	60.0
Valle de Arizaro	T-33	7,283,491	605,521	3,666	267.0	150.0
Chuqulaqui	T-34	7,257,300	599,000	3,542	50.0	0.0
Valle de Cori	T-35	7,240,550	605,600	3,547	150.0	65.0
Valle de las Burras	TW-10	7,308,654	672,060	3,570	206.0	76.9
Salar de Arizaro (brine)	TW-11	7,280,953	630,940	3,474	303.0	3.3

Valle de Arizaro

Further to the initial holes drilled at VdA in 2015, bore holes T-30 to T-33 were drilled during late 2018. The locations of these particular holes were chosen to constrain the depth to bedrock modelling based on geophysical prospecting completed earlier in the year. The drilling showed that the thickness of the saturated gravels was significantly greater than indicated from the geophysical profiles. Compared with the original drilling interpretations, the VdA gravel basin was interpreted to be larger than originally envisaged.

Valle de las Burras

Electrical geophysical prospecting was conducted during March 2018, and the results showed large volumes of gravels saturated with fresh water. The interpretation of this survey also showed that the water table becomes shallower, and slightly more saline, towards the level of the Salar de Arizaro.

Five exploration holes (for a total of 994 m) and one pumping well were drilled at Valle de las Burras, with the intention of testing the results of the electrical geophysical prospecting, defining the extent of saturated gravels, and testing the ability to pump large volumes of fresh water from the basin.

Figure 25-6 shows an oblique view of the bore holes drilled at Valle de las Burras in 2018. Holes TW-10, T-24, T-26, T-27 and T-28 are shown on this figure; hole T-25 which was drilled 10 km south west of T-24 is not shown. TW-10 was drilled for pump testing, whilst the nearby T-27 was drilled as a monitoring hole.

Figure 25-7 and Figure 25-8 show geophysical images for cross sections below the salar surface at Valle de las Burras. In these cross sections the blue shading represents fresh water, the green shading represents brackish water and the red shading is salt water. These cross sections correspond to the one shown in Figure 25-9, which is interpreted from drilling records and packer testing. These three cross sections indicate:

- the significant depth to bedrock beneath the surface
- the depth of fresh water in the vicinity of the T-28 exploratory bore is > 100 m, becoming deeper to the north east

Figure 25-10 shows the casing and well construction details for bore TW-10, whilst Figure 25-11 shows the pump testing results. This information indicates:

- 31.3 L/s constant pumping rate yielded a 21.98 m drop in water level over 24 hours.

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- Transmissivity (capacity of the aquifer to produce water) is about 310 m²/day and suggests that Valle de las Burras is likely to be a very good source of fresh water, subject to further pump testing, numerical modelling and analysis of long-term sustainability.
- The initial electrical conductivity of the water was 1,572 µS/cm, which increased to 1,834 µS/cm during the course of the pump testing, as shown in Figure 25-11. This suggests there may be an interaction between fresh-brackish-brine in the basin, which should be monitored during further pump testing.

Water quality analysis from bore TW-10 samples showed that the water is not fit for human consumption.

Figure 25-6 Water drilling locations at Valle de las Burras

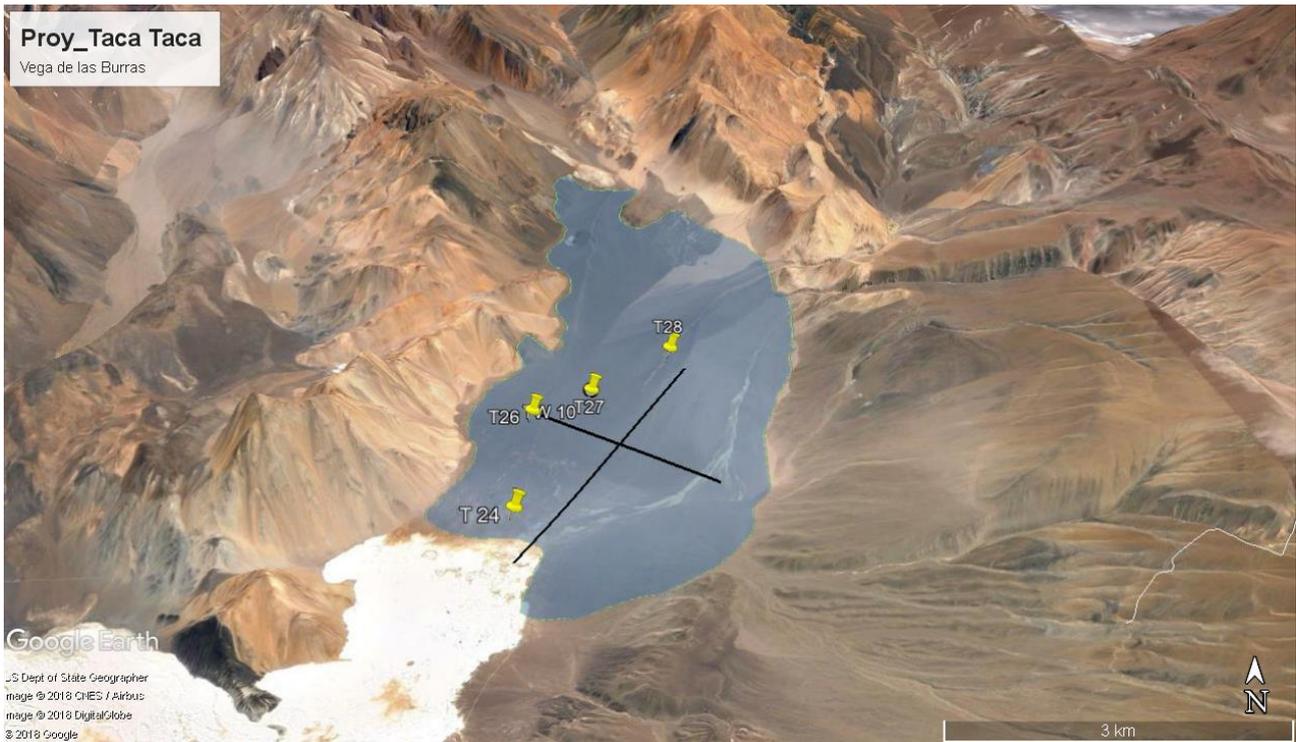


Figure 25-7 Geophysical image NE-SW cross section at Valle de las Burras

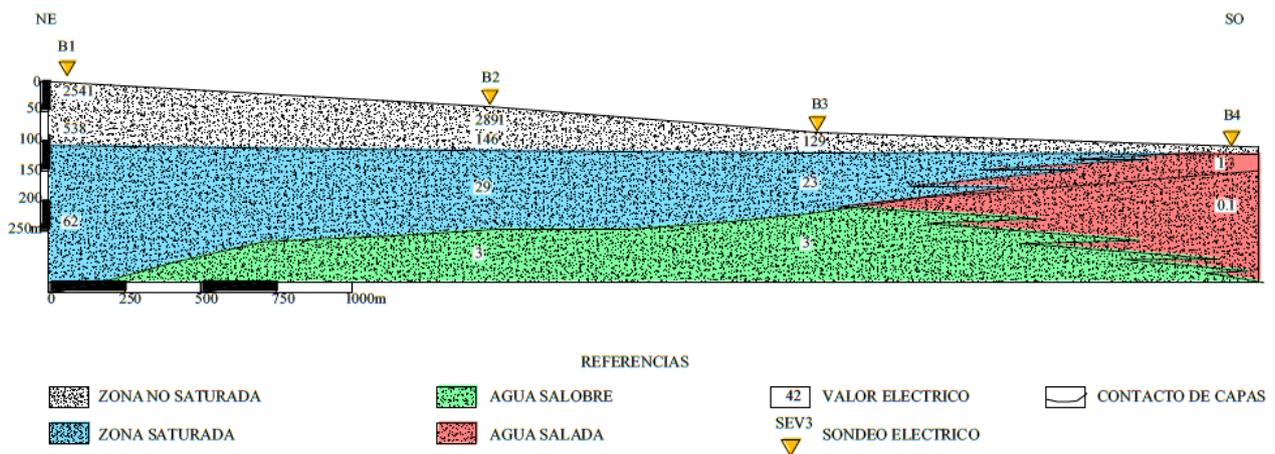


Figure 25-8 Geophysical image NW-SE cross section at Valle de las Burras

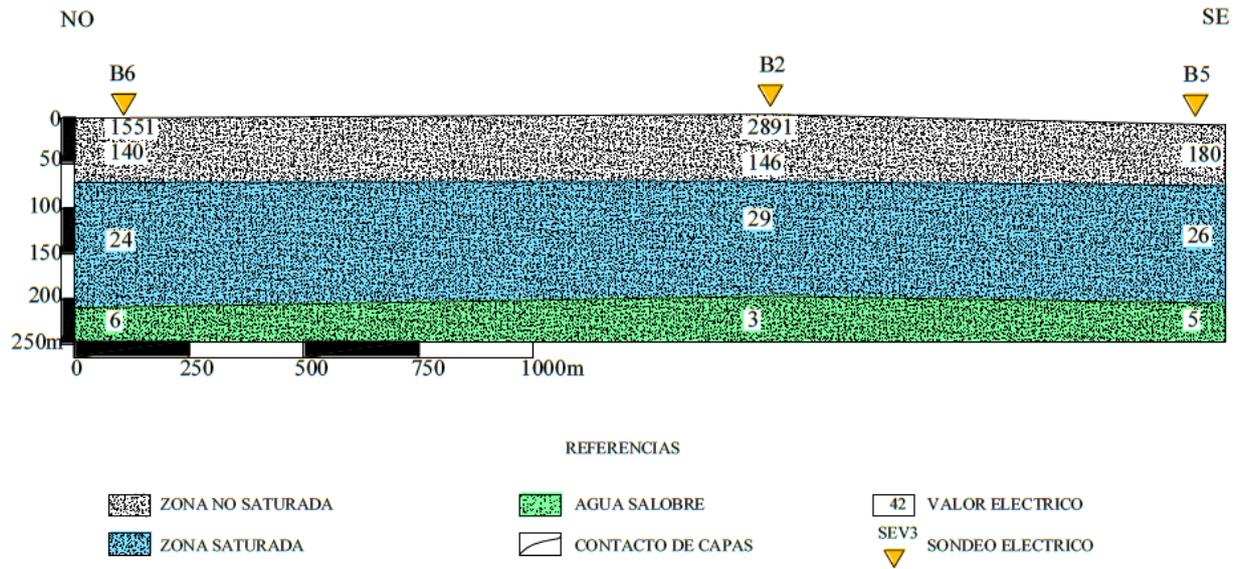


Figure 25-9 Water drilling SW-NE cross section at Valle de las Burras

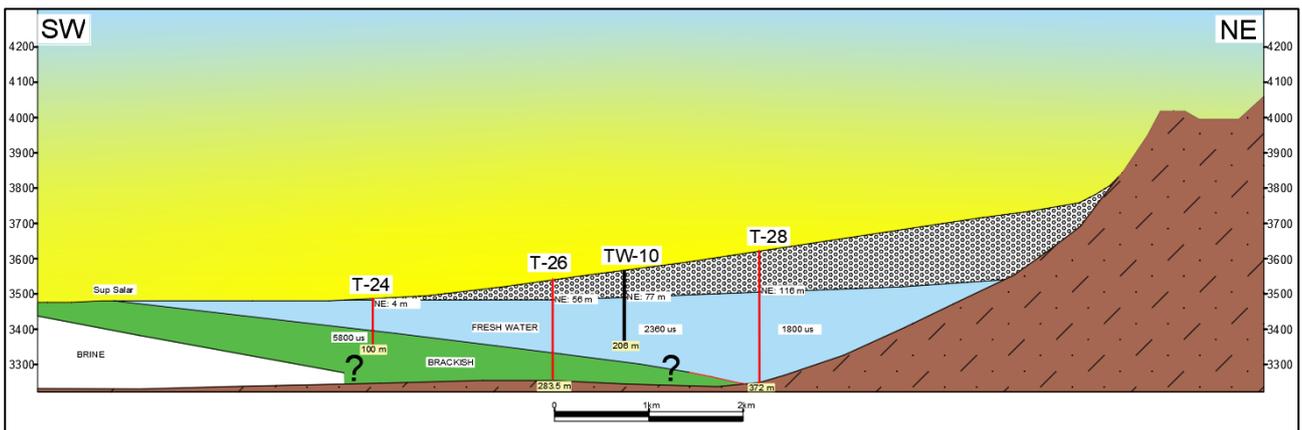
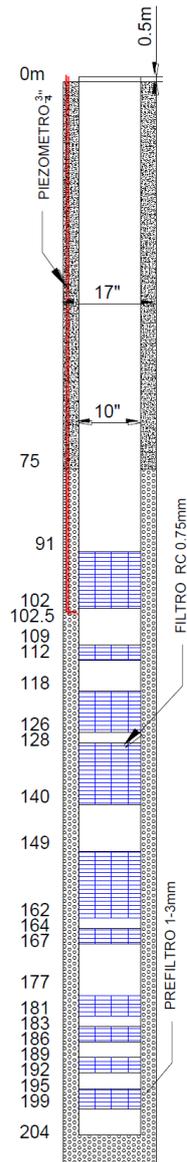


