Challenger Gold

ASX Release
8 November 2023

Hualilan Gold Project Scoping Study

Cautionary Statement

The Scoping Study referred to in this announcement has been undertaken to determine the viability of a development of Challenger Gold Limited's (CEL) Hualilan Gold Project, and confirm the business case to progress more definitive studies on the project as the next step towards production. It is a preliminary technical and economic study of the potential viability of the Hualilan Gold Project. It is based on low level technical and economic assessments that are not sufficient to support the estimation of Ore Reserves as per the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC). Further evaluation work and appropriate studies are required before CEL will be in a position to estimate any Ore Reserves or to provide any assurance of an economic development case.

The Scoping Study is based on the material assumptions outlined below. These include assumptions about the availability of funding. While CEL considers all of the material assumptions to be based on reasonable grounds, there is no certainty that they will prove to be correct or that the range of outcomes indicated by the Scoping Study will be achieved.

To achieve the range of outcomes indicated in the Scoping Study, funding in the order of US$150 million will be required. Investors should note that there is no certainty that CEL will be able to raise that quantum of funding when needed. It is also possible that such funding may only be available on terms that may be dilutive to or otherwise affect the value of CEL's existing shares. Furthermore, it is also possible that CEL could pursue other 'value realisation' strategies such as a sale, partial sale, or joint venture of the project. If it does, this could materially reduce CEL's proportionate ownership of the project.

Given the uncertainties involved, investors should not make any investment decisions based solely on the results of the Scoping Study.

The Scoping is presented in USD unless otherwise stated and to an accuracy of ±15% where costs have been sourced from vendor quotes or first principles analysis and the costs developed by benchmarking have a target accuracy of ±35%.

[3] There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised. CEL is satisfied that the proportion of Inferred Mineral Resources is not the determining factor in project viability.

[4] The viability of the development scenario demonstrated in the Scoping Study does not depend on the inclusion of the Inferred Mineral Resources. Removing the Inferred Mineral Resources from the mine plan still produces a positive NPV and attractive IRR but reduces the mine life to 5.8 years.

The Scoping Study contains forward looking statements and the Company has determined that it has a reasonable basis for doing so and believes there is a reasonable basis to fund the Hualilan Gold Project.

Hualilan Gold Project Scoping Study Delivers In Spades

Potential lowest quartile costs, rapid payback, strong cashflows

Highlights

- **Scoping Study (SS) focussed on a high-grade starter mine at Hualilan finds:**
  - Forecast EBITDA of US$738m (**A$1.1 billion**) over Life of Mine (LOM);
  - Rapid payback period of under 1.25 years based on current production target; and,
  - Forecast for Challenger Gold (CEL) to be one of the lowest cost Top 20 ASX-listed producers.
- **Key operational findings of the Scoping Study are for Hualilan to support:**
  - Average annual production target of 116,000 oz Au, 440,000 oz Ag, 9,175 t Zn (141koz AuEq[3]);
  - Global lowest-quartile C1[1] cash cost of US$527/oz (A$811) and AISC[2] of US$830/oz (A$1277);
  - An initial mine life of 7 years, with mineralisation open at depth potentially extending LOM;
  - Low-risk starter pit followed by conventional sub-level open stope (SLOS) underground mining;
  - Processing facility includes a crusher, mill, gravity recovery circuit, conventional sulphide floatation, and floatation-tails leaching (FTL), with schedule LOM throughput of 1.05 Mtpa;
  - Production schedule is comprised of 81% Indicated Resource and 19% Inferred Resource [3 4]
- **Compelling financial metrics of the Scoping Study include:**
  - Pre-tax NPV[5] US$409m (**A$629m**) at US$1,750/oz Au $20/oz silver (spot gold price US$1975);
  - Pre-tax NPV[5] increases to **A$820m** at current gold (US$1,975) and silver (US$23) prices;
  - Project IRR (Pre-Tax Real) of 75% and a breakeven gold price of US$983/oz.
- **Outstanding potential upside:**
  - Indicative NPV ignores residual value of the 1.7Moz AuEq[3] remaining after the SS LOM due to the high-grade/ low-tonnage focus and is not considered to reflect the full value of the asset.
  - Initial column testing indicates it may be economically viable to run a heap leach process option which would assist with recovering some of this mineralisation that has been excluded.
  - The underground optimisation at US$1700/oz Au excluded 880 kt of stopes grading 2.7 g/t AuEq[3] from the mine plan, which are likely economic to mine at today's gold price;
  - Mineralisation is open at depth, in both directions along strike, and there are numerous regional exploration opportunities on adjacent trends within the 600 Ha footprint; and
  - Current gold and silver pricing 14% higher than priced used in the Study Assumptions.
- **Company seeks to fast-track production by expediting metallurgical testwork while evaluating options to recover the lower grade mineralisation via a larger open pit.**

[1] C1 Cash Cost – operating costs include all mining and processing costs, site administration costs, transport, treatment/refining costs, by-product credits.
[2] ASIC – All in sustaining cost is the total cost of sustaining current mining operations and current production rates.
[3] AuEq information as required under JORC is provided on Page 23 of this ASX release as footnote to Table 13 (Hualilan MRE)

Table 1 - Key Scoping Study Assumption

Price Assumption	Study Assumption	5 Year Avgerage	Spot
Gold	**US$1750/ oz**	$1710	$1975
Silver	**US$20/ oz**	$20.72	$23
Zinc	**US$1.15/ lb**	$1.28	$1.15
Lead	**US$0.94/ lb**	$0.93	0.98
AUD/USD	**0.65**	0.70	0.65

Metallurgical Recoveries and Concentrate Payability	Recovery (%)	Avg Payability[1] (%)
Gold	95.8%	88.4%
Silver	93.0%	54.8%
Zinc	89.0%	73.1%
Lead	75.8%	93.6%
Concentrate Transport (site to smelter including insurance)		US$150/ wmt
Mining Physicals	**Open Pit**	**Underground**
Tonnes Ore	1.3 Mt	5.8 Mt
Tonnes Waste	8.4 Mt	2.1 Mt
Underground development		58,937m
Indicated and Inferred Resource (% indicated/% inferred)	82%/ 18%	81%/ 19%
Gold Grade (LOM average)	3.4 g/ t	3.6 g/ t
Silver Grade (LOM average)	22.3 g/ t	12.1 g/ t
Zinc Grade (LOM average Type C material)	3.9%	2.7%
Lead Grade (LOM average Type C material)	0.33%	0.14%
Unit operating Costs	**Unit**	**Unit Cost**
Open pit Mining (ore/waste)	**US$/ t mined**	**3.00**
Underground Mining	**US$/ t mined**	**34.74**
Underground Development		
Inclined Development (5 m x 5 m)	US$/ m	2,828
Horizontal development (5 m x 5 m)	US$/ m	2,828
Vertical Development	US$/ m	2,333
Slot Rises (included in underground mining cost)	US$/ m	1,500
Underground Development	**US$/ t mined**	**28.29**
Total Underground Mining and Development	**US$/ t mined**	**63.03**
Processing (Type C ≥1.5% Zn) 34% total PMI	**US$/ t processed**	**16.31**
Processing (Type B ≥1.5 g/ t Au, <1.5% Zn) 60% total PMI	**US$/ t processed**	**12.12**
Processing (Type A <1.5 g/ t Au) 7% total PMI`	**US$/ t processed**	**9.26**
G&A	**US$/ t processed**	**5.38**

[1] **Payability after Transport Cost . Refining Cost (TC/RC) / Penalties**

Challenger Gold (**ASX: CEL**) ("**CEL**" the "**Company**") is pleased to provide the outcomes of the Scoping Study ("the **Study**") completed on its 100% owned Hualilan Gold project located in San Juan, Argentina. The study presents an initial economic evaluation of the project and suggests that the project could become one of the lowest-cost ASX producers, with a rapid payback period, and average annual production of 116,000 oz gold (141,000 oz gold equivalent[3]), based on the Study production target. Notwithstanding the excellent outcome of the Study, the Company has identified several clear, and potentially material, opportunities for optimisation and improvement.

The key Study assumptions are outlined in Table 1, with a summary of the study financial outcomes provided in Table 2. Complete study details are available in the Scoping Study which is included as an Annexure to this ASX release. The Company is currently undertaking an external analysis of the projects carbon intensity, which will be released to the ASX upon completion.

The Study focused on the high-grade core of the Hualilan MRE to present a low startup-capital project capable of being funded by the Company in the current challenging market conditions. The Study presents two cases, both using a 1 Mtpa process plant. Case 1 involves crushing, milling, gravity recovery of gold, conventional floatation, and flotation tailings leach (FTL). Case 2 evaluated gravity recovery, followed by conventional carbon-in-leach (CIL). The floatation case provides a superior economic outcome, however the Pre-tax NPV_5 for both cases are within a margin of 7%, while the CIL case has a US$20 million lower capital cost estimate. Following the receipt of further metallurgical testing, to supplement the preliminary CIL recovery assumptions for Case 2, a decision will be made if both conventional flotation and CIL will be further evaluated in the next stage of studies.

Table 2 - Scoping Study Case 1 (conventional flotation) summary financial outcomes

Revenue **US$1.5B** **A$2.3B**	**EBITDA** **US$738m** **A$1.1B**
C1 Cash Costs **US$527/oz** **A$811/oz**	**Operating Margin** **50%** **EBITDA/Revenue**
All In Sustaining Cash Costs (ASIC) **US$830/0z** **A$1277oz**	**NPV[5] (Pre tax)** **US$409m** **A$630m**
IRR (Pre-tax) **75.2%**	**NPV[5] (Pre tax) - spot prices** **US$533m** **A$820m**
Payback Period **1.25 years** **(post tax)**	**Initial LOM** **7 Years** **(mineralisation open in all directions)**

Study Approach

The Project has inherent optionality given that the Hualilan MRE starts at surface and contains a high grade core of 8.1 Mt at 5.0 g/t Au, 17.4 g/t Ag, 1.8% Zn at the underground optimisation cut-off grade of 2.37 g/t AuEq, within the larger MRE of 60.5 Mt at 1.1 g/t Au, 6.0 g/t Ag, 0.44% Zn.

The Study commenced using benchmarked costs provided by Mining Plus from a sub-set of their internal mining database, specifically originating from operations in Latin America. This allowed for a preliminary options assessment of:

- Open-pit (OP) vs underground (UG) vs combination mining
- Different potential processing flowsheets which considered multiple circuit combinations which included or excluded: gravity concentration, floatation (including FTL) and CIL
- Multiple potential processing throughput rates; and
- Approximate capital cost of various options.

An initial OP optimisation, using the aforementioned Mining Plus benchmarked unit costs from Latin America, indicated that potential exists for a large open pit which would take advantage of economies of scale. Initial pit optimisation using a mining cost of US$2.00/t, pit wall slopes of 55° on the east, and a US$1800 gold price generated a pit, which at Revenue Factor 71, recovered 47.6 Mt of mineralisation and delivered an undiscounted value of US$1.0B, with a further 2 Mt of high-grade mineralisation potentially mineable via underground methods below the optimised shell.

After considering these initial results and the associated capital costs, the Company took the decision to focus the Study on the high-grade core of the mineralisation. This decision was primarily made to ensure that CEL had a credible pathway to fund production via a development plan with a low up-front capital cost and a rapid payback. The present, challenging market conditions did not appear conducive to the Company's ability to fund the higher capital cost (and/ or potentially longer payback period) required for the construction of a high-volume OP mine, despite it being likely that mine development could be staged to manage capital outflows.

A key part of the reasoning behind this decision was that the initial pit optimisation showed material levels of sensitivity to OP mining unit cost and to pit slope angle, particularly on the eastern side of the pit where the Hualilan hills would need to be pioneered and mined back. The geotechnical data required to support a 55° overall pit slope in the limestone unit of the Hualilan Hills was not available, and beyond the scope of the study. It should be noted that overall pit slopes greater than 55° are commonly observed in nearby limestone quarries.

Finally, the possibility of a low-grade heap leach as an additional processing stream was unable to be evaluated due to long-lead (90 day) metallurgical column test work in progress at the time of this release. Heap leaching is potentially an important opportunity to consider as studies continue as it could provide a low cost pathway to process the low-grade mineralisation via a large open pit.

The next phases of the study will be focussed on the practicality and economics of a larger open pit operation while continuing to investigate and optimise a concurrent or sequential UG mining option.

Table 3 - Key Scoping Study LOM Financial and Physical Outcomes

Key LOM Production Statistics	Year 1	LOM
Life of Mine (LOM)		7 years
Ore tonnes mined		7.1 million
Ore processing rate		1,050,000 tpa
Average Annual gold production (recovered)		116,000 oz
Average Annual silver production (recovered)		440,000 oz
Average Annual zinc production (recovered)		9,175 t
Average Annual lead production (recovered)		474 t
Average Annual production (Au equivalent)[3]		141,000 oz
Key LOM Financial Metric	**US$**	**A$**
Revenue (LOM)	$1,465 million	$2,254 million
EBITDA (LOM)	$738 million	**$1,135 million**
C1 Cost (Real – US$/ oz)	$527/ oz	$811/ oz
ASIC (real – US$/ oz)	$830/ oz	$1277/ oz
Free cashflow (Pre-tax) LOM	$527 million	$811 million
Free cashflow (Average per annum)	$78 million	$120 million
Pre Tax NPV[5]	$409 million	$630 million
Post Tax NPV[5]	$295 million	$454 million
Payback Period (Pre-Tax)		1.25 years
Payback Period (Post Tax)		1.25 years
Project IRR (Pre-Tax Real)		75.2%
Project IRR (Post Tax Real)		66.0%
Pre Production Capital Costs	**US$**	**A$**
Pre-production capital	$134 million	$206 million
Contingencies	$15 million	$23 million
Total Pre-Production Capital	$152 million	$234 million
Key Social Metrics	**US$**	**A$**
LOM royalties and corporate taxes	$166 million	$256 million
LOM Expenditure	$772 million	$1,187 million
LOM Economic Value Add Argentina	$938 million	$1,443 million

The focus on the underground option allowed for updating of the Mineral Resource Estimate at depth (below an optimised shell) which was previously categorised as Inferred Mineral Resource (Table 13). The focus on underground mining scenarios allows the mineralisation at depth to be categorised into both Indicated Resource and Inferred Resource according to the drill hole spacing and confidence in the geological model.

Opportunities

The Company has identified several clear and material opportunities for improvement of the SS outcome which will be evaluated in the next stage of studies. These include:

- A low-grade zinc concentration pathway, based on a recent flotation test on a composite grading 0.36% Zn which produced a saleable Zn concentrate grading 48% Zn. Based on prior flotation test work, an assumption was used in the Study that an economic zinc concentrate was only achievable from at a grade >1.5% Zn. The MRE contains approximately 267,000t of zinc of which only 70,000t is accessed in the Scoping Study mine plan. The ability to economically recover part of the additional 197,000t of zinc in the MRE could significantly enhance economics, given the recovered portion of the ~70,000t of zinc generates US$132m revenue based on the Study forecasts.
- Further improvement to the underground stope optimisation, development sequence and production scheduling. The underground stope optimisation was undertaken using an assumption of US$1700/oz gold. Additionally, some improvements in production and development unit costs in the order of 10-20% have already been identified in the intervening period since running the stope optimisation. These improvements in production and development costs are yet to be incorporated into the optimisation, and are likely to result in additional stopes being included in the mine plan. Additionally, optimisation included a Pseudoflow analysis on the underground design to remove uneconomic areas that sit above the stope cut-off grade. Pseudoflow removed 832kt containing 72,000 oz AuEq[3] from the underground mine plan that may be profitable at current spot prices and revised operating costs.
- The improvement in underground optimisation includes reviewing the staging of development during the pre-production period to optimise CAPEX whilst trading off against ensuring access to the highest value stopes in early phases of the UG mine.
- Recovery of the 30-metre crown pillar design which has been left between the base of the open pits and the underground workings. This crown pillar design contains approximately 15,000 Oz AuEq[3]. The study currently assumes no recovery of this crown pillar, however additional geotechnical information may support the recovery of this crown pillar.
- Inclusion of a heap leaching option which provides a process path for a significant proportion (~50%) of the MRE that was excluded in the high-grade/ low-tonnage SS production model. Preliminary column testing on a low-grade composite yielded promising results. As a result of this, a panel of column tests were initiated to test the three material types separately at a range of different head grades. Results from this current panel of column tests will not be

available until December 2023, but a positive outcome has potential to add significant value to the project.

- Reduction in open pit mining unit cost through owner-operator and bulk mining efficiencies. A unit cost of US$3.00/ t was assumed for the Study, initially as a conservative estimate based on the predicted reduced scale of the open pit operation, and later to account for contractor premiums. However, preliminary first-principles cost modelling by the Company, and discussions with equipment vendors around collaboration and operating partnerships, indicates that an owner operated unit cost around US$2.00/ t may be achievable at scale. This impact of a reduced mining unit cost is even more pronounced in a high-volume mining scenario that incorporates a low-grade heap leach. This cost estimate is supported by localised benchmarking at other owner-operator OP mines in Argentina.
- Potential processing of the Au-Ag concentrate on site to produce gold and silver dore. The project is forecast to produce 412kt of Au-Ag concentrate containing 634Koz Au and 1.9Moz Ag over the Life of Mine. The treatment of this concentrate on site to produce gold and silver dore rather than its transport and sale to off-takers as a concentrate could result in combined cost savings and additional revenue to the project of over US$165 million (before costs) based on the SS production forecast.

Next Steps

Next steps to add to the robustness of the current project and provide a pathway for future development for the project include:

- Taking receipt of the final results for the suite of Column Leach tests are currently underway, which will allow for an assessment of the viability of Heap Leach as a potential processing pathway for the low-grade mineralisation;;
- Completion of additional flotation testing on the potential low-grade zinc concentration pathway;
- Completion of additional flotation testing, including locked-cycle and variability test work, which will be required to provide sufficient data for the PFS;
- Testing to determine the liberation of the gold and silver in Au-Ag concentrate and evaluate options to produce dore on site from the Au-Ag concentrate.
- Development of a detailed first-principles open pit mining cost model, in collaboration with equipment vendors, to evaluate the potential owner operated bulk mining efficiencies;
- Completion of a suite of CIL test work (with dual-laboratory verification) to allow Au and Ag recoveries and NACN consumption to be modelled for both the high-grade and low-grade mineralisation, thereby allowing for a definitive evaluation of the CIL processing option;
- Update the first-principle cost models for the processing and general and administrative areas such that they can be utilised to assess the cost impact of variable process throughputs;

- Update the processing cost model to be inclusive of heap leaching, should the Column Test results be positive;
- Complete geotechnical data gathering, including: additional core logging; collection of Point Load Test data from existing drill core; gathering of televiewer data from existing drill holes; and, any drilling of additional geotechnical test holes;
- Updating the underground stope optimisation for final underground mining and development cost forecasts;
- Further optimisation of the open pit/ underground interface and which components of the orebody should be included in each; and,
- Additional drilling of some of the drill targets identified in the Hualilan regional exploration programme.

Mine Design and Production target

The Study mine plan was designed to supply Potential Mining Inventory (PMI) to the processing plant at a rate of approximately 1 Mtpa. The mine design is comprised of three high-grade starter open pits(North, Central, and South) which will be mined using conventional excavator and truck techniques over 4 years by a mining contractor. It was necessary to include a starter pit as an underground startup would not provide sufficient waste material for the construction of the Tailings Storage Facility (TSF). Additionally, a starter pit offers a reduced production risk profile in the early year of operation.

Owner-operated underground mining was modelled on the basis of sub-level open stoping (SLOS), with a 30 m crown pillar between the pit floor and the upper most underground stope. Based on the available geotechnical data, and on similar operations, the following stoping assumptions were used for the UG mine design:

- Level spacing (floor to floor) - 20 m;
- Strike length (regardless of stope width) - 20 m;
- Minimum stoping width - 2.5 m; and
- Minimum crown pillar of 30 m below the designed open pits.

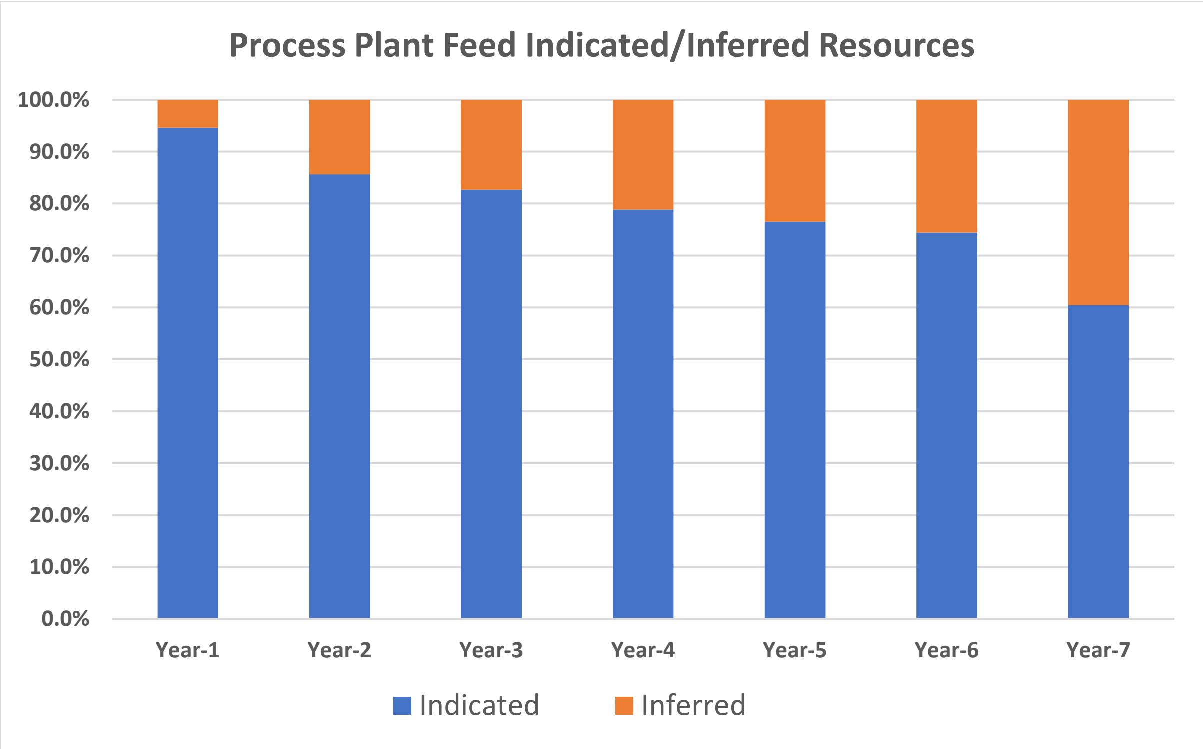
Hualilan will utilise paste backfill and as such there are no further requirements for UG pillars . The three underground portals have been located outside the open pits to decouple the UG development schedule from OP production rates and to provide flexibility to start underground development earlier. This was a critical factor in the earlier trade off studies that negatively affected the NPV of the combined open pit and underground options, with underground mining unable to commence prior to the completion of open pit mining creating material impact to NPV.

During the pre-production year, PMI will be extracted from the pit and stockpiled, and waste will be sent to TSF for construction of the first embankment lift, which will initially store two years of processing production. Underground mining will also commence at the start of the pre-production year in the north and south UG zones. In the first year of production, the plant will be fed with PMI from the stockpiles (SP), open pit and underground. From the second year onwards, the plant will be supplied with PMI primarily coming from the underground mine.

When processing commences there will be approximately 655 kt of PMI on stockpiles, which is equivalent to approximately 7 months mill feed, and approximately 400 kt of this is high-grade SP. This also opens the possibility to commence processing earlier if construction and commissioning is completed ahead of schedule.

The underground mine plan has three operating areas (north, central and south), each of which are capable of being scaled up in response to production issues in another. There is also scope to share equipment between the 3 underground mining areas, however this option has not been included in the Scoping Study analysis.

Figure 1: Breakdown of Scheduled Process Plant Feed by Resource Category



A breakdown of the PMI schedule for processing across the life of the project is shown in Figure 1. Over 80% of the PMI schedule for processing in classified as Indicated Resource, with the balance classified as Inferred Resource. In the early years of production the percentage of Indicated Resource processed is higher, with an average of 88% Indicated Resource scheduled for processing in the first 3 years of plant operation.

At the completion of the SS production plan the unmined component of the Hualilan Project MRE is:

53.5Mt at 1.0g/t AuEq (0.8 g/t Au, 4.9 g/t Ag, 0.32% Zn, 0.06% Pb) - containing 1.7 Moz AuEq

Capital Costs

The capital cost estimate was prepared by Mining Plus and a number of independent external consultants retained by CEL. There was limited use of benchmarking, with costs generally sourced from vendor quotes/ indicative prices, or detailed first principal cost analysis using vendor quotes based on the preliminary project design. Where benchmarking was used to provide any capital costs the primary source was the Mining Plus internal cost database, augmented by Challenger's consultants databases. Where benchmarking has been used to provide capital cost estimates this has been specifically stated in this ASX Release and the Attached Scoping Study report.