Figure 25-10 Casing and well construction details for TW-10 bore at Valle de las Burras



DISEÑO POZO TW10 LAS BURRAS

140.50m CAÑERIA CIEGA 10"
 64m FILTRO RAN CONT 0.75mm 10"
 15m³ GRAVA SELECCIONADA 1-3mm
 102.5m CAÑERIA PIEZOMETRICA 3/4"
 DISPERSANTE BENTONITICO

TRAMOS	CAÑERIA	LONG
0.5-91	CAÑO 10"	96.5
91 -102	FRC 0.75mm 10"	11
102-109	CAÑO 10"	7
109-112	FRC 0.75mm 10"	3
112-118	CAÑO 10"	6
118-126	FRC 0.75mm 10"	8
126-128	CAÑO 10"	2
128-140	FRC 0.75mm 10"	12
140-149	CAÑO 10"	9
149-162	FRC 0.75mm 10"	13
162-164	CAÑO 10"	2
164-167	FRC 0.75mm 10"	3
167-177	CAÑO 10"	10
177-181	FRC 0.75mm 10"	4
181-183	CAÑO 10"	2
183-186	FRC 0.75mm 10"	3
186-189	CAÑO 10"	3
189-192	FRC 0.75mm 10"	3
192-195	CAÑO 10"	3
195-199	FRC 0.75mm 10"	4
199-202	CAÑO 10"	3
	TOTAL FILTROS	64
	TOTAL DE CAÑOS	140.50

Figure 25-11 Pumping test results for bore TW-10 at Valle de las Burras

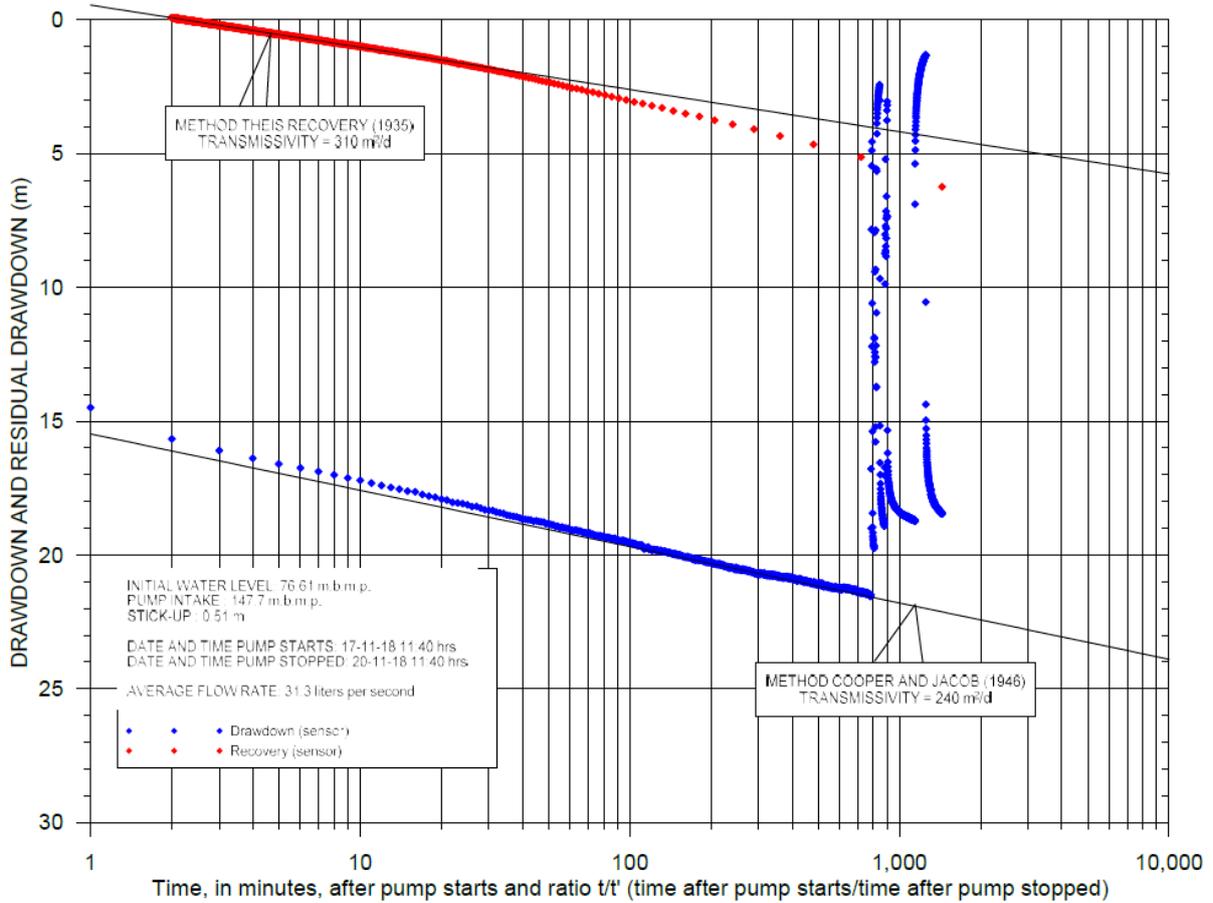
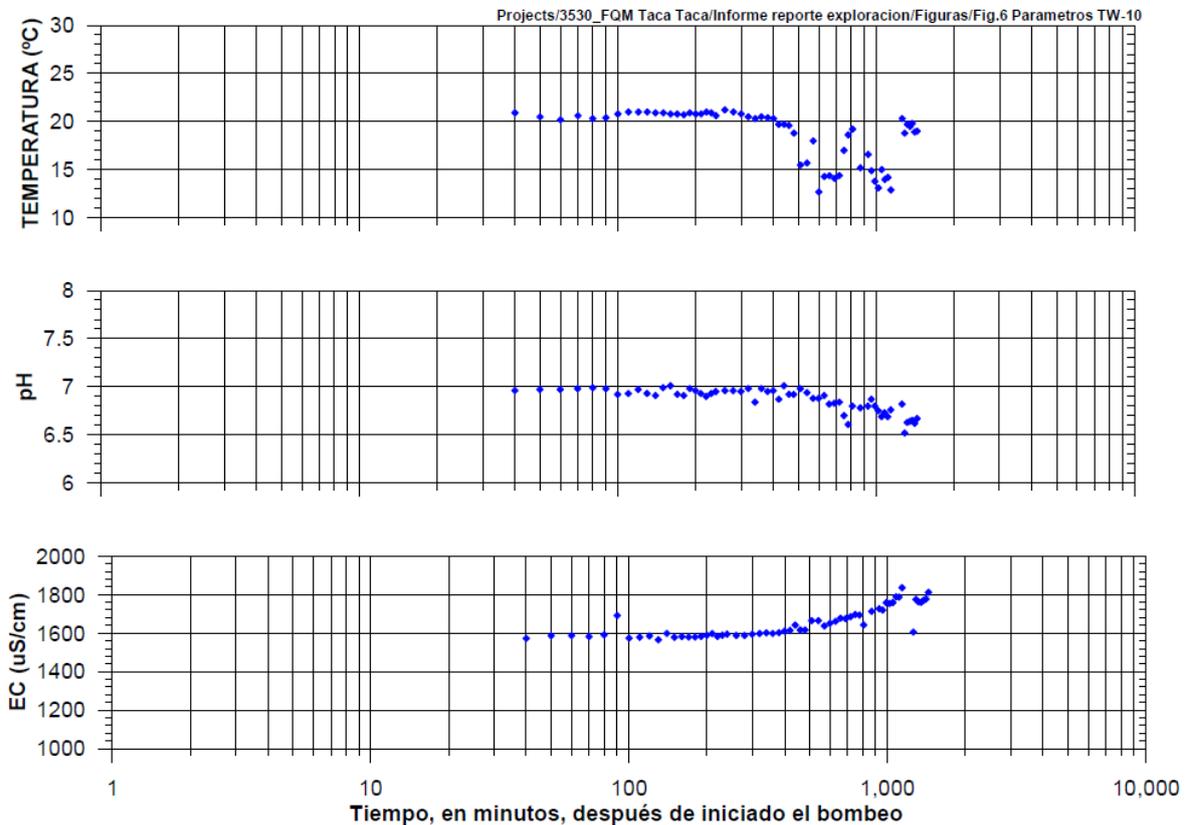


Figure 25-12 Physical parameters of water pumped from TW-10 (Valle de las Burras) over time



Valle de Chaschas

Geophysical prospecting, bore drilling and pump testing had been carried out by Fortuna, and related information was made available to the Company for desktop review. This information formed the basis of the proposed Phase III water exploration programme in that basin.

Socompa

Review of 2017 pump testing at Pozo Socompa showed drawdown of 1.5 m after pumping at 24.4 L/sec for 24 hours, and a high transmissivity of 5,400 m²/day. Testing at nearby Pozo Quebrada del Agua showed a drawdown of 1.2 m after pumping at 22.0 L/sec for 24 hours, and a transmissivity of 2,900 m²/day.

These results were a favourable indication of potentially plentiful fresh water from the Socompa basin.

Caipe

Bore hole T-29 was drilled in 2018 to test the gravel fans bordering the western side of the Salar de Arizaro. The intersected geology and electro-profile of hole T-29, in addition to the topography, indicated that the water level at this location was below the 150 m depth at which the hole was terminated. Water in storage was interpreted to be at or below the level of the adjacent salar, and therefore most likely to be brackish or saline.

On the basis of this single hole it was considered that Caipe would be an unlikely source of suitably large volumes of fresh processing water, and hence further drilling was abandoned in favour of further investigations and prospecting at Valle de Chaschas and Socompa.

Salar de Arizaro

Figure 25-13 and Figure 25-14 show the location of production bore TW-11 drilled on the surface of Salar de Arizaro, about 2 km offshore from the southern end of the Taca Taca deposit. TW-11 was drilled to 300 m depth with the intention of testing the ability of the salar to produce high volumes of brine. The hole was aimed at a paleochannel, previously identified and drilled by Rio Tinto, around 2 km offshore of the Taca Taca deposit. TW-11 encountered water at 4.5 m depth and passed through evaporites to 156 m depth, with clay rich gravels intersected below that.

Two pumping tests were conducted at different depths, and with packers to isolate the aquifers. This was in an attempt to assess the difference in flow between the halite zone, and the more sediment rich zone. The first test conducted with the pump located at 135.9 m below surface showed the following results, which are also shown in Figure 25-15:

- 12.7 L/s constant rate pumping test, had a drawdown of 54.05 m over 24 hours.
- Transmissivity of 30 m²/day was calculated, which is considered to be low.
- This test demonstrated the capability of pumping brine from bores in the salar, although further work is recommended to evaluate options for improving the flow rates from individual bores.

The second test was conducted with the pump at 110.3 m (Figure 25-16), with packers installed at 170 m, isolating the halite horizon, with the following results:

- 10.7 L/s constant rate pumping test for 24 hours resulting in a drawdown of 58.5 m
- A calculated transmissivity of 20 m²/day, which is considered to be low

These results indicate that the halite horizon provides greater transmissivity than the clastic aquifer beneath. The recommendation of Montgomery & Associates is that the eventual production holes should be drilled to a level that allows both the halite, and clastic, units to be tested.