The cost estimate is expressed in Q3 2023 US$ and used the USD/ARS exchange rate at the time the quotation was provided (average 200 ARS/USD) for any in-country costs provided in ARS. In practice in Argentina most cost quotes are generally provided in USD and converted into ARS based on the prevailing USD/ARS rate. The costs do not include allowances for escalation or exchange rate fluctuations. All costs are exclusive of the Argentinian value added tax (VAT), which is applied separately in the financial model used for economic evaluation.

The capital cost estimate for this scoping study has a target accuracy range of ±15% where costs have been sourced from vendor quotes or first principles analysis. The costs developed by benchmarking have a target accuracy of ±35%.

Table 4: Summary Capital Cost Estimate (all figures in US$)

Description	Pre-production Capital Costs	Sustaining Capital Cost	Total Capital Cost
1. Open Pit Development (inc. Truck Shop, Wash Bay, Tyre Bay)	5.8		5.8
2. Underground Development (inc. paste plant)	21.8	45.0	66.8
3. Process Plant	59.0	8.9	67.9
4. TSF	5.4	3.2	8.6
5. On-site Infrastructure	8.7	1.5	10.2
6. Off-site infrastructure	0.0	0.0	0.0
7. Owners Costs	15.6		15.6
8. Indirect Costs	2.7		2.7
9. Contingency	14.7	0.5	15.2
Total Capital Expenditure	**133.7**	**59.0**	**192.7**
10. Other Pre-production Costs[3]	18.4		18.4
Total Pre-Production Capital	**152.1**	**59.0**	**211.1**

1. All figures are rounded to reflect the relative accuracy of the estimate.
2. Totals may not sum due to rounding as required by reporting guidelines.
3. Pre-production costs are operating costs that occur prior to the mill operating.

The following areas were included in the Pre-Production Capital Cost estimate:

1. Open Pit Mine (open pit mine development, equipment fleet, pre-stripping/ pioneering and supporting infrastructure and services);
2. Underground Mine (underground development, equipment fleet, paste backfill plant and supporting infrastructure and services);
3. Process plant (gold-silver, zinc-gold-silver, and lead–gold-silver concentrates), conventional 1-1.2 Mtpa concentrator and Flotation Tails Leach circuit with supporting plant infrastructure and services;
4. TSF;
5. On-site infrastructure (earthworks, sitework, roads, water treatment and distribution, camp and other general facilities);
6. Off-site infrastructure;
7. Owners Costs including EPCM, spares, first fills, transport costs and import costs;
8. Indirect costs;
9. Other Pre-production Costs (other operating costs prior to commercial production/ processing) ; and,
10. Contingency (applied at +15%).

Total capital costs are US$133.7 million, not including US$18.4 million of capitalised mining costs. Total Pre-development capital costs of US$152.1 million are summarised in Table 4. More complete details of pre-development capital costs are provided in the Chapter 24 of the Scoping Study.

Operating Costs

The operating cost estimates in the Study are based on: contractor operated truck and excavator open pit mining; owner operated underground mining via longitudinal SLOS with paste backfill; processing which includes gravity recovery, conventional flotation and with Floatation Tail Leach (FTL); and, deposition of the tails not consumed in the paste backfill process in a Tailing Storage Facility (TSF).

Operating cost estimates have generally been derived from first principles costs analysis prepared by external consultants, rather than by benchmarking. These cost estimates include local labour rates derived from San Juan industry standards, costs sourced by vendor/ supplier quotations both in Argentina and externally, and productivity rates that reflect the local workforce and conditions. Unless otherwise stated in this ASX Release or the Annexure Scoping Study Report the operating cost estimates have an expected accuracy range of ±15%.

The operating estimate is expressed in Q3 US$ and used USD/ARS exchange rate at the time the quotation was provided for any in country costs provided in ARS. In practice, in Argentina, most quotes are generally provided in USD and converted into ARS based on the prevailing USD/ARS. This includes diesel, equipment hire for both general and specialised mining equipment, reagents and consumables. The exceptions are Government provided services such as grid power and in-country labour. Generally, the rate of increase in the ARS price tracks the decline in the ARS/USD rate for power and labour, however there is a 1-3 month lag in the repricing of ARS denominated costs. For any ARS denominated input costs, such as grid power and labour, Challenger has used the 2022H2/2023H1 prices, as converting the current local ARS costs into a USD denominated price at the prevailing ARS rate would artificially lower these input costs on a USD basis.

Underground Mining Costs

Table 5 provides a breakdown of the underground mining costs which were derived from a detailed first principles analysis prepared by external underground mining consultants via a bottom up analysis. The mine operating cost estimate included the costs associated with stope preparation, drilling, blasting, ground support, backfill, underground loading and hauling and material transport to the primary crusher on surface, as well as support and ancillary equipment operations and maintenance, power, direct labour, and mine operations supervision staff.

Table 5 – Underground Mining Costs

<table>
<tr><th>Category</th><th>US$/t Processed Incremental</th><th>US$/t Processed Total</th></tr>
<tr><td>Stoping</td><td>4.56</td><td rowspan="2">7.75</td></tr>
<tr><td>Slot Rises</td><td>3.19</td></tr>
<tr><td>Production Bogging</td><td></td><td>2.89</td></tr>
<tr><td>Trucking</td><td></td><td>4.43</td></tr>
<tr><td>Mine Auxiliary – Pumping</td><td>0.05</td><td rowspan="6">14.66</td></tr>
<tr><td>Mine Auxiliary – Ventilation</td><td>0.51</td></tr>
<tr><td>Mine Auxiliary – Backfill</td><td>3.13</td></tr>
<tr><td>Mine Auxiliary – Power</td><td>2.67</td></tr>
<tr><td>Mine Auxiliary – Labour</td><td>4.07</td></tr>
<tr><td>Mine Auxiliary – General</td><td>4.22</td></tr>
<tr><td>Mine Supervision</td><td></td><td>5.01</td></tr>
<tr><td colspan="2">Total Underground Mining Cost</td><td>34.74</td></tr>
</table>

Process Costs

The process operating cost estimate accounts for the operating and maintenance costs associated with the process plant operation, supporting services, infrastructure, and tailings filtering. Operating costs associated with the paste backfill plant are included in the mine operating cost estimate.

Process plant operating costs have been estimated by Challenger's consulting metallurgists from first principles, using mechanical equipment specifications for estimation of power consumption, metallurgical test-work for reagent and grinding media consumption estimates, preliminary labour schedules and salary build-ups for process labour and maintenance labour. The cost of spares was estimated as a fixed percentage of 5% of the mechanical equipment supply cost.

Quotations for consumables such as reagents, lime, binder and grinding media were obtained from suppliers inclusive of transportation to site. A unit power cost of US$0.07/ kWh was assumed with power consumption based on the results of comminution testing and desired grind size. An allowance equal to the power usage of the comminution circuit was applied to the rest of the process plant. Grid power is currently US$0.06/ kWh in San Juan.

The PMI feed has been divided into three separate categories based on gold and zinc grades. Each type of PMI has a slightly different flow sheet:

1. **Type A material -** the lower grade PMI containing <1.5 g/t Au and <1.5% Zn (Type A) processed via bulk flotation with cleaning stages.
2. **Type B material -** The higher grade PMI ≥1.5 g/t Au with <1.5% Zn (Type B) follows the same flow sheet as Ore Type A with the addition of flotation tails leach (FTL).
3. **Type C material -** For the PMI containing ≥1.5% Zn (Type C) a stage of Pb-Cu rougher flotation and Zn flotation is added.

The plant has been designed to batch all three PMI types by bypassing the Cu-Pb and Zn flotation and/ or the FTL circuit. For this reason, an availability of 72% has been assumed for the flotation circuit.

Given the slightly different flowsheets, the three types of PMI have different reagent consumption which drives process costs. Annual and LOM operating costs for the process and surface infrastructure for the three types of PMI are shown in Table 6, Table 7 and Table 8 with gravity/ CIL costs In Table 9.

Table 6: Process Operating Cost for PMI Type C

Type C (Au≥ 1.5 g/t Au, Zn≥ 1.5%) Sequential Flotation + FTL		
Category	Cost	
	Annual US$	Unit US$/t Ore
Operating Labor	1,915,984	1.82
Maintenance Labor	1,084,274	1.08
Power	2,100,000	2.10
Reagents and Consumables	9,499,572	9.50
Spares	1,712,329	1.71
Assays	100,000	0.10
Totals	**16,312,159**	**16.31**

Table 7: Process Operating Cost for PMI Type B

Type B (Au≥ 1.5 g/t Au, Zn< 1.5%) Bulk Flotation + FTL		
Category	Cost	
	Annual US$	Unit US$/t Ore
Operating Labor	1,915,984	1.82
Maintenance Labor	1,084,274	1.08
Power	2,100,000	2.10
Reagents and Consumables	5,306,407	5.31
Spares	1,712,329	1.71
Assays	100,000	0.10
Totals	**12,118,994**	**12.12**

Table 8: Process Operating Cost for PMI Type A

Type A (Au< 1.5 g/t Au, Zn< 1.5%) Bulk Flotation no FTL		
Category	Cost	
	Annual US$	Unit US$/t Ore
Operating Labor	1,915,984	1.82
Maintenance Labor	1,084,274	1.08
Power	2,100,000	2.10
Reagents and Consumables	2,443,723	2.44
Spares	1,712,329	1.71
Assays	100,000	0.10
Totals	**9,256,310**	**9.26**

Table 9: Process Operating Costs Gravity plus CIL

Process Costs Gravity plus CIL		
Category	Cost	
	Annual US$	Unit US$/t Ore
Operating Labor	1,676,320	1.68
Maintenance Labor	964,442	0.96
Power	2,100,000	2.10
Reagents and Consumables	5,460,322	5.46
Spares	1,092,538	1.09
Assays	100,000	0.10
Totals	**11,393,622**	**11.39**

Economic and Sensitivity Analysis

Project economics for are presented in the table below. These base case economic results for the Hualilan Gold project are favourable, however further work to improve the economics is ongoing.

Table 10: Scoping Study Economic Summary

	Units	
Project Life	Years	7.75
Total Potential Mining Inventory (contained)[1]	7.1Mt @ 3.57 g/t Au (for 816 koz), 13.98 g/t Ag (for 3,194 koz), 1.38% Zn (for 197,066 klbs) and 0.11% Pb for (15,066 klbs)	
Payable Metal		
Gold Sales (payable Au)	Oz	737,922
Silver Sales (payable Ag)	Oz	1,776,948
Zinc Sales (payable Zn)	klbs	114,683
Lead Sales (payable Pb)	klbs	6,645
Revenue	**$M**	**1,465**
Treatment and refining costs	$M	(58.7)
Transport and freight costs	$M	(83.4)
Net Revenue before Royalties (NSR)	**$M**	**1,323**
Royalties and state taxes	$M	(166.3)
Net Revenue after Royalties	$M	1,157
Mining Operating expenses	$M	(287.7)
Process Operating expenses	$M	(93.1)
G&A Operating expenses (during production)	$M	(37.7)
Operating Margin (EBITDA)	**$M**	**738**
Initial Plant and Infrastructure Capex	$M	(133.7)
Initial Mine Development and Mobilisation	$M	(18.42)
Initial Capex (incl. Pre-production Operating costs)	**$M**	**(152)**
Underground Mine Sustaining capex	$M	(45.0)
Mine infrastructure and plant sustaining capex	$M	(14.0)
Sustaining Capex: Total	**$M**	**(60)**
Total Capital and Sustaining Capital	**$M**	**(211)**
Undiscounted Cashflow Pre-tax	**$M**	**527.0**
Tax Payable	$M	(142.5)
Undiscounted Cashflow after tax	**$M**	**384.5**

1: The total PMI contained underpinning the above production target has been prepared by a Competent Person or Persons in accordance with the requirements of the JORC (2012) Code. Refer to JORC tables, Qualifications and Competent Persons Statements. Based on assumed throughput of 1.05 Mtpa.

2. All figures are presented in nominal United States dollars, tax is applied at a flat corporate rate of 35%, unadjusted for inflation

3. Rounding errors may occur

The project is most sensitive to changes in the gold price. The NPV[5] and IRR sensitivity to changes in gold price are shown in Table 10 on a LOM basis.