Figure 25-13 Water drilling location on Salar de Arizaro



Figure 25-14 Water drilling W-E cross section on Salar de Arizaro

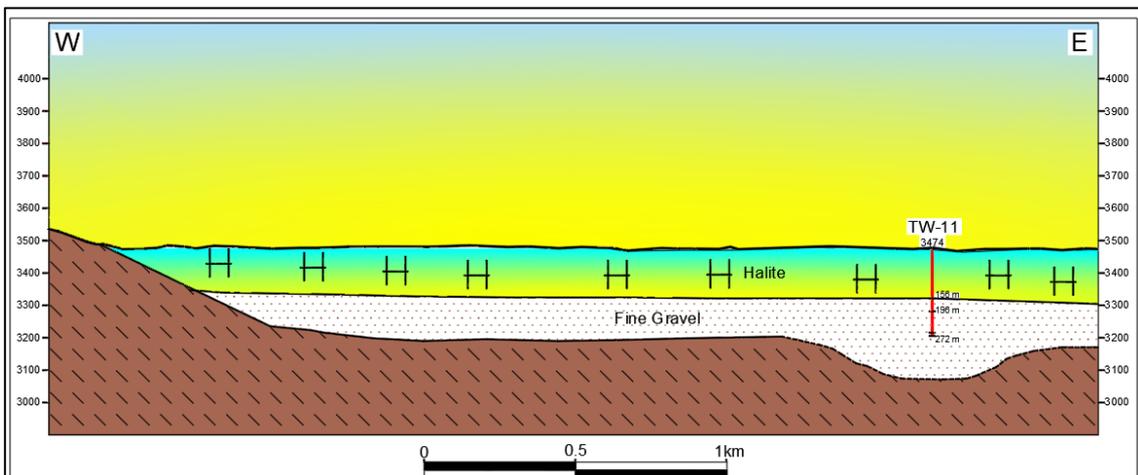


Figure 25-15 Results of first pump test at TW-11 (Salar de Arizaro) – pump at 135.9 m depth

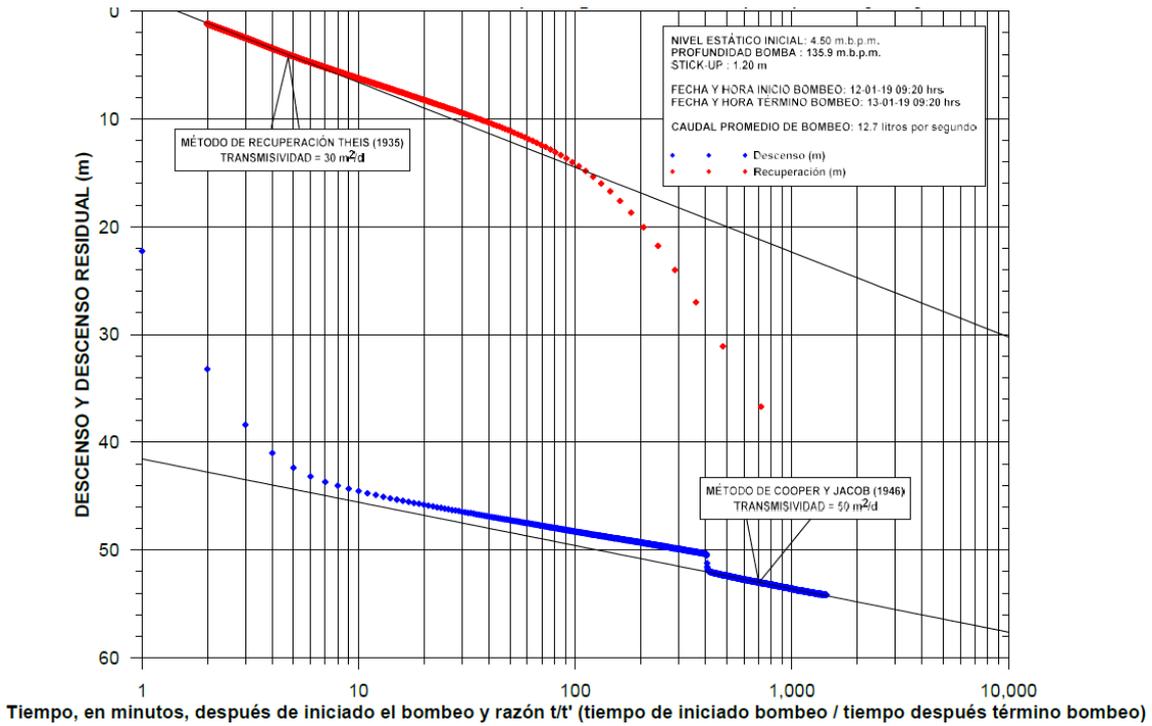
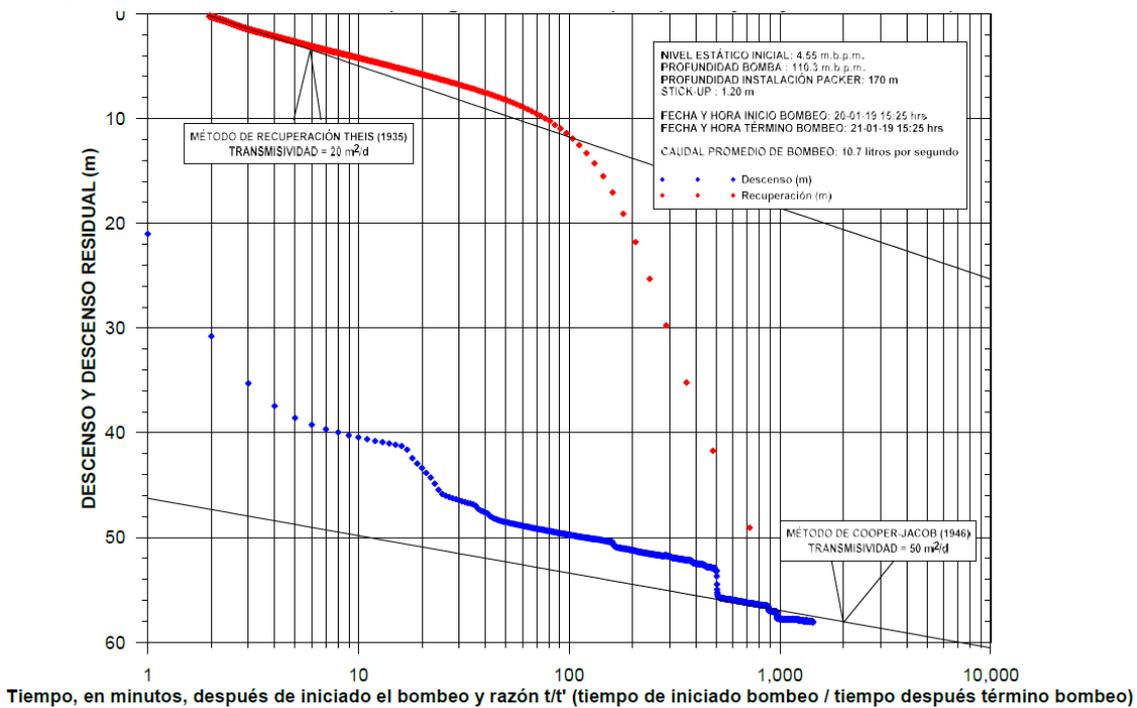


Figure 25-16 Results of second pump test at TW-11 (Salar de Arizaro) – pump at 110.3 m depth



25.5.4 Phase III investigations, 2019 to 2020

The Phase III water exploration programme commenced in September 2019. The aim of the programme was to confirm a water supply plan for the Project through developing bores, pump testing, and numerical modelling of aquifers, in five principal catchment areas: Valle de Arizaro, Valle de las Burras, Valle de Chaschas, Valle de Socompa, and Salar de Arizaro (brine).

The results of this work were intended to confirm these water supply sources for the Project. The programme also aimed to develop the required information to compile water exploitation permits for submission to the water authorities for securing water rights for the Project. The programme was halted in March 2020 due to

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the COVID-19 pandemic, and therefore not all planned objectives were met. The programme will now be completed at a later date (Item 24.6).

Table 25-4 lists the drilling and testing details for bores completed at Valle de Arizaro (Figure 25-17). The proposed drilling and testing at Valle de las Burras was not commenced, nor was it at Valle de Chaschas and Socompa.

Table 25-4 Phase III drilling programme (2019) details - Valle de Arizaro

Drilling type	Hole ID	Sector	East_UTM19s	North_UTM19s	RL (m)	Max_Depth		Depth to water (m)
						Test Bore	Production Bore	
Rotary	TW12	Valle de Arizaro	607108	7283240	3587		152.00	77
Rotary	TW13	Valle de Arizaro	604909	7285126	3623		232.00	110
Diamond	TW14	Valle de Arizaro	604888	2604930	3586	163.00		-

Valle de Arizaro

384 m of rotary drilling was conducted in two holes (TW-12 and TW-13), to a cased diameter of 12" (Figure 25-18). TW-12 was pump tested and TW-13 has been left ready to be pump tested. TW-14 was drilled at 8 1/4" diameter and left ready to be reamed out to a wider diameter to complete the planned pump testing at a later date.

Piezometers have been installed in the margins of the Valle de Arizaro, and water-level monitoring of these and all previously drilled boreholes in the area will be on-going to aid in development of environmental baselines for the Valle. These levels, in addition to evaporation measurement, result from the weather station installed in the mountain range on the western side of Valle de Arizaro. Improved satellite weather data, will eventually be integrated into the basin yield estimates.

TW-12 was the only hole pump tested and the results are as follows:

- 25.0 L/s constant rate pumping test, resulted in a drawdown of 19.4 m over 72 hours.
- Transmissivity was calculated to be 1050 m²/day, a good result, and an improvement on the pumping test results from T-22 in 2015.

Analysis of the water from TW-12 shows that it would be considered as fresh water for processing purposes, although the quality is below the Argentine drinking water standards on multiple parameters, including total dissolved solids, CaCO₃, chloride, arsenic, and manganese content.

Figure 25-17 Location of Phase III boreholes (red), and geophysical profiles at Valle de Arizaro; position of previous drilling shown in blue

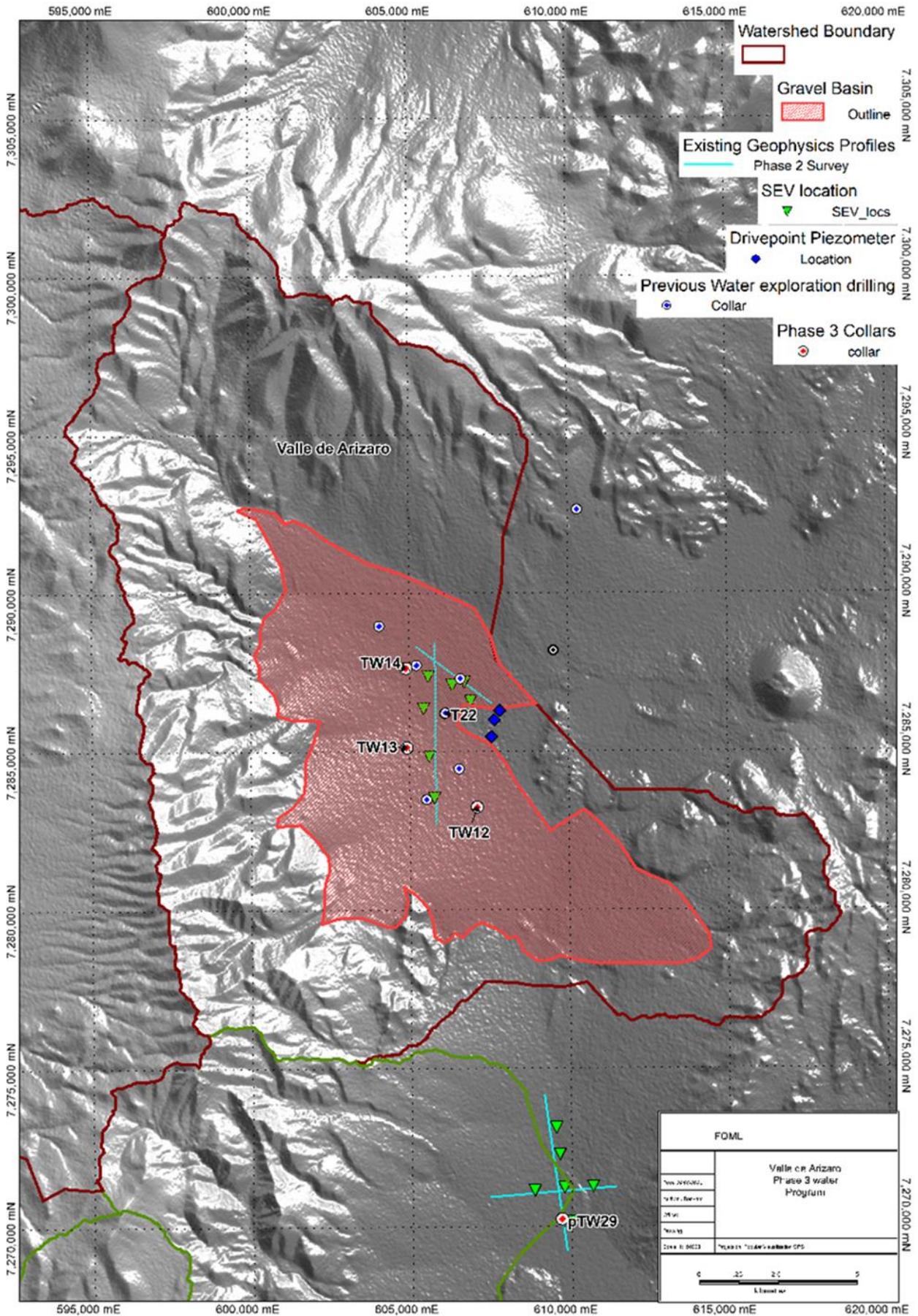
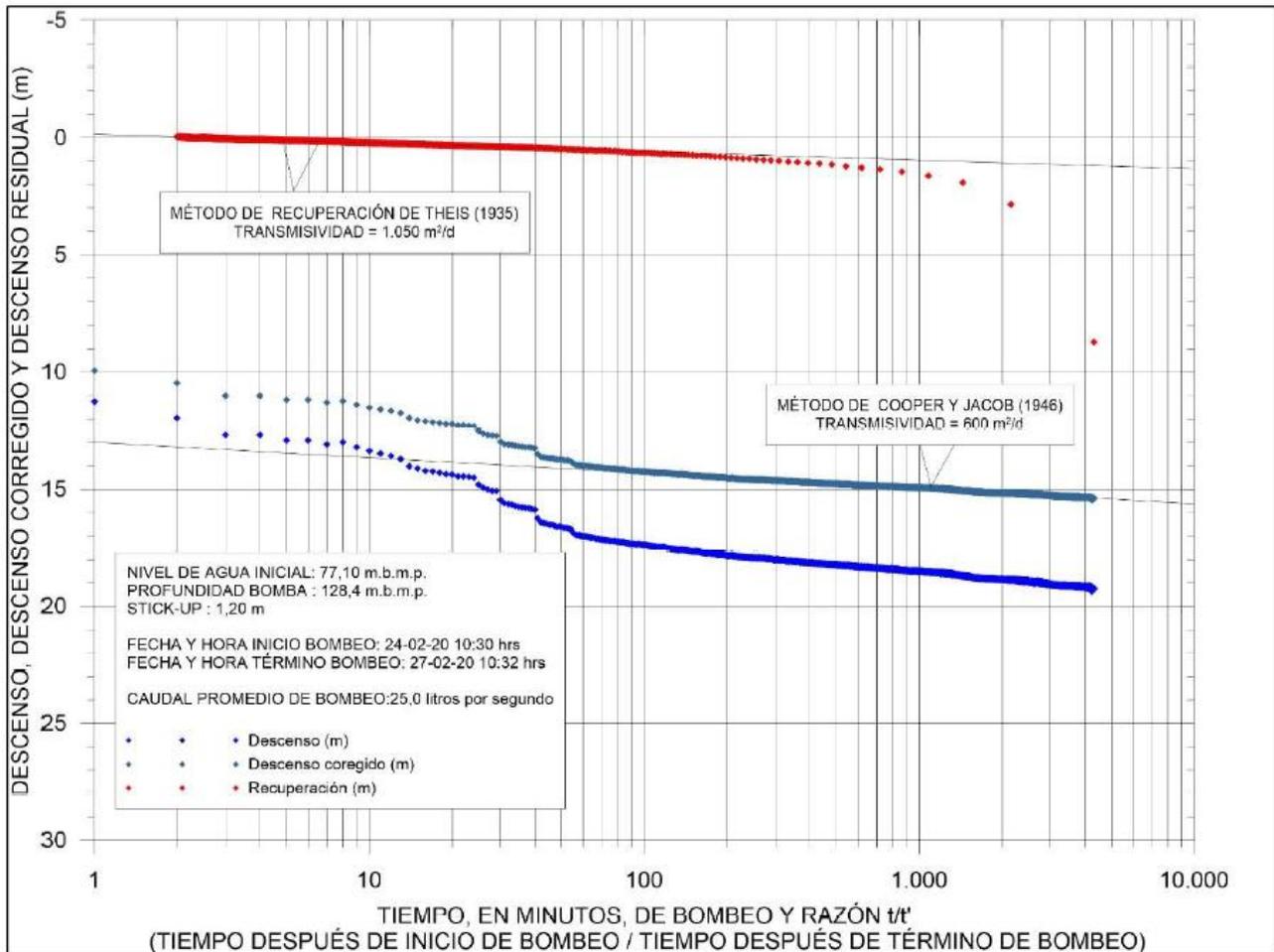


Figure 25-18 Results of pump testing TW-12 at Valle de Arizaro



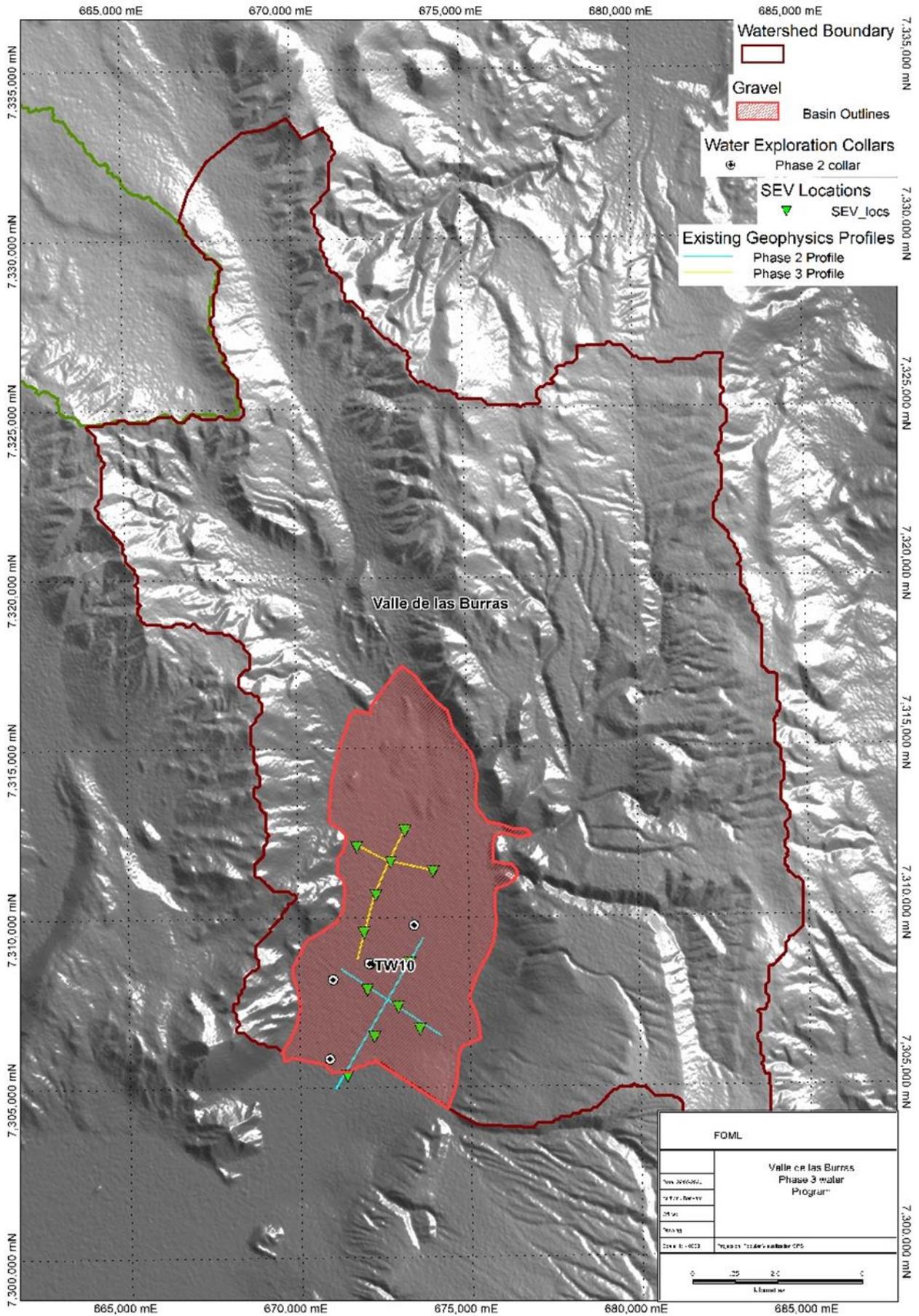
Valle de las Burras

Due to the suspension of the Phase III drilling programme, the planned drilling and pump testing at Valle de las Burras has not yet been completed. However, additional geophysical prospecting was conducted (Figure 25-19).

Initial geophysical prospecting had been conducted in March 2018, comprising six SEV locations (B1-B6). This was followed up in September 2019 with work focused on the northern end of the gravel basin. Six SEVs (SEV1-SEV6) were targeted, with two profiles created with the objective of checking the depth to water, and the extent of saturated gravels in areas with no drillhole coverage. No significant updates to the volume of extractable water have been made using this new data, however have confirmed the suspected presence of transmissive gravels in the NE of the basin.

Piezometers have been installed in the outflow zones at the southern margin of the valley, water-level monitoring of these, and all previously drilled boreholes in the area is ongoing to aid in development of environmental baselines for the Valle. These levels, in addition to evaporation measurements, and improved satellite weather data, will be integrated into the water balance estimates.

Figure 25-19 Location of geophysical profiles at Valle de las Burras; position of previous drilling shown in blue



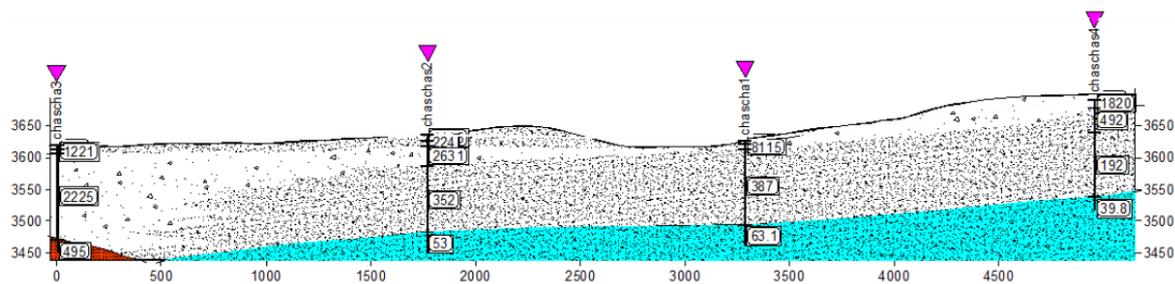
Valle de Chaschas

The planned drilling and pump testing at Valle de Chaschas has also not been completed. Information obtained from Fortuna regarding development and construction of their Lindero project suggests that their overall demand will average 27.2 L/s (98 m³/h). The Lindero project is currently extracting water from the boreholes located in Valle de Chaschas, for which they have a permit for 47.2 L/s.

The geophysical prospecting conducted by Fortuna has now been supplemented by further prospecting conducted by FQM during September 2019. This prospecting comprised two WNW-ESE profiles, for a total of eight SEVs, with the intention of adding a further dimension to the existing profiles and attempting an initial water volume estimate. The profiles show that the basalt plug on the eastern side of the basin curtails the depth extent of the gravels, with the basement recorded at 139 m. The chosen locations for proposed drilling remain valid.