The project sensitivity to +/-10% changes in key operating parameters are also shown below. These include changes to gold price, Plant and G&A opex, mining opex, all metals prices, capex and overall opex with their sensitivity to the staged post-tax, ungeared NPV[5] presented in Figure 10.

A review of the sensitivity figures indicates that the post-tax, ungeared NPV_5, and IRR are most sensitive to changes in the gold price, and all metals prices.

The project is more sensitive to changes in operating costs (mining, processing, site G&A) than capital costs, as a result of the low base case capital costs for the project.

Table 11 Gold Price Project Sensitivity

Gold Price Sensitivity - USD	$/oz	1,650	1,750	1,850	Spot (1,975)
Project Net Cashflow, Pre-Tax (USD)					
Project Net Cashflow, Pre-Tax	$M	462.43	527.03	591.63	677.45
Pre-tax NPV[5]	$M	355.70	409.03	462.37	533.24
Pre-tax IRR	% p.a.	66.23	75.23	84.29	96.50
Payback period from production start	Years	1.5	1.3	1.3	1.3
Project Net Cashflow, Ungeared, Post-Tax (USD)					
Project Net Cashflow, Post-Tax	$M	342.50	384.49	426.47	482.26
Post-tax NPV[5]	$M	260.40	294.78	329.05	374.53
Post-tax IRR	% p.a.	59.11	66.04	72.66	81.18

Due to the low level of capital expenditure required to go into production, the project economics are not overly sensitive to CAPEX within the Scoping Study estimation accuracy of plus or minus 35%. To illustrate, consider the following:

- A 20% reduction in gold price alone would reduce the LOM project NPV[5] (pre-tax) to around US$221.56M, whilst a 20% increase would deliver a LOM project NPV[5] of US$596.5M;
- A 20% increase in operating cost reduces the LOM project NPV[5] to around US$302.02M, whilst a cost reduction of 20% results in US$516.04M NPV_5;
- A 20% increase in all metals prices (including gold) to a LOM NPV[5] of US$622.01 M, whilst a 20% reduction delivers a NPV[5] of US$196.06M.

The sensitivity analysis demonstrates the project would require a grade reduction or gold price reduction in the order of 44% to reach the breakeven gold price of US$983 (A$1512), from an NPV perspective.

Funding

Challenger has 100% ownership of the Hualilan Gold Project, with US$15M unsecured debt, no other covenants, and no security held over the project. This clean ownership structure enhances opportunities and provides maximum flexibility for potential funding structures for the Project's development.

The Study has provided positive economic metrics and a timetable of activities to deliver key development milestones that directors and management believe is conducive to the funding of the Project. The positive technical and economic fundamentals provide a platform for discussions on debt and equity finance, and forward sales arrangements.

The Company has held early stage fruitful discussions with potential project finance providers in Argentina, the US and Europe, and various international royalty and stream providers. These initial discussions have been positive, and indicate that the Company will likely have a range of non-dilutive finance options available for consideration.

CEL's board has experience in financing and in developing projects internationally, and several board members and senior executives have been involved with Challenger since the listing of the Company in 2019. Current management has international experience in bringing companies into production, including Dominion Minerals Corp, which acquired and developed the Cerro Corcha Gold/ Copper project in Panama, with a project NPV in excess of US$500 million, and the Antas Copper Mine in Brazil, which was acquired by Oz Minerals Limited in 2018 for approximately $400 million.

The Company's major shareholders comprise high quality investment funds including the BlackRock Group (BlackRock Inc. and its subsidiaries). The Company's aim will be to avoid dilution to existing shareholders as much as possible.

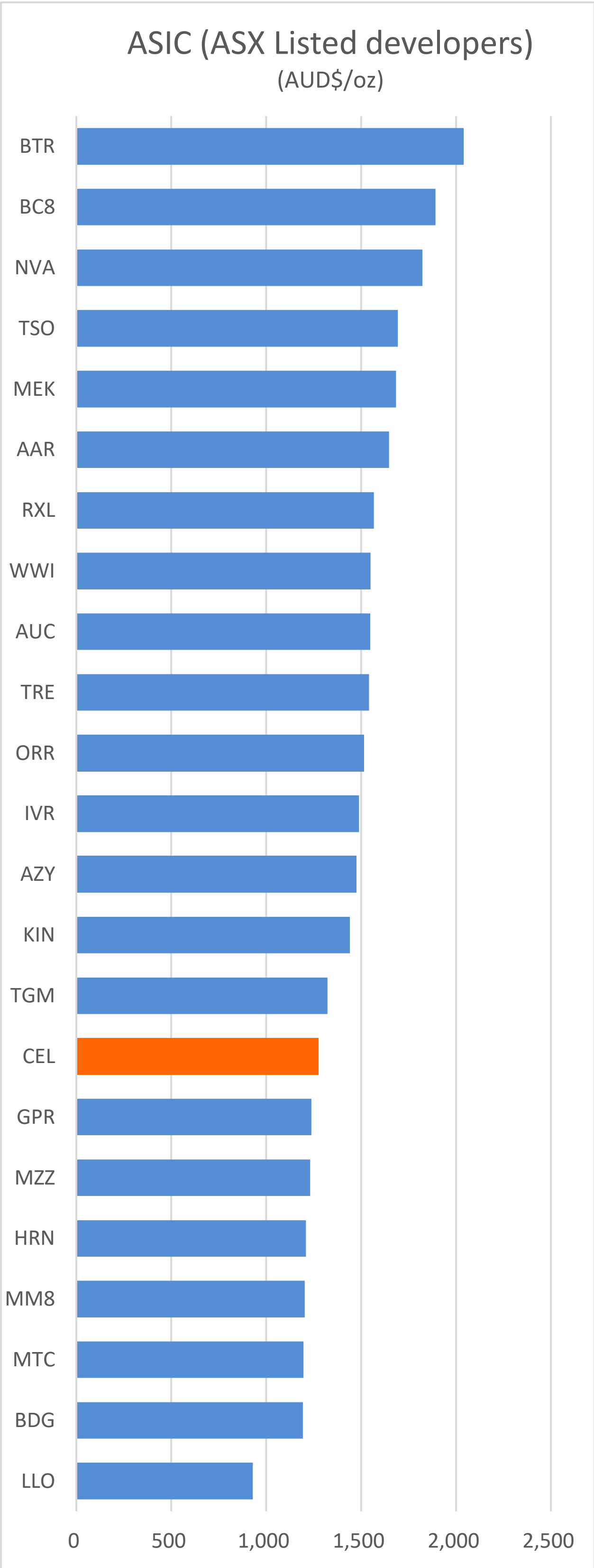
All the material assumptions on which the forecast financial information is based has been included in this Scoping Study. For the reasons outlined above, the board believes that there is a 'reasonable basis' to assume that future funding will be available and securable.

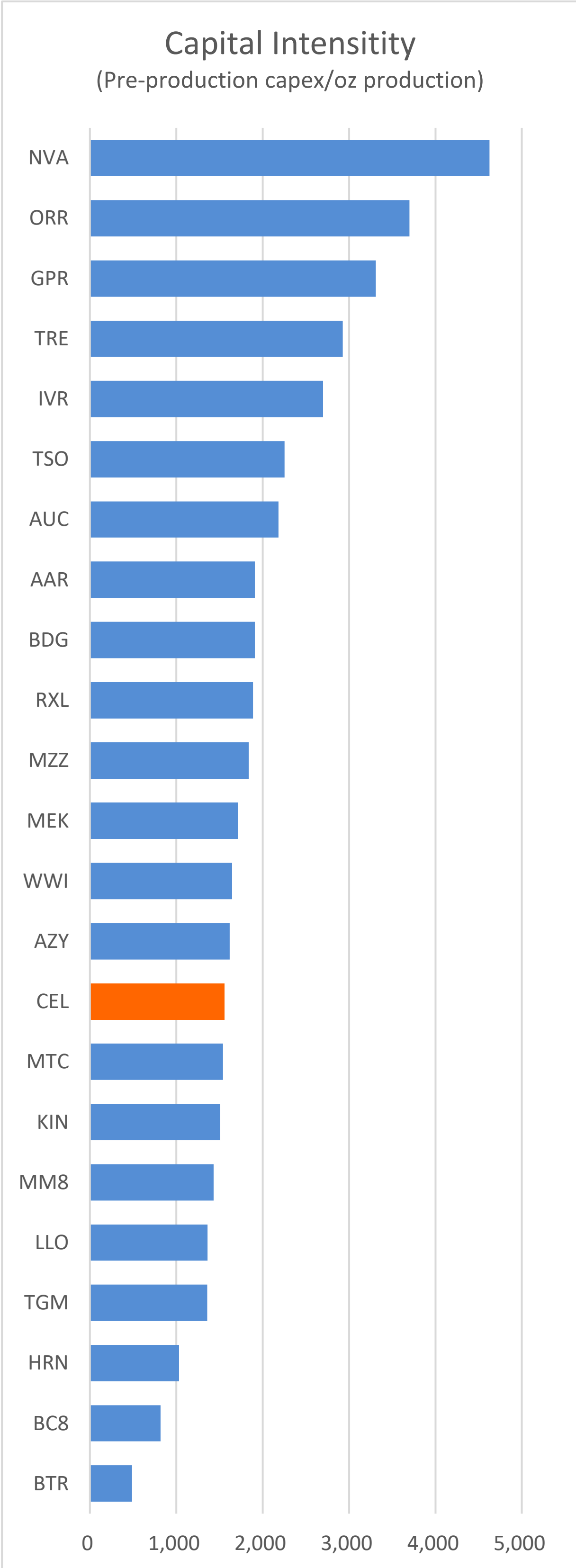
Project Positioning

The Hualilan Gold Project would be a low-cost producer compared with current Australian producing gold companies, with a projected average AISC of $1,277/ oz (US$830) over the LOM, placing the Project the lowest quartile of ASX listed gold producers[5]. The average LOM Study production target would place the Company inside the current Top 20 ASX listed gold producers[6].

The projects position relative to other ASX listed peers (Developers <$500m market cap) is shown in Figures 3 and 4 over the page. Details of peer comparisons over the last 12 months are provided in Appendix B of this release.

[5] Supporting data contained in Appendix 2 CEL ASIC costs based on the Study analysis.
[6] Supporting data in contained in Appendix 2 CEL production based on the Study production target in AuEq – see [3] for AuEq information





Source Gold Nerds and ASX Releases – see Appendix 2

CIL Process Option (Case 2)

Following higher than expected recoveries from metallurgical test work which was re-examining cyanide leaching technology on Hualilan mineralisation, the decision was made to delay the release of the study, so that follow up testing could be undertaken, potentially allowing for CIL to be included as an option in the study. This initial test work, undertaken on a composite designed to reflect the mineralisation in a large open pit mining scenario, indicated gold recoveries of up to 89% at a primary grind size ($_{P80}$) of 100 µm, and low sodium cyanide consumption of 0.7 kg/t.

Follow up testing on a composite designed to be representative of the higher-grade (predominantly skarn) PMI in the Scoping Study mine plan was conducted at SGS Laboratories in Lakefield, Canada. The results of this test work, which were received on October 24th, indicate gold recoveries of 85% and silver recoveries up to 62% from a combination of gravity and CIL in the high-grade skarn mineralisation. The test results indicate that the finer grind tests were starved of cyanide, and there is a significant variation in the final calculated head grades for each set of tests. Accordingly the results may potentially understate the recoveries using cyanide leaching. Additional and more definitive testing at SGS and a second lab is programmed.

Table 12: CIL (Case 2) Key Assumptions

Metallurgical Recoveries and Concentrate Payability	Recovery (%)	Payability[1] (%)
Gold	**85.1%**	**99.75%**
Silver	**62.0%**	**99.75%**
Unit operating Costs	**Unit**	**Unit Cost**
Processing Cost (gravity plus CIL)	**US$/t processed**	**12.37**
Capital Costs	**Contingency**	**Total Capital Cost**
Capital Expenditure	**$11.8m**	**$113.4m**
Total Pre-Production Capital	**$11.8m**	**$131.8m**

Table 13 - CIL (Case 2) Key Financial Outcomes

Economic metric	Flotation	CIL	Differential
Gross Revenue (LOM)	US$1,465.0m	US$1,248.1m	-216.9
EBITDA (LOM)	US$735.9m	US$679.63m	-56.3
C1 Cost (Real – US$/oz)	$ 527 / oz	$ 543 / oz	16.0
ASIC (real – US$/oz)	$830 / oz	$ 849 / oz	19.0
Pre Tax NPV[5]	US$409.0m	US$381.14m	-27.9
Post tax NPV[5]	US$294.8m	US$277.32m	-17.5
Payback Period	1.25 years	1.25 years	n/a
Pre production Capital	US$152.1	US$131.8	-20.3
Project IRR (Pre-Tax Real)	75.2%	78.3%	2.9%

The CIL economic evaluation used the same cost assumptions as the Gravity/Flotation/FTL, with gold and silver recoveries and payability, and process operating costs updated as per Table 12 Additionally, the CIL processing case has the advantage of a $20.3M reduction in up front CAPEX, given the flotation and regrind circuits are not required.

The results of an economic analysis of the CIL processing case produce the following results, which are compared to the flotation case Iin Table 13:

The CIL case has a pre-tax NPV[5] of $381.1M (A$586.3M), US$28M below the Flotation case, but with the same 1.25 year payback period. This is within 7% of the Flotation case, with the advantage of $20 million less pre-production CAPEX being required. If gold recoveries of 89% (which were achieved in the initial CIL testing) are used for economic modelling, the CIL case generates a Pre-tax NPV_5 of US$422.3m (A$650m) at an ASIC of US$823/ oz and a payback period of 13 months (previously 15 months), which is a superior economic outcome to the flotation case. Additionally, CIL with a heap leach option may have a lower CAPEX profile and would allow for scaling up into a larger open pit option, which will be evaluated once column test results are returned.

Accordingly, the CIL Processing case will be evaluated in the next round of studies subject to the results of a suite of CIL testwork (with dual-laboratory verification) to allow Au and Ag recoveries and NACN consumption to be modelled for both the high-grade and low-grade mineralisation, thereby allowing for a definitive evaluation of the CIL processing option.

Vesting of Performance Shares and Performance Rights

Challenger advises that the following Performance Shares and Performance Rights have vested having met the applicable vesting criteria with the release of this Scoping Study. The Company notes that all owners of the vesting performance Shares and Performance Rights are existing shareholders with freely trading existing shares.

Performance Shares	Number	Expiry
Class B	60,000,000	N/A
Performance Rights	**Number**	**Expiry**
Performance B	9,500,000	4 July 2026
Performance C	2,500,000	N/A

The vesting conditions for the Performance Shares and the Performance B Rights were as follows:

- Completion and announcement by CEL (subject to the provision of information allowable at the time of completion) of a positive Scoping Study (as defined in the JORC Code) on either Project by an independent third-party expert which evidences an internal rate of return of US Ten Year Bond Rate plus 10% (using publicly available industry assumptions, including deliverable spot commodity / mineral prices, which are independently verifiable) provided that the total cumulative EBITDA over the project life is over US$50m.

The vesting conditions for the Performance C Rights were as follows:

- Upon the successful completion of a Scoping Study that leads to an announcement that the Hualilan Project will progress to a Pre-feasibility study (PFS).

Ends

This ASX release was approved by the Board.

For further information contact:

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Previous announcements referred to in this release include:

21 July 2020 - Further outstanding metallurgical results from the Hualilan Gold Project
22 Feb 2021 - Gold recoveries of 91-94% from Phase 1 metallurgical testing at Challenger's Hualilan Gold Project
17 May 2021 - CEL Delivers Exceptional Metallurgical Test Work Results from the Hualilan Gold Project
3 May 2022 - Outstanding results from metallurgical testing significantly upgrade CEL's Hualilan Gold Project
29 May 2023 - CEL Delivers Significant High-Grade Mineral Resource Estimate of 1.6 Moz at 5.0 g/t AuEq[1] within 2.8Moz AuEq[1] at Hualilan

COMPETENT PERSON STATEMENT – EXPLORATION RESULTS AND MINERAL RESOURCES

The information that relates to sampling techniques and data, exploration results, geological interpretation and Mineral Resource Estimate has been compiled Dr Stuart Munroe , BSc (Hons), PhD (Structural Geology), GDip (AppFin&Inv) who is a full-time employee of the Company. Dr Munroe is a Member of the AusIMM. Dr Munroe has over 20 years' experience in the mining and metals industry and qualifies as a Competent Person as defined in the JORC Code (2012).

Dr Munroe has sufficient experience of relevance to the styles of mineralisation and the types of deposits under consideration, and to the activities undertaken, to qualify as a Competent Person as defined in the 2012 Edition of the Joint Ore Reserves Committee (JORC) Australasian Code for Reporting of Exploration Results and Mineral Resources. Dr Munroe consents to the inclusion in this report of the matters based on information in the form and context in which it appears. The Australian Securities Exchange has not reviewed and does not accept responsibility for the accuracy or adequacy of this release.

The Mineral Resource Estimate for the Hualilan Gold Project was first announced to the ASX on 1 June 2022 and updated 29 March 2023. The Company confirms it is not aware of any information or assumptions that materially impacts the information included in the announcements and that the material assumptions and technical parameters underpinning the Mineral Resource Estimates continue to apply and have not materially changed.

Table 13 Hualilan Hold Project Mineral Resource Estimate (March 2023)

Domain	Category	Mt	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	AuEq (g/t)	AuEq (Mozs)
US$1800 optimised shell > 0.30 ppm AuEq	Indicated	45.5	1.0	5.1	0.38	0.06	1.3	1.9
	Inferred	9.6	1.1	7.3	0.43	0.06	1.4	0.44
Below US$1800 shell >1.0ppm AuEq	Indicated	2.7	2.0	9.0	0.89	0.05	2.5	0.22
	Inferred	2.8	2.1	12.4	1.1	0.07	2.8	0.24
Total		60.6	1.1	6.0	0.4	0.06	1.4	2.8

Note: Some rounding errors may be present

[1] **Gold Equivalent (AuEq) values - Requirements under the JORC Code**

- Assumed commodity prices for the calculation of AuEq is Au US$1900 Oz, Ag US$24 Oz, Zn US$4,000/t, Pb US$2000/t
- Metallurgical recoveries are estimated to be Au (95%), Ag (91%), Zn (67%) Pb (58%) across all ore types (see ***JORC Table 1 Section 3 Metallurgical assumptions)*** based on metallurgical test work.
- The formula used: AuEq (g/t) = Au (g/t) + [Ag (g/t) x 0.012106] + [Zn (%) x 0.46204] + [Pb (%) x 0.19961]
- *CEL confirms that it is the Company's opinion that all the elements included in the metal equivalents calculation have a reasonable potential to be recovered and sold.*

About Challenger Gold

Challenger Gold Limited's (ASX: CEL) aspiration is to become a globally significant gold producer. The Company is developing two complementary gold/copper projects in South America with the Company's flagship Hualilan Gold Project in San Juan, Argentina containing resources of **2.8 million ounces gold equivalent**.

The Company strategy is for the 100% owned Hualilan Gold Project to provide a high-grade low capex operation in the near term while it prepares for larger bulk gold operation at El Guayabo in Ecuador.

1. **Hualilan Gold Project**, located in San Juan Province Argentina, is a near term development opportunity. It has extensive drilling with over 150 historical and almost 900 CEL drill-holes. The Company has released a JORC 2012 Compliant resource of **2.8 Moz AuEq** which remains open in most directions. This resource contains a high-grade core **9.9 Mt at 5.0 g/t AuEq for 1.6 Moz** AuEq and **29.1Mt at 2.2 g/t AuEq for 2.4 Moz AuEq** within the larger MRE of **60.6 Mt at 1.4 g/t AuEq for 2.8** Moz AuEq. The resource was based on approximately 220,000 metres of CEL drilling. Drill results have included **6.1m @ 34.6 g/t Au, 21.9 g/t Ag, 2.9% Zn**, **67.7m @ 7.3 g/t Au, 5.7 g/t Ag, 0.6% Zn, and 63.3m @ 8.5 g/t Au, 7.6 g/t Ag, 2.8% Zn**. This drilling intersected high-grade gold over 3.5 kilometres of strike and extended the known mineralisation along strike and at depth in multiple locations. Recent drilling has demonstrated this high-grade skarn mineralisation is underlain by a significant intrusion-hosted gold system with intercepts including **209.0m at 1.0 g/t Au, 1.4 g/t Ag, 0.1% Zn** and **110.5m at 2.5 g/t Au, 7.4 g/t Au, 0.90% Zn** in intrusives. The Hualilan Scoping Study demonstrates production of 116,000 oz Au, 440,000 oz Ag, 9175t Zn (141,000 oz AuEq) at an ASIC of US$830/oz over an Initial 7 year mine life. CEL's current program will include a Pre-Feasibility Study, and regional exploration along the previously unexplored 30 kilometres of prospective stratigraphy.
2. **El Guayabo Gold/Copper Project** covers 35 sq kms in southern Ecuador and is located 5 kilometres along strike from the 20.5 million ounce Cangrejos Gold Project[1]. Prior to CEL the project was last drilled by Newmont Mining in 1995 and 1997 targeting gold in hydrothermal breccias. Historical drilling demonstrated potential to host significant gold and associated copper and silver mineralisation. Historical drilling has returned a number of intersections including 156m @ 2.6 g/t Au, 9.7 g/t Ag, 0.2% Cu and 112m @ 0.6 % Cu, 0.7 g/t Au, 14.7 g/t Ag which have never been followed up. CEL's maiden drilling program confirmed the discovery of a major Au-Cu-Ag-Mo gold system spanning several zones of significant scale. The Company has drilled thirteen regionally significant Au-soil anomalies with over 500 metres of mineralisation intersected at seven of these thirteen anomalies, confirming the potential for a major bulk gold system at El Guayabo. The Company reported a **maiden 4.5 Moz gold equivalent MRE**. This MRE is based on 34 drill holes, for 22,572 metres, from the Company's Phase 1 and 2 diamond core drill program at its 100% owned El Guayabo concession. The drilling has focussed on 2 of the 7 anomalies that have returned plus 500 metre drill intercepts and mineralisation remains open in all directions.

[1] Source : Lumina Gold (TSX : LUM) July 2020 43-101 Technical Report

Challenger Gold Limited, Hualilan Scoping Study

MINING PLUS

Document Control Information

CHALLENGER GOLD	Challenger Gold Limited, Hualilan Scoping Study	REVISION	
		No.	DATE
	Hualilan Scoping Study	00	30/10/23

Revision Tracking

Revision	Prepared By	Reviewed By	Issued For	Approved By	Date
00	Choose an item.	Choose an item.	Choose an item.	Choose an item.	Select date
01	Choose an item.	Choose an item.	Choose an item.	Choose an item.	Select date
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Issued For: Review and Comment (RC), Information Only (IO), Implementation (IM), Final Version (FV).

1 EXECUTIVE SUMMARY

1.1 Project Highlights

- Scoping Study (SS) focussed on a high grade starter mine at Hualilan finding:
  - Forecast EBITDA of US$738 M (A$1.1 billion) over Life of Mine (LOM).
  - Rapid payback period of under 1.25 years based on current production target.
  - Forecast for Challenger Gold (CEL) to be one of the lowest cost ASX-listed producers.
- Key operational findings of the Scoping Study are for Hualilan to have:
  - Average annual production target of 116,000 oz gold, 440,000 oz silver and 9,175 tonnes zinc.
  - Global lowest-quartile C1 cash cost of US$528/oz (A$812) and AISC of US$830/oz (A$1277).
  - An initial mine life of 7 years, with mineralisation open at depth potentially extending LOM.
  - Low-risk starter pit followed by conventional sub-level open stope (SLOS) underground mining.
  - A processing facility includes a crusher, mill, gravity recovery circuit, conventional sulphide floatation, and floatation-tails leaching (FTL), with schedule LOM throughput of 1.05 Mtpa.
  - Production schedule is comprised of 81% Indicated Resource and 19% Inferred Resource.
- Compelling financial metrics of the Scoping Study include:
  - Pre-tax NPV5 US$409 M (A$629 M) at US$1,750/oz Au, US$20/oz silver (spot gold price US$1,988).
  - Pre-tax NPV5 increases to A$820 M at current gold (US$1,975) and silver (US$23) prices.
  - Project IRR (Pre-Tax Real) of 75%.
  - Breakeven gold price of US$983/oz.
- Outstanding potential upside:
  - Indicative NPV ignores residual value of the 1.7 Moz AuEq[3] remaining after the SS LOM due to the high-grade/low-tonnage focus and is not considered to reflect the full value of the asset.
  - Initial column testing indicates it may be economically viable to run a heap leach processing option which would assist with recovering some of this mineralisation that has been excluded.
  - The underground optimisation at US$1,700/oz Au excluded 880 kt of stopes grading 2.7 g/t AuEq[3] from the mine plan, which would likely be economic to mine at today's gold price.