The SEV3-SEV4 profile (Figure 25-20) shows the greatest depth to water, as expected as it is the furthest south. The gradient shows the water is shallower in the east, suggesting there is greater inflow from the high elevation volcanoes (Cerro Tebenquiche at 5,700 m) on that side of the watershed. This also suggests a channelising flow of water, on the eastern side of the watershed, as can be observed in some of the principal drainages features at surface.

Figure 25-20 Profile SEV3-SEV4 looking North, showing a clear gradient from East to west across the gravel storage area



Piezometers have been installed in the outflow zones at the northern outflow area of the valley, water-level monitoring of these, is ongoing to aid in development of environmental baselines for the Valle. These levels, in addition to evaporation measurements, and improved satellite weather data, will be integrated into our water balance estimates.

Socompa

The planned drilling and pump testing at Socompa was not commenced, however geophysical prospecting was conducted across the Laguna Socompa area during September 2019 (Figure 25-21 and Figure 25-22). Four electrical profiles and 15 SEVs were surveyed in order to gain a greater perspective on the volume of saturated gravels around the laguna and attempt to conduct a preliminary water volume estimate for the area.

Overall, the results indicate that the gravels in Socompa may not be very deep, with bedrock typically less than 100 m depth. However previous drilling conducted to 130 m depth did not intersect bedrock, so the current gravel and water estimates may be considered conservative. There are, however, saturated gravels close to the surface, at typically less than 25 m depth. The suspected bedrock high between SEV5 and SEV6 has been confirmed, although it appears that the two gravels basins are hydraulically connected.

Piezometers have been installed in the peripheral zones of the laguna. Water-level monitoring of these, and all previously drilled boreholes in the area, is ongoing to aid in development of environmental baselines for the Valle. These levels, in addition to evaporation measurements, together with improved satellite weather data, will be integrated into the water balance estimates.

Figure 25-21 The position of geophysical prospecting lines in Socompa, with previous drill locations marked

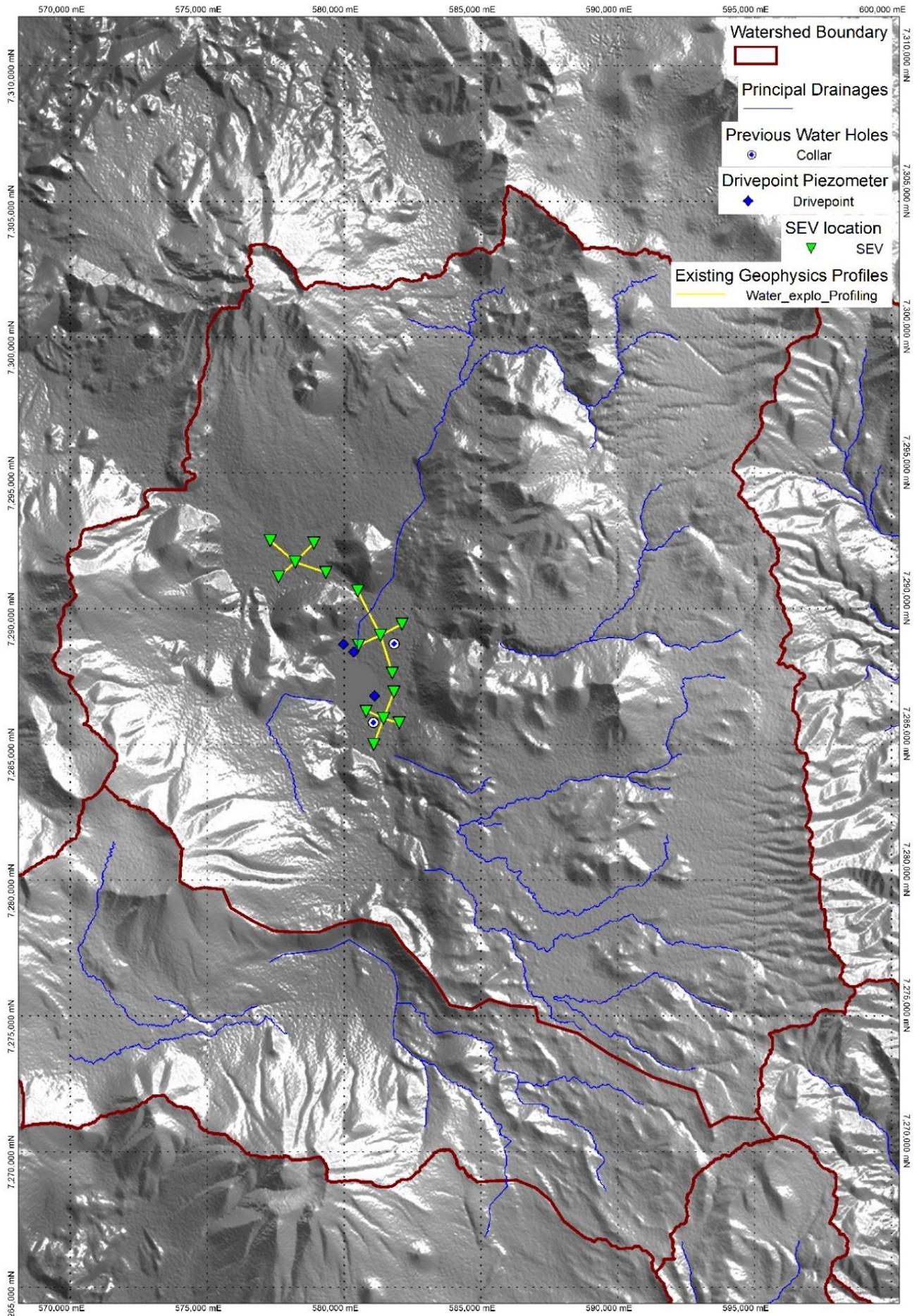
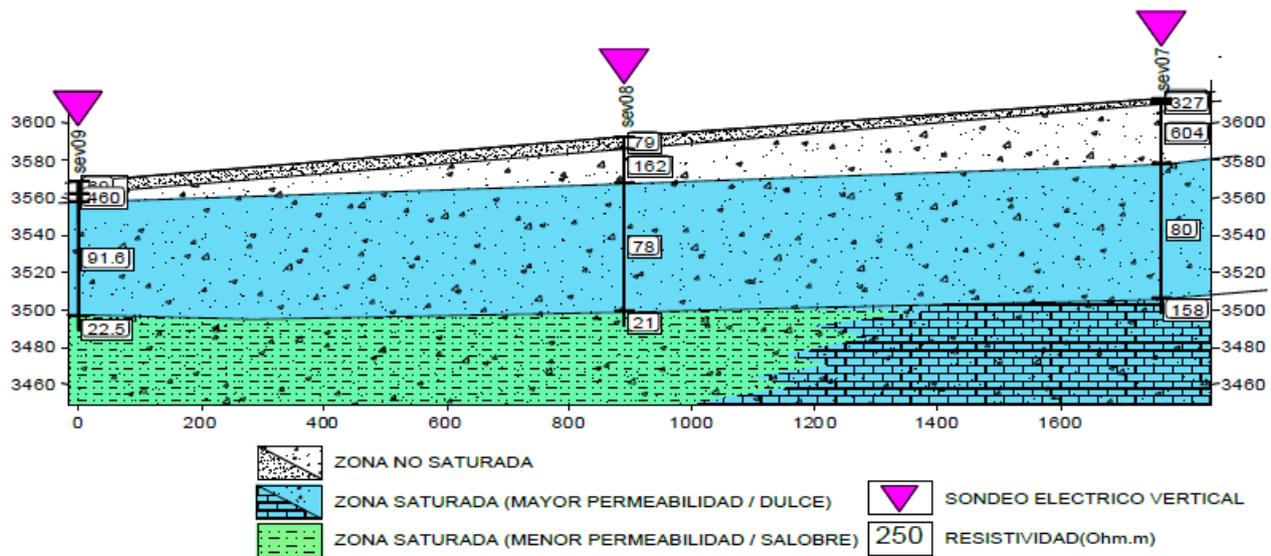


Figure 25-22 Electro profile SEV9-SEV07, looking north. Showing over 80 m of saturated gravel thickness, with brine in permeable bedrock towards the lake edge



25.6 Proposed drilling and testing on resumption of the Phase III investigation

Due to the suspension of the Phase III programme, a proposed resumed drilling and testing programme is planned as a next stage of development and confirming sustainable extraction from the basins of interest. The programme would involve:

- drilling and pump testing of twelve boreholes at various locations, mostly focused in Valle de Arizaro, Valle de las Burras, Valle de Chaschas, and Socompa
- drilling and pump testing of four brine-supply boreholes in the Salar de Arizaro,
- drilling of two brine water exploration wells within the TSF extents on the Salar de Taca Taca
- drilling of two exploration wells at Antofallita
- the continued collection of weather station data, field measurements of evaporation rates, and remote sensing data (CMORPH), to be followed by:
 - updated water balance estimates, and
 - numerical modelling for wellfield planning, long term groundwater modelling and assessment of the impact to environmentally sensitive areas

25.7 Water supply summary

Montgomery & Associates (M&A, December, 2018) completed a review of Project water supply and operational requirements for inclusion with the Project EIA submission. In their review, M&A focussed the fresh water supply source alternatives assessment on Valle de Arizaro, Caipe and Valle de las Burras; Socompa and Valle de Chaschas were not considered at the time.

Fresh water borefields were considered as two within the Vega de Arizaro basin, two within the Vega de las Burras basin and one at Caipe. The borefields were conceptualised as a central tank and pump station associated with five to seven pumping bores at each site. Nominal bore spacing was considered to be 800 m to 1,000 m and preliminary pumping rates of 15 L/s to 25 L/s were nominated.

Consistent with previous concepts, M&A assumed that brine supply would come primarily from a number of bores on the Salar de Arizaro, located in close proximity to the processing plant and serving a dual role of depressurising the open pit eastern wall.

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Based on the Phase III investigations, to the extent completed, M&A subsequently amended and updated their inventory of supply sources and borefield requirements. Table 25-5 lists the updated details of the bores required, along with the production rates.

Drilling and borefield testing will resume as the Project engineering phase proceeds. Regional knowledge of fresh water sources and sustainability confidence will increase, thereby allowing various potential borefield production rate vs engineering/logistical trade off studies to be undertaken.

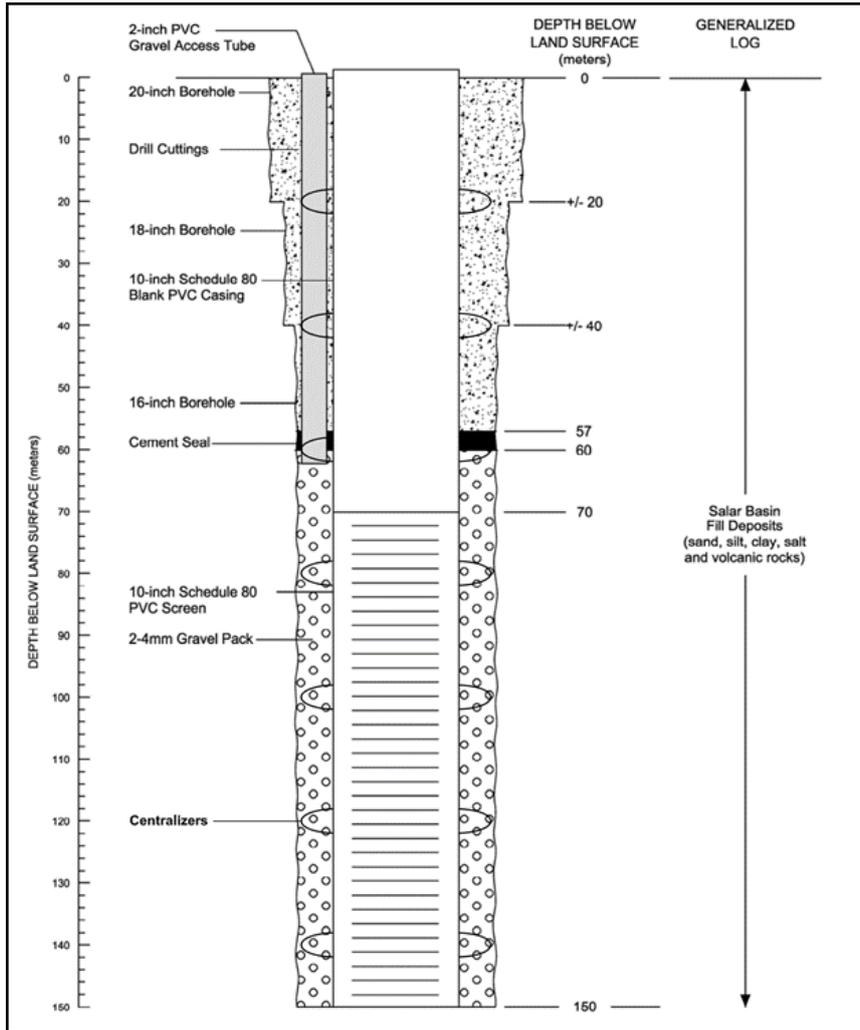
In terms of brine supply, possible layouts for salar wellfields, pit slope depressurisation wells and drains, pipelines, and infrastructure will continue to be evaluated as the engineering phase proceeds.