- Mineralisation is open at depth, in both directions along strike, and there are numerous regional exploration opportunities on adjacent trends within the 600 Ha footprint.
- Current gold and silver pricing 14% higher than priced used in the Study Assumptions.

1.2 Introduction

Challenger Gold Limited (ASX:CEL) is a gold and base metal exploration and development company with projects in San Juan Province, Argentina (Hualilan) and El Oro Province, Ecuador (El Guayabo).

The flagship Hualilan Project is a zinc-enriched skarn with a late-stage gold overprint. A Mineral Resource Estimate (MRE) dated 29 March 2023 included 60.6 Mt at 1.1 g/t Au, 6.0 g/t Ag, 0.44% Zn and 0.06% Pb containing 2.8 Moz AuEq at an average AuEq grade of 1.4 g/t. This MRE has a high grade core of 1.6 Moz AuEq at 5.0 g/t AuEq. The high grade component of the MRE forms the basis of this scoping study.

This scoping study has examined a range of mining and processing options for Hualilan and includes a preliminary economic assessment which was conducted to identify a combination of the highest-return pathway for the project that can realistically be funded by the Company in the current challenging environment as it moves towards more detailed studies. It is an order of magnitude (+/- 35%) technical and economic study of the potential viability of the MRE, notwithstanding that the majority of the capital and operating cost estimates in the study were done via detailed first principles analysis targeting an accuracy of (+/- 15%).

The options identified in this study will be explored and optimised further in later study phases. In tandem with this a study aimed to explore the viability of a larger open pit based development targeting recovery of the lower-grade material, in addition to the high grade material, will be progressed.

1.3 Property and Location

The Hualilan property is situated in northwestern Argentina in the province of San Juan. It is located within the Department of Ullum, at a distance of approximately 120 km from the provincial capital, San Juan City. The province has a population of approximately 800 thousand, and contains the satellite offices of many of the leading mining service providers.

In 2022, Challenger finalised an agreement to purchase 20,000 ha of land which contains the Hualilan project and the area surrounding the current mineral resources.

The size of the land acquisition was designed to:

- cover possible extensions within 5 kilometres in all directions; and,
- provide a sufficient footprint such that all mine infrastructure including the process plant, camp, office, waste dumps and TSF could be located on land owned by the Company.

The acquisition will greatly simplify the permitting process as the Company progresses towards production by removing the need to reach compensation and access agreements with landowners for mining, the provision of site infrastructure, and private contracts for the supply of water.

1.4 Geology and mineralisation

The project is hosted by limestone of Ordovician age overlain by fine grained marine sandstone and siltstone of Silurian age and intruded by dacite of presumed Miocene age. The mineralised sequence is located east of a locally extensive thrust fault (the Hualilan Fault) which juxtaposes the limestone against coarse sandstone of Miocene age. The Hualilan Fault is part of a network of faults initially responsible for basin formation which have been reactivated in response to arc convergence since Permian time. Also active at that time and during mineralisation are cross-faults which strike approximately 070° and regionally extensive transfer faults striking 120-130°.

Mineralisation is zinc skarn (Zn-Pb-Ag-Cu) overprinted by a later stage, lower temperature gold mineralising event (Au-Ag). Hydrothermal alteration is dependent on the host rock lithology and is consistent with similar typical deposits of this style. Mineralisation occurs in all rock types and so is presumed to be of post-orogenic Miocene or younger age.

1.5 Exploration

There is an incomplete history of discovery and early mining at the project. The discovery of mineralisation is attributed to a mule driver in 1751. Prior to Argentinian Independence, supergene deposits were worked on a small scale by family owners. From 1846 there were mining lease sales to larger companies that installed roasting ovens and cyanide leach circuits from the early 1900's. From the mid 1900's there were various attempts to re-treat tailings and mine high grade remnants with limited success.

Modern Exploration started in 1984. Some retreatments of tailings by cyanidation also occurred at that time. From 1984 to 2005, at least 156 drill holes were completed (total of 17,283 metres), although records are incomplete. Detailed surface mapping, channel sampling, geophysical surveys and metallurgical test work was completed.

From late 2019 until January 2023 Challenger Gold Limited (CEL) completed sampling of historic tailings, surface geophysical surveys (magnetic and IP), surface and underground channel sampling (1,767 samples), 224,180 metres of diamond core drilling and 2,923 metres of Reverse Circulation (RC) drilling, geological modelling, and produced a Mineral Resource Estimation (MRE). All diamond core is drilled with triple tube casing and has average 96.8 % core recovery. Drill core and RC chips are logged for lithology, weathering, alteration and mineralisation. Drill core is also logged for Rock Quality Designation (RQD), structure, and regular Specific Gravity (SG) measurements of core have been made. Drill and channel logging data was captured by geologists and the Geographic Information System (GIS) team then proceeded to verify each of the tabs of the logging.

RC drilling cuttings were sampled over 1 metre intervals. Half-cut drill core and channel samples intervals were taken according to the lithology, alteration and mineralisation with an average sample length of 1.8 metres. Channel samples were cut into the rock face using diamond edge circular cutting tool such that the weight and interval of the sample is like that of the drill core. Duplicate and blank samples were submitted at the rate of 1 duplicate or blank for approximately 30 samples taken. One certified reference pulp sample was included in the sample sequence for every 30-40 samples taken.

All samples, including blanks and duplicates were crushed to approximately 85% passing 2 mm. A 500 g or a 1 kg sub-sample was taken and pulverized to 85% passing 75 µm. A 50 g charge was analysed for Au by fire assay with Flame Atomic Absorption (AA) determination. A 10 g charge was analysed for at least 48 elements by 4-acid digest with Inductively Coupled Plasma - Mass Spectrometry (ICP-MS) determination.

Drill hole data, logs and assays are stored in a customised cloud-based database. Importing the data using Structured Query Language (SQL) Server involves several data checks made by the software. The data is accessible via Microsoft SQL Server Management Studio. There have been no independent audits of the database.

1.6 Mineral Resource Estimate

The MRE for Hualilan that has been used in this Study is dated 29 March 2023. The MRE used data collected from selected historic drill holes (8,030 metres), underground and surface channel samples (1,767 samples), 37 RC drill holes and 762 diamond core holes (total of 227,103 metres) completed by GMSA/ CEL.

31 mineralised domains were explicitly built in Micromine according to the geology with a nominal 0.2 g/t AuEq grade boundary. In fresh rock, a correlation between SG and Fe+S grade was used to estimate density. In partially oxidised rock a constant SG value was applied.

Assay data within the mineralised domains was composited to 2 metres. Top cuts were applied to domains according to the mineralisation style, being most readily determined by the host lithology. Group variography was carried out using Leapfrog Edge software on the composited data from each of the domains for each variable. For each domain, the orientation of the plane of mineralisation was aligned with the interpreted wireframe.

Variograms were modelled in three directions for each domain and each element as well as a downhole direction. The nugget effect was modelled by extrapolation of the first two experimental data points from the down-hole variogram set at the same lag as the composite length.

A block model was set up with a parent cell size of 10 m (E) x 20 m (N) x 10 m (RL) consistent with the drill spacing. Standard sub-celling to 2.5 m (E) x 5.0 m (N) x 2.5 m (RL) was applied to maintain the resolution within the mineralised domains. All relevant variables (Au, Ag, Pb, Zn, Fe and S) in each domain were estimated using Ordinary Kriging using only data from within the domain. A three-pass estimation search was conducted, with expanding search ellipsoid dimensions and decreasing minimum number of samples with each successive pass.

Validation checks included statistical comparison between drill sample grades block estimate results for each domain on standard 40 metre spaced sections. The Mineral Resource has been classified based upon semi-qualitative assessment of the geological understanding of the deposit, geological and mineralisation continuity, drill hole spacing, QC results, search and interpolation parameters, and an analysis of available density information.

The MRE has assumed that near surface mineralisation would be amenable to open pit mining given that the mineralisation is exposed at surface and under relatively thin unconsolidated cover. A surface mine optimiser has been used to determine the proportion of the MRE that would be amenable to eventual economic extraction by open pit mining methods using:

- Au price of US$1,800/oz, Ag price of US$23.40/oz, Zn price of US$3,825/t and Pb price of US$1,980/t;
- Average metallurgical recoveries of 94.9% for Au, 90.9% for Ag and 67% for Zn and 57.8% for Pb;
- Assumed concentrate payability of 94.1% for Au, 82.9% for Ag, 90% for Zn and 95% for Pb;
- Mining cost of US$2.00/t, processing cost of US$10.00/t, transport and marketing cost of US$50/oz AuEq; and,
- 45° pit slopes on the western side of the pit and 55° on the eastern side of the pit.

For AuEq, the following metals and metal prices were been assumed: Au US$1900/oz Ag US$24/oz, Zn US$4,000/t and Pb US$2,000/t. For the AuEq calculation average metallurgical recovery is estimated as 94.9% for gold, 90.9% for silver, 67.0% for Zn and 57.8% for Pb.

Accordingly, the formula used for Au Equivalent was: AuEq (/g/t) = Au (/g/t) + [Ag (/g/t) x (24/1900) x (0.909/0.949)] + [Zn (%) x (40.00*31.1/1900) x (0.670/0.949)] + (Pb (%) x 20.00*31.1/1900) x (0.578/.9490}.

The MRE has been reported consistent with the JORC Code at 0.3 g/t AuEq within the volume defined by the surface mine optimiser, and at 1.0 g/t AuEq below the volume defined by the surface mine optimiser.

1.7 Mining

The Scoping Study for the Hualilan project involved two critical trade-off assessments to optimise processing, mining methods, and sequences for project value enhancement.

After the decision to focus the study on the high grade mineralisation the initial trade-off analysis aimed to determine the ideal processing method and mining approach for a 1.0 Mtpa plant throughput. Nine cases were considered, involving three mining methods (open pit, underground, or a combination) and three metallurgical processes (Flotation for Au and Ag, Flotation for Au, Ag, and Zn, and Carbon in Leach (CIL) for Au and Ag). Economic evaluations and high-level schedules were developed for each case. This study determined that underground mining was the preferred method, however underground mining did not produce enough waste material for infrastructure construction (e.g. Tailings facilities).

The second trade-off focused on defining the optimal pit size for initial tailings waste requirements and selecting the best underground cut-off grade. This combined mining strategy aimed at maximising Net Present Value (NPV) and served as the basis for the Scoping Study.

Pit optimisation analysis was executed using Geovia Whittle® software and the Lerchs-Grossmann algorithm. The analysis considered gold, silver, and zinc, utilising Indicated, and Inferred Resources, noting that lead (Pb) was excluded from the economic analysis although present in the model. The preferred pit size option (RF39 pit) provided the required tailings waste for the tailings embankment and facilitated mining of high grades near the surface.

The underground design and optimisation process was completed utilising Deswik Suite applications (CAD and Stope Optimiser). Four cut-off grades were analysed for underground mining to maximise Potential Mineral Inventory (PMI) and economic benefit.

Like the open pit optimisation, the stope optimiser indicated that there were three discrete underground mining centres. Considering the strike and geometry of mining shapes that were produced by the optimiser, it was decided that longitudinal sub level stoping was the most appropriate method of extraction. Following the completion of the initial design Deswik Pseudo Flow was used to remove any uneconomic stopes and subsequently the design was updated.

During the design iteration phase, the decision was made to incorporate a Paste Backfill Plant into the design. This inclusion assisted with reducing the life of mine waste requirements for the tailings dam, reducing the overall pit sizes. Paste backfill also introduces significant flexibility to the underground design and schedule and increases the volume of recoverable ore.

Once both designs and schedules (open pit and underground) were completed utilising Deswik Scheduler, the two schedules were combined. The combined schedule sees 7.1 Mt of Potential Mining Inventory (PMI) mined at an equivalent AuEq grade of 4.32g/t for approximately 988 koz contained AuEq.

The project involves three open pits (North, Central, and South), mined using conventional excavator and truck methods. Underground mining beneath each pit utilises the sub-level stopping method, with a 30 m crown pillar separating the open pit floor from the uppermost underground stope.

The mine plan targets an average monthly processing plant feed rate over of 87.75 kt ore per month (average of 1.05 Mtpa over 6.75 years), with slightly higher rates in the early production years when the facility is newly commissioned, and gradually reducing over the Life of Mine (LOM).

From initial findings, the project's total life is estimated to be approximately eight years, including one year of pre-production and seven years of production. Pre-production is preceded by six months that involve mobilising the open pit mining contractor for pioneering work and initial road construction.

During pre-production, PMI from the pit will be stockpiled, while waste will be directed to TSF for dam construction, supporting the initial two years of processing production. Underground development in the north and south zones begins in the pre-production year. The plant will receive PMI from high grade stockpiles, open pit, and underground sources in the first production year, shifting primarily to underground sources from the second year onward.

Open pit mining costs were benchmarked against South American operations, accounting for drill, blast, load, haul, auxiliary equipment, and mine owner's costs. Benchmark costs were adjusted to Q3 2023 prices. Additionally, a mining cost model was created, considering labour, fuel, and maintenance estimates. Ongoing discussions with mining contractors have been initiated based on a steady mining rate of 200 kt per month.

Challenger's mining consultants developed detailed cost models for underground operating expenses, categorising costs by period and activity. Development unit rates were calculated using a first principles cost models based on preliminary schedules by Mining Plus. Input costs, primarily from Argentine suppliers, were adjusted with industry productivity estimates. The initial three years are expected to utilise expatriate labour for supervision of underground mine operations, operation of jumbos and loaders, as well as maintenance supervision and support, until local operator training is completed.

1.8 Mineral Processing and Metallurgical Test work

Metallurgical testwork has been conducted at SGS Lakefield to evaluate the processing of mineralisation from the Hualilan project. Testing has been ongoing since 2021, however, the cut-off for inclusion in this study was October 2023.

SGS's scope was to undertake gravity, flotation, leaching and comminution testwork on composite samples representing the major mineralisation types from the project. Flowsheets for the beneficiation process for these different mineralisation types were developed in conjunction with the testwork program, with the flowsheets evolving as more results were received and evaluated.

The results demonstrated that it was possible to generate a high quality zinc concentrate grading 50.5% zinc, at a zinc recovery of 89.1%, based on sequential flotation testwork conducted on a high grade skarn composite. It was also possible to produce a high grade lead concentrate from this material grading 55.9% lead, at a lead recovery of 77.5%. Additionally, a precious metal concentrate was produced from a blend of both cleaner circuit tailings and gravity concentrate. Gold recovery to this concentrate was 63%, at a grade of 118 g/t gold. Combined recovery in the three concentrates was 83.1% (gold) and 75% (silver) with flotation tailings cyanide leached to recover 69.6% of the flotation residue gold and silver, increasing overall gold recovery to 94.9%.

Variability testwork conducted on the individual components of the high grade skarn composite, using the sequential flowsheet, showed that product grade is reduced with coarser primary grind and when there is increased pyrite content in feed. In the next phase of the project it is recommended that optimisation work is performed to ensure the milling circuit and the pyrite depression strategy is capable of dealing with the range of feed materials.

For feed material with relatively lower zinc grade it wasn't possible to produce a high grade zinc concentrate using the sequential flowsheet. However, a bulk flotation circuit was shown to be successful in generating a precious metal concentrate, with a gold grade of 46.8 g/t and at a gold recovery of 91.5% from a low grade dacite sample, and with a gold grade of 54.2 g/t at a recovery of 82.3% from skarn material.

Preliminary testwork using different reagents and different flowsheets achieved a high grade zinc concentrate (48.2% zinc) from a low head grade of only 0.36% zinc at another laboratory, however, follow up testwork is required on this option in the next phase of the project.

1.9 Plant Design and Operation

The design of the 1 Mtpa treatment plant selected for the Hualilan project allows for certain flotation stages and the Flotation Tails Leach (FTL) to be bypassed, thereby allowing the three different types of mineralised material to be processed separately on a campaign basis, using the same plant reconfigured for processing each material type. These different types of mineralised material were named:

- Type A (Au <1.5 g/t, Zn <1.5%);
- Type B (Au ≥1.5 g/t Au, Zn <1.5%); and,
- Type C (Au ≥1.5 g/t, Zn ≥1.5% and Au <1.5 g/t, Zn ≥1.5%).

For the Type A material, essentially Hualilan's 'low grade', the circuit involves crushing and milling, followed by gravity gold concentration and collection, and simple bulk sulphide flotation and the gravity recovered gold is blended with the sulphide concentrate to make a precious metal concentrate. The flotation tailings does not pass through the FTL due to low gold grades.

For the Type B material (low grade zinc with high grade gold component), the circuit is the same as for Type A material, except that that FTL is added to the end for the bulk flotation tails, and the gravity recovered gold is blended with the sulphide concentrate to make a precious metal concentrate.

For Type C material, which is a combination high grade zinc material with both low- and high grade gold, the circuit commences with gravity gold concentration and collection, followed by sequential copper and lead flotation, then zinc flotation, with the flotation tailings being subject to FTL, and ending with the gravity recovered gold being blended with cleaner tailings to generate a precious metal concentrate in addition to Zn and a Pb-Au concentrates.

Oxide material with economic gold content can also be treated by bypassing all flotation circuits to feed the gold leaching circuit directly from milling.

A lean, fit-for-purpose plant design standard was used to cover the required duty for the projected mine life. The design will accommodate nominal operation with a small degree of capacity flexibility for periodic variability but has no consideration or allowance for any capacity expansion.

The processing plant will consist of the following unit operations:

- Primary jaw crusher
- Crushed ore storage and reclaim
- Semi-Autogenous (SAG) mill
- Ball mill secondary grinding in a closed circuit with hydrocyclone classification
- Gravity treatment of hydrocyclone underflow using high G-force gravity concentrators for gold recovery
- Copper and lead flotation circuit, including rougher flotation and cleaner flotation
- Zinc flotation circuit, including rougher flotation, regrind and cleaner flotation

- Blending of gravity gold concentrate with cleaner tailings to produce a precious metal concentrate
- Concentrate dewatering (precious metal, zinc and lead concentrates) for export
- Flotation tailings thickening and leaching using a Carbon-in-Leach circuit for gold recovery (the flotation tailings leach is bypassed for lower gold grade material)
- FTL tailings are thickened and treated through a cyanide detoxification circuit, with detoxified tailings sent to the TSF
- Material not treated through the FTL circuit bypasses the detoxification circuit and is sent direct to the TSF
- Fresh and reclaim water supply
- Reagent preparation and distribution.

Approximately 50% of the tailings directed to the TSF will instead be re-routed to a paste plant, where the tailings will be mixed with cement and used as structural mine backfill material in the underground operations. The remainder will be thickened then disposed of in the TSF. Process water will be recycled as much as possible to minimise water usage.

1.10 Water sources and Supply

The Company has drilled three exploratory water bores to prove sufficient groundwater resources for the project. All three wells intersected ground water within 40 metres of surface over broad intervals in porous and permeable unconsolidated sediments. The water bores were drilled and flow testing completed as part of a successful hydrogeology study undertaken by the Company in 2023. This study indicated a maximum of 13 additional water bores, including monitoring stations, are required to supply the water needs for the project.

A local cost estimate was obtained for the costs of drilling these 13 additional bores for a combined 535 metres of drilling, their completion including submersible pumps and the cost of water pipelines to provide water to the project. This cost estimate of US$1.67 million (exclusive of VAT) did not include the cost of pumping from the bore field to the project or water storage reservoir. An additional allowance of US$0.3 million has been included for the construction of a reservoir and pumping.

1.11 Tailings Disposal

A desktop options study was completed to assess the advantages and disadvantages of all possible types of tailings storage, with the objective of assessing which style of construction minimised the overall risk and viability of the project. This desktop produced two possible TSF options which can reasonably be considered as viable. The side-hill storage was selected for the design. This utilises the natural topography and is constructed from a combination of mine waste and/or borrow from in-situ materials which totally surrounds and encapsulates the tailings within the HDPE-lined facility. The side-hill storage has robust embankments on the downstream slope and sides and is easier to construct given the need for HDPE liner placement.

The assumed project design parameters are as follows:

- Total tailings production – 7.07 Mtpa;
- Total tailings production discharged to the TSF – 4.2 Mt;
- Total tailings production to paste backfill – 2.87 Mt;
- TSF Life – 7 years;
- Project Life – 8 years;
- Slurry density – 50% solids;
- Deposited in-situ of the tailings – 1.5 t/m^3 (assumed based on the TSF being well-managed, with very high-water recovery); and
- Supernatant water recovery target (min) – 70%.

Materials sourced from mine waste from the open pit, from the development of the underground mine and external borrow areas, will be utilised for construction of the perimeter embankments. The TSF is double HDPE-lined with an underdrainage connected to an external sump and a leak detection system between the liners on the floor and Stage 1 upstream embankment.

Tailings will be deposited using sub-aerial deposition techniques from multi-spigot locations on the perimeter deposition embankments. Tailings spigotting or deposition is to be executed in thin layers of not more than 300 mm, to ensure a uniform tailings beach with a fall of 1% towards the decant is developed. The spigotting sequence is to be formulated such that the supernatant water pond is always maintained around a decant structure.

A decant is to be installed in the centre of the TSF to facilitate surface water recovery. The decant system comprises a rock ring filter. The decant water recovery system must be designed for not less than minimum water recovery of 70% of the slurry water to achieve the minimum design in-situ dry density of 1.5 t/m3.

Whilst the TSF is designed as a "no-discharge facility", under standard operating conditions in accordance with ANCOLD (2019) Guidelines2, the final stage of the design will need to consider an emergency spillway, which provides a controlled discharge point following an extreme rainfall event which exceeds the available storage capacity within the TSF.

1.12 Infrastructure and Services

Notwithstanding the existing infrastructure, which includes a portable 100 bed camp and offices, the Hualilan project is a greenfield site that requires significant infrastructure to support the mine operations. The infrastructure required includes:

- Access roads
- Water supply
- Energy and power supply
- Accommodation village
- Process plant site buildings for administration, workshops, and warehouses
- Mining area buildings and other infrastructure to support mining operations
- Washdown facilities
- Fuel supply, storage, and distribution
- Communications and information technology
- Fire protection
- Surface water management infrastructure
- Logistics support for the project operations.

The proposed site layout is shown in Figure 1-1.

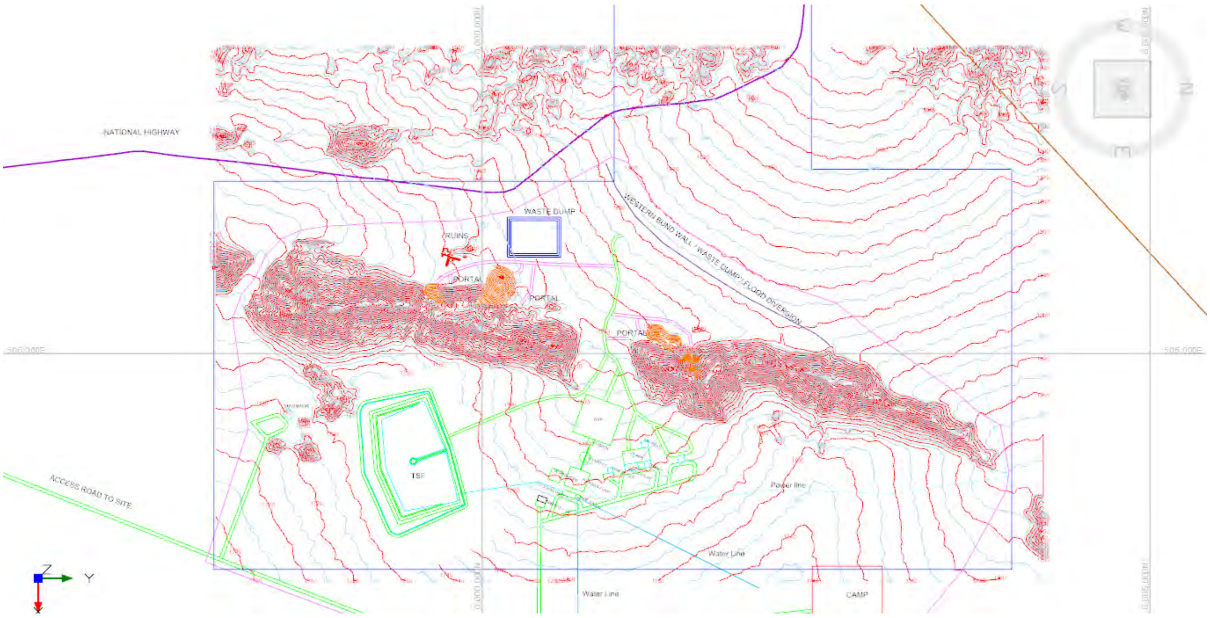

Figure 1-1: Proposed Site Layout

1.13 Environmental Impact and Management

Environmental baseline monitoring has been ongoing at the project since March 2021. This monitoring includes:

- Air Quality

- Noise and Vibration levels
- Flora
- Fauna
- Climate.