Table 25-5 Borefield summary

Source/location	Average water supply		Peak water supply		Borefield summary		
	m ³ /h	L/s	m ³ /h	L/s	no. of production bores	av. bore production rate	
						m ³ /h	L/s
Fresh water supply							
Valle de Arizaro	435.5	121.0	792.0	220.0	9	90.0	25.0
Valle de las Burras	525.0	145.8	954.0	265.0	11	90.0	25.0
Valle de Chaschas	495.0	137.5	900.0	250.0	10	90.0	25.0
Socompa	517.0	143.6	954.0	265.0	8	126.0	35.0
Subtotal	1,972.5	547.9	3,600.0	1,000.0	38	99.0	27.5
Brine supply							
pit depressurisation	133.0	36.9	194.8	54.1	4	6.8	1.9
Salar de Arizaro	5,312.1	1,475.6	6,157.0	1,710.3	115	60.8	16.9
Subtotal	5,445.1	1,512.5	6,351.8	1,764.4	119	67.6	18.8

The schematic design (by M&A) of a typical production bore is shown in Figure 25-23. PVC casing is assumed for the brackish and brine wells. In areas of fresher water, the PVC casing may be replaced with mild steel.

Figure 25-23 Schematic design of a typical production bore (M&A, December 2018)



ITEM 26 INTERPRETATION AND CONCLUSIONS

26.1 Mineral Resource modelling and estimation

Taca Taca porphyry copper-gold-molybdenum mineralisation is located below a leached cap horizon and is associated with both supergene and hypogene styles of mineralisation. The porphyry is zoned with a central potassic core which grades into outer phyllic and argillc zones. Phyllic alteration is most common and is closely associated with mineralisation.

Conclusions with regards to this Mineral Resource estimate are as follows:

- Mineralisation is largely disseminated and in fractures and veinlets (stockworks).
- A supergene zone of mixed secondary and primary sulphides is located as a blanket below the leached horizon.
- Primary sulphide mineralisation becomes more dominant with increasing depth.
- The estimate is supported by 435 drillholes having good quality sampling and element analysis.
- Drill grid spacing is around 150 m and wider.
- There was sufficient density data to support block estimates of density values.
- Domains of mineralisation were defined via comprehensive data analysis including neural network analysis of multiple variables for defining key data groupings (domains).
- Geological modelling from drill core data suggests the presence of a strong structural framework.
- Univariate statistics of elements estimated support near normal distributions suitable for ordinary kriging.
- Grade estimate validate well with input data.
- Of the Measured and Indicated Resources, 80% were classed as Indicated.
- The estimate is of sufficient confidence and accuracy to support conversion into Mineral Reserves.

26.1.1 Uncertainty and risk

Domains of mineralisation are typically comprised of mixed primary and secondary mineralisation. This results in the respective domains having statistically mixed populations which limits optimal variography and estimate accuracy.

The wider drill grid spacings pose some risk to the relative position of geological structure as well as the ore and waste contact positions. The presence of structural faults is known to increase the depth of leaching.

Some areas of the deposit edges are only covered by wide drill grid spacings, with opportunity for extension outwards.

26.2 Mine planning and Mineral Reserve estimation

The Mineral Reserve estimate for the Taca Taca Project is the product of a thorough and conventional process reflecting detailed phase and ultimate pit designs constrained by an appropriate optimal pit shell. Volume comparisons between the design ultimate pit and the optimisation shell indicate acceptable minor differences.

The optimisation process incorporates the best available information, including a new and updated Mineral Resource model, improved definition of mineralogical domains and the assignment of copper process recovery to metallurgical domains within the mine planning model. Both planned and unplanned mining dilution have been considered in the modelling and optimisation process.

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The optimisation inputs included mining, processing and G&A unit costs derived from a first principles estimation. In particular, mine operating costs have been comprehensively estimated using defined ore and waste haul routes applicable over the complete range of mining and haulage horizons. This information was able to be carried through into the estimation of mining fleet requirements.

A phased, deep open pit mine design has been developed, and the ultimate pit design has been constrained from transgressing into the adjacent brine saturated sediments of the Salar de Arizaro. This approach is cognisant of the latest mine geotechnical advice provided by consultants Wyllie and Norrish. For each of the pit phase designs, and also for the ultimate pit design, the layouts have incorporated batter, berm and pit slopes which conform to geotechnical advice. The designs include haulage ramps to accommodate truck trolley assist routes.

Working areas for ultra-class mining equipment are considered to be adequate on the basis of multiple mining phases being mined at any one time.

In the opinion of Michael Lawlor (QP), therefore, the Mineral Reserve estimate has considered typical modifying factors and reflects an achievable mining plan and production schedule which is appropriate for the Project at this stage of evaluation.

26.2.1 Uncertainty and risk

Mine geotechnical engineering

Given the significant depth of the proposed open pit (i.e. +700 m), the mine geotechnical engineering work completed to date is somewhat limited by the paucity of structural information obtained from a few inclined geotechnical drill holes, and from rock quality information from a number of vertical drill holes. An aspect of geotechnical risk that has been addressed is the proximity of the pit eastern wall to the Salar de Arizaro. Geotechnical advice has been to remove the risk of potential pit inundation due to inflow from the salar, by altering the design and moving the optimal pit crest off the salar.

The pit slope design parameters assume depressurised slope conditions. The assurance of these conditions is particularly important for the eastern wall adjacent to, but not transgressing onto, the salar.

A programme of further geotechnical drilling and mine hydrogeological investigations is required and the recommendations for such are outlined in Item 26.

Mine operating costs

The estimated unit mining cost estimates have been built-up from first principles and have benefited from the simulation of ore and waste haulage profiles over the life of the mine. This work has enabled the calculation of haulage costs which vary by depth through each of the mining phases, and also the calculation of primary mining equipment requirements over the life of the mine. The estimates have accounted for identified trolley assist haulage routes and hence they reflect potential cost savings due to faster ore and waste hauls. At this time, all mined ore is simulated to be hauled to a surface ROM pad adjacent to the processing plant. This aspect will continue to be evaluated as the Project engineering phase proceeds, with the prospect that there may be a further potential ore haulage cost saving due to shorter hauls to in-pit crushers.

At this stage of Project evaluation, the mine operating cost estimates are considered to be less of an uncertainty than other elements of the Project.

Mining and production scheduling

Conventional open pit mining practices are proposed, making use of ultra-class equipment suited to the scale of production. The nature of the orebody and mineralisation is that a considerable waste rock pre-strip is

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required to expose the initial plant feed. Over 250 Mt of waste is scheduled to be mined over a three-year period leading up to plant commissioning.

To offset this initial hurdle, a sequence of pit phase development and a production schedule have been devised which seek to mine higher grade ores over a twelve-year period from the outset. In addition, the majority of this ore would be hauled and direct fed to the primary crushers, thereby minimising active stockpiling and reclaim.

ARD management

Around 9% of the mined waste will be NAF material, and as such, may be insufficient for encapsulation and/or coverage of PAF material hauled to the waste dumps. Although the dry climate may have a bearing on the potential for ARD run-off, kinetic testwork studies would need to be done to assess the timeframe and rate for generation of acid conditions. If additional NAF material is required, it could be quarried from other sources.

26.3 Metallurgy

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 was 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. This was done for the distinct ore types, and on the ranges of copper head grades, mineralogy and pyrite to be expected over the mine life. These recoveries and concentrate grades were used in the production schedules and cashflow modelling.

Separation of copper and molybdenum concentrates to produce a high grade molybdenum concentrate at acceptable recoveries was demonstrated to be achievable in the early testwork. This work was not included in the most recent testwork programme.

Testwork to provide data for equipment sizing has been sufficient for process designs, with some additional work recommended.

There are several areas of the process design that have not been adequately covered by testwork. Some of these items can be designed with confidence using the experience gained by the company from its operations at Sentinel and Le Cobre Panamá. Some additional testwork and trade off studies have been identified to define other areas of uncertainty.

26.3.1 Uncertainty and risk

Sampling and testing representivity

The original testwork programmes undertaken in 2010 to 2012 included testing of various composite samples of supergene and primary ores, and of both ore types representing the first five production years, and for Years 6 to 10 of the mine plan as developed by Lumina. Variability testwork was also performed on 15 individual supergene and 25 primary ore samples.

The testwork undertaken by the Company in 2019 and 2020 was conducted on ten composite samples obtained from four bore holes drilled within the now proposed starter pit area, and representing the first five years of operations.

It is believed therefore, that the material designated as plant feed for the first 5 to 10 years of operations has been adequately sampled and tested.

Variable mineralisation styles

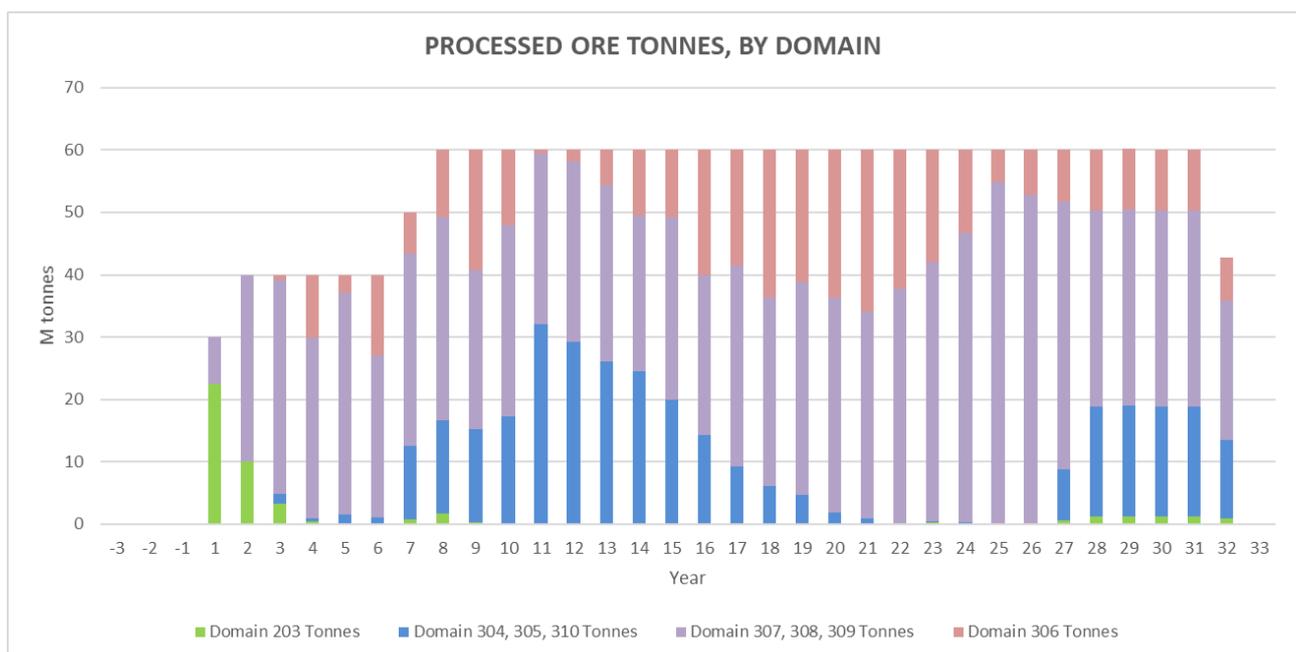
Varying ore types, mineralogy, and copper and pyrite ranges, are expected in the ore feed to the plant over the course of the Project life. Figure 26-1 shows the scheduled annual ore feed types. Testwork results have been used to estimate copper recoveries and concentrate grades for all of these feed grades and mineralisation styles.

High confidence has been assigned to data supported by locked cycle flotation results from the most recent testwork programme using brine, whilst low confidence has been given to material types supported by previous testwork in tap water.

There is low confidence in the recovery assigned to Domain 203, i.e., supergene ore containing acid it comprises only 3% of the ore body.

Similarly, Domain 306 primary biotite-granite ores (approximately 10% of the ore body) have been poorly represented in the testwork campaigns; this ore type is located at depth in the ore body and will comprise a minor portion of the feed in the first five years of operation.

Figure 26-1 Chart of ore feed types to the plant



Sequential copper assays on samples for metallurgical testwork have indicated significant amounts of acid soluble copper in all samples. However, no oxide copper minerals were identified in the QEMSCAN work, and the flotation testwork produced good recoveries and high concentrate grades from both the supergene and the primary ore samples, which would not be expected if significant amounts of oxide minerals were present.

No material can be classified as being 100% primary or secondary mineralisation. Primary ores are defined as those containing more than 50% of the copper present as chalcopyrite. Consequently, when treating primary ores, significant amounts of secondary sulphides will be present in the feed, and there may be some tarnished minerals.

Metallurgical recoveries from oxidised and tarnished minerals may be improved by the addition of NaHS in the flotation circuit (controlled sulphidisation) and this technology will be included in the process design to mitigate any inefficiencies when treating these ores.

Also, throughout the Project life the plant feed will always comprise mixed supergene, supergene and hypogene (primary) ores. The testwork has, however, shown that the optimum processing parameters (grind size, reagent types, etc.), are similar for all ore types.

From the domain modelling process, an average annual concentrate grade of 25.3% Cu is evident in the schedule of concentrate produced, despite the presence of secondary copper mineralisation which typically produces higher concentrate grades (and which were seen in testwork).

Conservative fixed value estimates have been used for the recovery of molybdenum and gold. These numbers may be adjusted after a further testwork programme is complete.

26.4 Processing

As indicated above, the feed to the processing plant during the first five years of operation will contain minimal primary ores, but it will contain significant proportions of mixed supergene material, containing acid soluble minerals.

Testwork has indicated that all of the feed types can be successfully treated in a flotation circuit, and thus the preferred process route follows that of conventional porphyry Cu-Mo concentrators common throughout South America.

There are two significant aspects of the Taca Taca Project which will complicate the flotation processing flowsheet, i.e.:

1. Water quality: the process water most readily available to the site would be highly saline water from the salar. This water could be used for the milling operation and for rougher flotation, but testwork has highlighted the need for better quality water for cleaner flotation in order to achieve the necessary recoveries and concentrate grades. Hence, the flowsheet incorporates:
 - a) a dewatering step for the bulk rougher concentrate prior to cleaner flotation, and
 - b) dilution of the feed to cleaner flotation with good quality water, which must be sourced from a fresh water borefield.
2. Controlled potential sulphidisation (CPS): the plant feed is expected to contain acid soluble copper minerals, secondary sulphide minerals and tarnished surfaces, particularly in the first years of operation. Recovery performance of these minerals could be poor unless the surfaces are sulphidised through the addition of NaHS to a controlled potential. This process is applied successfully at other Company's operations for mixed or transition ores.

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Flotation testwork has been performed in brine and brackish water obtained from site, and the locked cycle testwork replicated as near as possible the proposed process flowsheet.

No work has been performed using NaHS in flotation. Recoveries and concentrate grades have been predicted from the testwork performed (without NaHS), and are thus considered conservative as any improvements in recovery or concentrates through the use of NaHS have not been included in the estimates.

The Company has considerable experience in the design of copper concentrators, and recent designs at Sentinel (Zambia) and at Cobre Panamá form the basis of equipment sizing together with the testwork data.

The comminution circuit will comprise two trains with the largest SAG mills available (40 ft. diameter, 28 MW drives). The ball mills have been sized using the Bond mill work indices from testwork (of which there are 74 data points for samples from 202 to 795 m in depth).

Several design considerations and enhancements will be addressed as the Project engineering phase proceeds. The outstanding items that have been identified at present are listed below:

- review the location of the primary crushers (in conjunction with the mine planning engineers in respect of the potential for in-pit crushing and conveying)
- trade-off study for options on rougher concentrate dewatering (filtration vs. thickening)
- configuration of the cleaner flotation circuit and inclusion of flotation columns and Jameson cells
- review concentrate optimal regrind size and define the regrind circuit power requirements and mill sizing
- review the economics of molybdenum production (production of less than 10,000 tpa of concentrate is indicated) and design the Cu-Mo separation circuit
- review the sizing of the reagent makeup systems in light of the uncertainty with respect to some reagent consumption rates
- define the optimum location for copper concentrate dewatering (main plant or load-out facilities)
- review of the economics of leaching the auriferous material from the near-surface leached cap
- specification of construction requirements to suit the climatic conditions, notably wind loads and catering for the corrosive aspects of brine
- review the options for stage construction of the processing facilities
- specify major equipment that can be deferred, but ensure constructability at a later date in an operating plant
- update the concentrator flowsheet, the layout plan and related facilities

26.4.1 Uncertainty and risk

As noted in Item 25.3.1, there has been testwork performed on material representing the first years of operation, and data is available for equipment sizing.

The company has considerable experience in the design of copper concentrators, and recent designs at Sentinel (Zambia) and at Cobre Panamá form the basis of equipment sizing, together with the testwork data.

Testwork to provide additional data to confirm SAG mill sizing is planned; the samples for this testwork will again be from the proposed starter pit, representing the first years of operations. The recommended testwork programme is discussed in Item 26.

The flotation circuit will be sized using conservative residence times. The CPS flotation involves the provision of conditioning tanks for NaHS and xanthate collector addition after every second cell in the rougher flotation circuit. The process has not been tested in the laboratory (good recoveries and concentrate grades have

been achieved in testwork using conventional flotation), and is seen as an enhancement to the circuit. If found to be of no benefit, it can simply be bypassed.

The uncertainty regarding the process design is in the sizing of the thickeners in the circuit that will operate with brine solutions. Current testwork for thickener sizing was conducted in tap water, and this work must be repeated in brine for several material types.

26.5 Water supply and infrastructure

The adverse climatic conditions in the Project area and specifically the scarcity of fresh water, will remain a significant engineering task to address in the continuing Project engineering phase. Investigations into sources and sustainability of fresh water supplies have been ongoing since 2011. However, due to the COVID-19 pandemic, a Phase III investigation had to be suspended in early 2020. Nevertheless, a summary update on the status of water supply investigations as at the end of October 2020 is as follows:

- Major water resources have been identified at Valle de Arizaro, Valle de las Burras, Valle de Chaschas, and Socompa, 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 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.
- Remote sensing weather data, monitoring, and environmental baselining works are continuing, thereby allowing for increased confidence in water balance predictions for the Project.
- Brine extraction from the Salar de Arizaro is still being investigated, although indications to date are that a significant number of bores will need to be located in the adjacent salar in order to supply the quantity of brine required for the Project.
- Estimates have been made on the capital costs for developing the regional borefields, for equipping bores and for constructing pipelines from these distant sites.

When the Phase III investigation resumes, and as the Project engineering phase proceeds, regional knowledge of these and other fresh water sources and sustainability will increase, thereby allowing various potential borefield production rate vs engineering/logistical trade off studies to be undertaken. Recommendations around continuing work are provided in Item 26.

A preferred route for the Project power supply transmission line has been identified and work is proceeding on the submission of an ESIA in respect of that route. A specialist consultant has been appointed to assist with ongoing power engineering studies and a request for a proposal on the supply of power to the Project has been sent out to selected utilities.

A preferred road access route into the Project site has been designed as a by-pass avoiding an unsuitable stretch of the existing access road. The design is based on a detailed topographic survey.

26.5.1 Uncertainty and risk

In relation to infrastructure items other than water supply, uncertainty, risk and specific comments are as follows:

Waste dump geotechnical engineering

Apart from some initial bearing capacity testwork, no detailed geotechnical engineering or analysis has been carried out for the proposed waste dump located on the salar. Notwithstanding that investigations and

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further design work will be required as the Project engineering phase proceeds, the location relative to the open pit and the currently designed dump slope angles are considered as not posing an appreciable hazard.

TSF location

The preferred TSF site is located within a natural impoundment provided by the Salar de Taca Taca. In this expansive, almost entirely enclosed location, there is the requirement for a relatively low height and short length embankment at the entrance to the salar. A comprehensive assessment and design of the starter embankment with subsequent upstream raises needs to be completed during the Project engineering phase. Nevertheless, for the purposes of this Technical Report, the conceptual design incorporates general design parameters and seismicity considerations gleaned from other Company projects and operations. Independent consultants have been engaged to review the TSF site selection and proposed design.

ARD generation from tailings

Despite the uncertain results from limited geochemical testwork, it is considered that the potential for ARD generation from tailings leachate will be limited by the prevailing arid climatic conditions and the nature of the tailings deposition layers:

- The deposited tailings will be essentially a neutral or slightly alkaline paste, characterised by ground granitic rock particles with a low sulphur content and trace levels of processing reagents, entrained in brine.
- Under the layered tailings deposition conditions it is considered that oxygen levels will be insufficient for acid generating reactions to thrive:
 - Tailings surface layer: Fresh tailings will be continually deposited onto the surface of this layer, and evaporation and crystallisation of the brine salts will take place.
 - Unsaturated sub-surface layer: In a 10 cm to 50 cm thick horizon below the tailings surface there will be a layer of low porosity homogeneous tailings particles limiting the diffusion of oxygen.
 - Saturated layer: There will be no oxygen present in the brine saturated layer at the base of the tailings.
- Whilst it is believed that the potential leachate from the TSF will be benign and may not permeate from the Salar de Taca Taca basin into the Salar de Arizaro, a curtain of monitoring and interception wells is proposed immediately to the south of the TSF embankment.

Rail access

Several consultants have assessed the condition of and logistics associated with the existing railway line between Salta and Mejillones, and passing immediately to the north of the Project site. Route preferences have been determined for the rail freight of construction materials, concentrate and consumables. Cost estimates have been produced for different options regarding rail upgrade extents and rolling stock requirements.

There is an assumption in the cashflow modelling, that the cost of the rail upgrades would be borne by the Project, although there remain alternatives that the cost would actually be borne by the several rail operators, and that this cost would then be passed on to the Company within the rail freight charges over the operations period. Commercial negotiations on such matters have not yet taken place, and consequently the cost of the upgrades have been carried as a Project development capital expense.

Port facility

Preliminary discussions have been held with port owners and operators at Mejillones Bay and whilst there appears to be an interest in handling concentrate from Taca Taca, commercial negotiations will need to be

had. As with the rail upgrades, there is an assumption that any potential expansion or upgrade of port facilities by these port owners ought not be at the Company's expense. In view of the uncertainty at this stage of Project engineering, an allowance for a port facility has been carried as a Project development capital expense.

26.6 Environmental studies and permitting

Environmental studies have been ongoing for some years, and in respect of the Project site, have culminated in the preparation of a detailed Alternatives Analysis and a separate and equally detailed Project Description. The ESIA documentation for the Project was submitted to the authorities in February 2019.

Related ESIA documents are in preparation for approval of the 345 kV transmission line, and separately for the proposed road access diversion. The Project ESIA will have to be updated every two years, whereas the transmission line and road access ESIAs will have indefinite validity.

26.7 Cost estimation and economic outcomes

The capital cost estimate for the processing plant and related site infrastructure, as comprehensively listed by Lumina, has been reviewed, benchmarked against Company projects and operations, and adjusted accordingly. Contingency factors of up to 20% have then been applied.

Updated estimates were produced for the road access diversion, power transmission line and water supply infrastructure. The *Auraxis* (2018) estimates on rail upgrade and rail load-out costs carry a revised 15% contingency. These preliminary capital cost estimates are considered to be suitable for adoption at this stage of Project engineering and evaluation, and will be improved upon as the engineering phase proceeds.

The order of accuracy of the updated capital cost estimates is considered to be in the order of minus 15% to plus 15%. Contingency provisions on the itemised costs vary from 0% to 20%, and with an overall average of 11%.

In relation to Project operating costs, newly estimated mining costs are considered to be more accurately determined than has been the case previously. The processing costs, G&A costs and metal costs (TCRCs) were determined by benchmarking and derivation from first principles.

Against the background of the interpretations and information presented above, and bearing in mind the level of accuracy on the capital and operating cost estimates, the Project cashflow at this preliminary stage appears to be robust in terms of Project NPV and IRR.

The annual cashflows have been optimised by the manner in which an elevated cut-off strategy has been applied, along with the preferential sequencing of the first three phases of mining. Important to note is the relative sensitivity over other variables, that copper pricing and recovery variations will have to the Project.

ITEM 27 RECOMMENDATIONS

27.1 Mineral Resource

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.

27.1.1 Infill drilling

Although estimates validate well at the deposit scale, infill drilling of the current drill grid spacing is required to 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:

1. Starter Pit

A 75 m by 150 m drill programme across the first two years of in-pit ore mining would enhance confidence in the relative positions, volumes and grades of the high grade supergene mineralisation directly below the leached cap. Results would improve estimate confidence of early ore feed for mine planning and would support future drilling requirements, such as grade control drill grid spacing.

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 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.

27.1.2 Extensional drilling

Mineralised areas external to the current pit design have limited to poor drill support. Several priority areas have been identified for the drilling of potential ore extensions that could lead to modified pit/infrastructure designs and production schedules.

27.1.3 Geological work

It is recommended that the following geological work be undertaken for inclusion in future Mineral Resource estimate updates:

1. Refinement of estimation domains to reduce mixing of mineralisation styles should be explored. Increased domain resolution based on copper grade, pyrite content, and recent metallurgical test results may be possible. All future infill drilling will include the immediate 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. A 3D structural model should to be compiled from the integration of multiple data sets, including a recent high-resolution ground-magnetic geophysical survey. This model will allow for the improved definition of weathering profiles and mineralisation continuity at a mining scale. It will also contribute to geotechnical and hydrogeological modelling.
3. The 3D alteration model should be further developed, focusing on relative pyrite abundance and vein type/intensity alongside the delineation of broad gangue mineralogy. Such information would contribute to optimising metal recovery in the mine plan.

27.1.4 Sterilisation drilling

A programme of sterilisation drilling is recommended, and a total of 36 holes for 1,800 m of drilling is proposed across the plant site.

27.2 Mineral Reserve

27.2.1 Geotechnical drilling and investigations

Additional geotechnical drilling is recommended for both the east and west pit walls. 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 required. 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 all possible fault and fracture orientations.

Figure 26-1 shows the proposed geotechnical drill holes with blue collars; the holes shown with red collars are existing (mostly vertical), whilst the holes shown with black collars are exploration drill holes, core from which has been logged for geotechnical information.

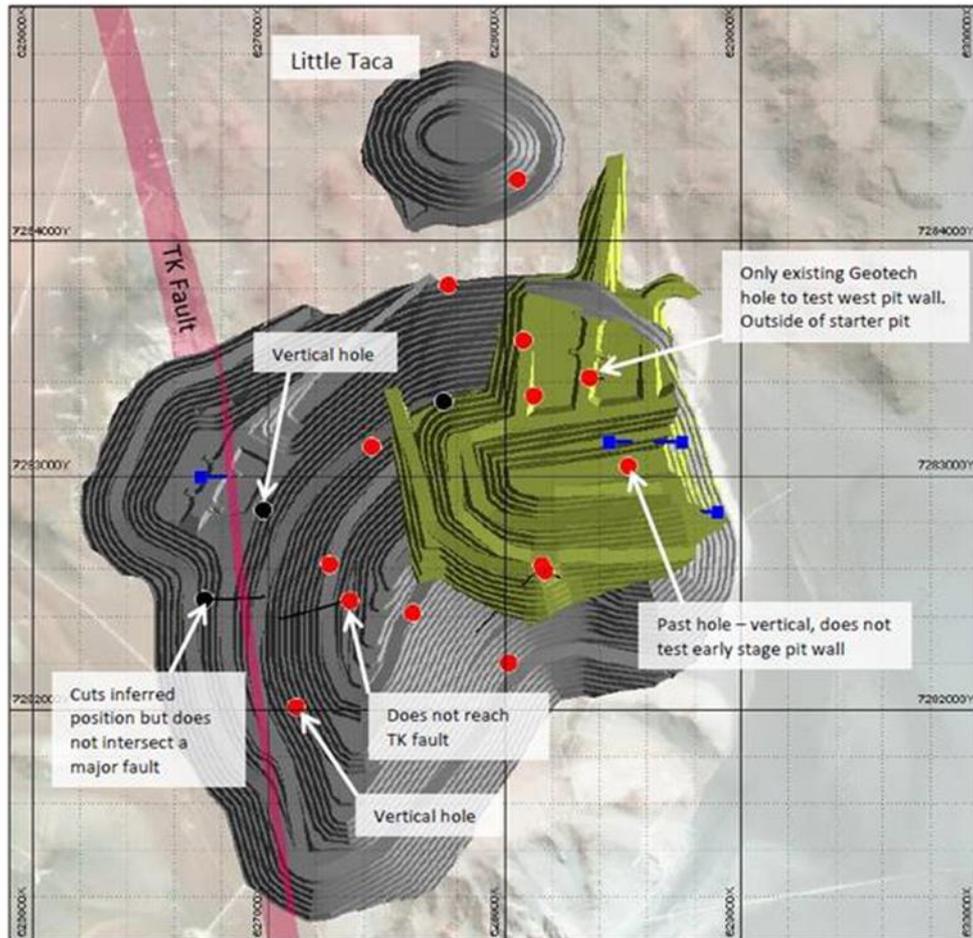
The western pit wall includes a later stage ramp and potentially an in-pit crusher pocket (if IPCC is adopted)¹⁸.

The ramp and potential crusher pocket positions coincide with an interpreted fault and therefore further geotechnical analysis is required. Only one previous geotechnical drill hole intersected this fault. Fortunately, the core from this hole shows no signs of a major broken zone, suggesting either the fault is an early stage, possible feeder structure that has been healed, or it is mislocated, or less significant than currently interpreted. Note that the planned infill resource drilling along this fault zone would help to better locate the fault position prior to drilling of the geotechnical hole.

Given that most of the known structures in the deposit are sub-vertical, the structural information from previous vertical geotechnical drilling is somewhat biased.

¹⁸ The pit design shown in Figure 26-1 is an earlier concept and represents a notional layout incorporating in-pit crusher pockets.

Figure 27-1 Taca Taca pit overlain with proposed geotechnical drill holes (blue collars)



Recommendations from the W&N report (2016) will be addressed in the geotechnical drilling and assessments, as follows:

- Geotechnical drilling should include televiwer 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 televiwer logged to verify the population of steeply-dipping discontinuities and counter-balance the data set developed from the preponderance of vertical holes previously televiwer 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 televiwer 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 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.

27.3 Metallurgy and processing

The majority of metallurgical testwork conducted by Lumina in 2010 to 2012 was done using Lima tap water. A very small number of tests were conducted in saline water and these indicated that rougher flotation efficiencies were similar to those in potable water, however, concentrate grades and copper recoveries to final concentrates were both reduced if cleaner flotation was conducted in saline water.

These results were confirmed in a more recent testwork programme undertaken by the Company in 2019 to 2020, using the water actually available to the Project, i.e. saline water from the Salar de Arizaro, and fresh or brackish water available from Valle de las Burras and from Valle de Arizaro.

This work was performed on ten composite samples from four boreholes that represent the material to be mined from the starter pit and treated in the first five years of operations.

Testwork results were used for equipment sizing and for confirming recoveries and concentrate grades from the different ore types and copper and pyrite grades in the feed. These recoveries and concentrate grades have been used in the mine production schedules to define metal production and in the cashflow models.

Some additional testwork has been identified to answer specific questions related to the process design. This work will be performed in the next phase of project development.

Similarly, some trade-off studies have been suggested for future consideration to optimise the plant design for both the initial 40 Mtpa phase (Years 1-8) and the final throughput of 60 Mtpa.

27.4 Water supply and infrastructure

With the resumption of the Phase III water supply investigations, pump testing for assessment of potential extraction rates, in fresh-water and brine basins, followed by numerical modelling of the principal basins of interest, should take priority. The key deliverables of this exploration work remain:

- Drilling and pump testing of 8" and 10" diameter boreholes at Valle de Arizaro, Valle de las Burras, Valle de Chaschas, Socompa, and Salar de Arizaro, for the purposes of testing potential extraction rates and assessing borefield designs.
- Updated water balances using latest remote sensing, together with field measurement of key input parameters, and thereby improving confidence in the existing models and mitigation plans.
- Long-term numerical modelling of aquifers and simulation of long-term abstraction and drawdown, with quantification of the effects of pumping, in order to better understand the sustainability of fresh-water supply over the life of mine.
- Optimisation of borefield design based on the results of pump testing and aquifer modelling.
- Using the data collected in the Phase III water supply definition programme, compile water exploitation permits and file for permits with the relevant local authorities.

There are a number of infrastructure aspects for the Project that should advance beyond the current stage of engineering. These aspects are as follows:

- There are several elements of the site layout plan that require optimisation review and possible enhancement. The conceptual layout of the process plant is based on the Sentinel configuration and as such, should be designed to suit the prevailing topography at the site selected. Fundamentally, and should it be necessary, the selected location might be discarded in favour of some other convenient site. This review and design should be carried out in conjunction with the civil geotechnical programme.

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- Other conceptual infrastructure elements requiring further engineering include the train load-out facility (especially the means by which concentrate will be delivered from the plant site and transferred into railway cars), fuel delivery/storage, mine services area/layout, warehouses, workshops and administration buildings.
- Additionally, there is the matter of the camp site for which there is only a conceptual location identified at this stage of engineering. The site has been reconnoitred and there is some concern over whether it will be suitably sheltered from the wind. An anemometer has been installed at the site.
- A waste landfill study has been completed and a preferred site selected which is possibly too close to the conceptual camp location. It is recommended that the landfill options be reviewed.
- 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.
- 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.

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ITEM 29 CERTIFICATES

David Gray
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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, Amended and Restated NI 43-101 Technical Report”, dated effective 2021 (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 Member of the Australasian Institute of Mining and Metallurgy and a Fellow Member of the Australian Institute of Geoscientists (AIG).
1. I have worked as a geologist for a total of twenty eight years since my graduation from university. I have gained over fifteen years of experience in production geology and over five years of exploration management of precious, base metal and copper deposits. Over the last ten years I have held senior technical mineral resource positions in copper mining companies operating in Central Africa and worldwide.
2. 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.
3. I most recently personally inspected the Taca Taca property described in the Technical Report in March 2019.
4. 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).
5. I am not independent (as defined by Section 1.5 of NI 43-101) of First Quantum Minerals Ltd.
6. 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.
7. I have read NI 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
8. 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 day of 2021 at West Perth, Western Australia, Australia.



David Gray

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Michael Lawlor
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I, Michael Lawlor, do hereby certify that:

1. I am a Consultant Mining Engineer employed by First Quantum Minerals Ltd.
2. This certificate applies to the technical report entitled "Taca Taca Project, Salta Province Argentina, Amended and Restated NI 43-101 Technical Report", dated effective 2021 (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 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 26.
9. I am not independent (as defined by Section 1.5 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 day of 2021 at West Perth, Western Australia, Australia.



Michael Lawlor

Taca Taca Project | Amended and Restated NI 43-101 Technical Report

Andrew Briggs
First Quantum Minerals Ltd
24 Outram St, 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, Amended and Restated NI 43-101 Technical Report", dated effective 2021 (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.
9. I am not independent (as defined by Section 1.5 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 day of 2021 at West Perth, Western Australia, Australia.



Andrew Briggs