Additionally, specific independent consultant reports have been prepared and lodged with the San Juan Department of Mines including:

- Archaeological report
- Report on Palaeontology.

Neither report indicated any potential areas of concern, with the historic buildings on site at Hualilan deemed to have no archaeological significance due to several periods of modification.

The flora survey program indicated that no irreversible impacts are expected on ecological processes in the project footprint and areas surroundings. No mitigation measures are planned for the flora resource given that, in the Hualilan project, there is no impact, either directly or indirectly, on sensitive vegetation units. Additionally, it should be noted that in the footprint of the project and water study area there are no high plains or wetlands which are subject to an additional set of regulations. Similarly, the Fauna monitoring program indicated that no issues regarding Red List of Mammals of Argentina or Near Threatened species.

The Hualilan project is distant from any Natural Protected Areas (NPA) of the Province of San Juan or any NPA of national, provincial and/or municipal jurisdiction.

1.14 Social and Community

The Hualilan project is located in the Ullum Department in the northwest of the Capital city of San Juan. The Ullum department has an area of 4,391 km^2 and its population is 6,463 according to the last Census carried out in the year 2022. The nearest town to the project is Iglesia located with a population of 366 located 44 kilometres from the project on national route 149.

The acquisition of the 20,000 Ha of land containing the project simplifies the stakeholders for the project. The primary stakeholders apart from the Company and its employees are inhabitants of the Ullum Department and the province of San Juan.

Challenger has developed a community relations program, which began to be implemented during the exploration stage of the Hualilan project and will continue throughout the useful life of the mine. The main objective of this program is to generate two-way communication channels between the company and the community, particularly from the Ullum Department, for the purposes of:

- Informing the community about the environmental impacts of the project in the stages of construction, operation and closure.
- Prevention and mitigation measures for those impacts.
- The opportunities generated by the new activity to be located in the area.
- Managing community expectations and concerns.

The social license is of particular importance in the development of a mining operation, so the key to maintaining the existing social licence will be adherence to sound social practices.

Challenger currently supports the community in several ways including:

- 2000 families from the Ullum community benefited from a drinking water well, which represents more than USD$30,000 of investment.
- Geology students trained via our local internship program with San Juan Universities with some assisted via our scholarship program with the San Juan School of Mines.
- New local businesses supported via CEL's local supplier development plan in Argentina.
- Locals employed with a 95% retention rate.
- Special needs students were sponsored to participate in the Ullum science fair in Argentina.
- Children benefited from educational programs in Argentina.
- The San Juan Mining Ministry confirms that Challenger Exploration and Barrick Gold are the only exploration companies operating in San Juan that meet the government's +80% local employment quota.

1.15 Marketing and Sales

Reach Partners, a third-party concentrate marketing specialist company based in London undertook concentrate market study on behalf of the Company. The Company has expanded this work done by Reach with discussions with concentrate off-takers, other concentrate marketing groups and concentrate traders. As part of these expanded discussions additional indicative concentrate off-take terms were provided by some off-takers.

The Hualilan project will produce three types of concentrate:

- Gold-Silver concentrate (approximately 60 g/t Au, 250 g/t Ag)
- Lead-gold-silver concentrate (approx. 50% Pb, 150 g/t Au, 700 g/t Ag)
- Zinc Concentrate with gold and silver credits (approx. 50% Zn, 10 g/t Au, 180 g/t Ag)

Additionally, Hualilan will produce gold and silver as dore from the Flotation Tails Leach (FTL).

All concentrates are excellent quality with only the Zn concentrate likely to attract any penalty charges due to Fe content of 11.5% being above the penalty limit. For each 1.0% by which the final Fe assay exceeds 8.0%, seller pays a penalty charge of $2.0 per DMT of concentrate. Thus, the penalty charge payable on the Zn concentrate is ~ US$7/dmt.

A summary of the modelled concentrate off-take terms is provided in Table 1-1.

Table 1-1: Summary of Concentrate Off-take Terms

Product	Payability (%)	TC/RC (US$/t)	Penalties (US$/t)
Zn-Au-Ag concentrate			
Zinc	84	160	7
Gold	63		
Silver	35		
Pb-Au-Ag concentrate			
Lead	95	125	nil
Gold	95		
Silver	95		
Au-Ag concentrate	92.9		
Gold	95	100	nil
Silver	60		

Concentrate transportation contracts will be negotiated and finalised during the BFS phase of the project. For the purposes of this Scoping Study, it is assumed that all concentrates will be loaded in bags and placed in shipping containers at site for transport.

Total freight costs including overland freight by truck, rail freight, port costs, ocean freight, insurance, and assay fees are estimated to be US$150/wmt. These costs were provided by a concentrate logistics study prepared by Steinweg with the Company's local import/export agent, located in Buenos Aires, also provided shipping quotes from third party shipping groups. Payability after transport/ insurance and TC/RC's and penalties are shown in in Table 1-2.

Table 1-2: Summary of Average Metal Payability after Transport, TC/RC, Penalties

Concentrate Payability after TC/RC and Transport Costs	Average Payability[1] (%)
Gold	88.4%
Silver	54.8%
Zinc	73.1%
Lead	93.6%
Concentrate Transport (site to smelter including insurance)	US$150/wmt

[1] Payability after Transport Cost, Refining Cost (TC/RC) / Penalties

1.16 Capital Cost Estimates

The capital cost estimate was prepared by Mining Plus and a number of independent external consultants retained by CEL.

There was limited use of benchmarking, with costs generally sourced from vendor quotes/ indicative prices, or detailed first principle cost analysis using vendor quotes based on the preliminary project design. Where benchmarking was used to provide any capital costs the primary source was the Mining Plus internal cost database, augmented by Challenger's consultants' databases. Where benchmarking has been used to provide capital cost estimates this has been specifically stated in this Scoping Study report.

The cost estimate is expressed in Q3 2023 US$ and used the USD/ARS exchange rate at the time the quotation was provided (average 200 ARS/USD) for any in-country costs provided in ARS. In practice in Argentina most cost quotes are generally provided in USD and converted into ARS based on the prevailing USD/ARS rate. The costs do not include allowances for escalation or exchange rate fluctuations. All costs are exclusive of the Argentinian value added tax (VAT), which is applied separately in the financial model used for economic evaluation.

The capital cost estimate for this scoping study has a target accuracy range of ±15% where costs have been sourced from vendor quotes or first principles analysis. The costs developed by benchmarking have a target accuracy of ±35%.

The following areas were included in the Pre-Production Capital Cost estimate:

1. Open Pit Mine (open pit mine development, equipment fleet, pre-stripping/ pioneering and supporting infrastructure and services)
2. Underground Mine (underground development, equipment fleet, paste backfill plant and supporting infrastructure and services)
3. Process plant (gold-silver, zinc-gold-silver, and lead–gold-silver concentrates), conventional 1-1.2 Mtpa concentrator and Flotation Tails Leach circuit with supporting plant infrastructure and services
4. TSF
5. On-site infrastructure (earthworks, sitework, roads, water treatment and distribution, camp and other general facilities)
6. Off-site infrastructure
7. Owners Costs including EPCM, spares, first fills, transport costs and import costs;
8. Indirect costs
9. Other Pre-production Costs (other operating costs prior to commercial production/ processing)
10. Contingency (applied at +15%).

Total capital costs are estimated at US$133.7 million, not including US$18.4 million of capitalised mining costs. Total Pre-development capital costs of US$152.1 million are summarised in Table 1-3. More complete details of pre-development capital costs are provided in section 22.

Table 1-3: Summary Capital Cost Estimate

Description	Pre-production Capital Costs US$	Sustaining Capital Cost US$	Total Capital Cost US$
1. Open Pit Development (inc. Truck Shop, Wash Bay, Tyre Bay)	5.8		5.8
2. Underground Development (inc. paste plant)	21.8	45.0	66.8
3. Process Plant	59.0	9.0	68.0
4. TSF	5.4	3.0	8.4
5. On-site Infrastructure	8.7	1.5	10.2
6. Off-site infrastructure	0.0	0.0	0.0
7. Owners Costs	15.6		15.6
8. Indirect Costs	2.7		2.7
9. Contingency	14.7	0.5	15.2
Total Capital Expenditure	**133.7**	**59.0**	**192.7**
10. Other Pre-production Costs[3]	18.4		18.4
Total Pre-Production Capital	**152.1**	**59.0**	**211.1**

1. All figures are rounded to reflect the relative accuracy of the estimate.
2. Totals may not sum due to rounding as required by reporting guidelines.
3. Pre-production costs are operating costs that occur prior to the mill operating.

1.17 Operating Cost Estimates

The operating cost estimates in the Study are based on: contractor operated truck and excavator open pit mining; owner operated underground mining via longitudinal SLOS with paste backfill; processing which includes gravity recovery, conventional flotation and Floatation Tail Leach (FTL); and deposition of the tails not consumed in the paste backfill process in a Tailing Storage Facility (TSF).

Operating cost estimates have generally been derived from first principles costs analysis prepared by external consultants, rather than by benchmarking. These cost estimates include local labour rates derived from San Juan industry standards, costs sourced by vendor/supplier quotations both in Argentina and externally, and productivity rates that reflect the local workforce and conditions. Unless otherwise stated in this scoping study report the operating cost estimates have an expected accuracy range of ±15%.

The operating estimate is expressed in Q3 US$ and used USD/ARS exchange rate at the time the quotation was provided for any in country costs provided in ARS. In practice, in Argentina, most quotes are generally provided in USD and converted into ARS based on the prevailing USD/ARS. This includes diesel, equipment hire for both general and specialised mining equipment, reagents and consumables. The exceptions are Government provided services such as grid power and in-country labour. Generally, the rate of increase in the ARS price tracks the decline in the ARS/USD rate for power and labour, however there is a 1-3 month lag in the repricing of ARS denominated costs. For any ARS denominated input costs, such as grid power and labour, Challenger has used the 2022H2/2023H1 prices, as converting the current local ARS costs into a USD denominated price at the prevailing ARS rate would artificially lower these input costs on a USD basis.

Table 1-4 provides a breakdown of the key operating cost assumptions used in the Study.

Table 1-4: Summary of Key Operating Cost Estimates

Unit Operating Costs	Unit	Unit Cost
Open pit Mining (ore/waste)	**US$/t mined**	**3.00**
Underground Mining	**US$/t mined**	**34.74**
Underground Development		
Inclined Development (5 m x 5 m)	US$/m	2,828
Horizontal development (5 m x 5 m)	US$/m	2,828
Vertical Development	US$/m	2,333
Slot Rises (included in underground mining cost)	US$/m	1,500
Underground Development	**US$/t mined**	**28.29**
Total Underground Mining and Development	**US$/t mined**	**63.03**
Processing (Type C ≥1.5% Zn) 34.0% total PMI	**US$/t processed**	**16.31**
Processing (Type B ≥1.5 g/t Au, <1.5% Zn) 59.1% total PMI	**US$/t processed**	**12.12**
Processing (Type A <1.5 g/t Au) 6.9% total PMI	**US$/t processed**	**9.26**
G&A	**US$/t processed**	**5.38**

1.18 Economic Analysis

The summary results of the Economic Analysis of the Hualilan project are summarised in Table 1-5. The results of the analysis show the Hualilan Gold project to be economically robust. The net present value of the net cashflow with a 5% discount rate (NPV5) is US$409.30 million on a pre-tax project basis, and US$294.78 million post-tax, using a base gold price of US$1,750/oz. Project internal rates of return (IRR) are 75.23% pre-tax and 66.04% post-tax.

The project payback period (i.e. process plant start-up until all initial expenditures are recovered) at a gold price of US$1,750/oz is expected to be 1 year 3 months from production start date.

Like most gold mining projects, the key economic indicators (NPV and IRR) are most sensitive to changes in gold price. As such, a 10% decrease in gold price would reduce the post-tax NPV5 to US$315.30 million pre-tax, US$234.17 million post tax and the IRR to 59.50% pre-tax and 53.59% post tax.

The sensitivity analysis demonstrates the project would require a grade reduction or gold price reduction in the order of 44% to reach the breakeven gold price of US$953/oz from a NPV perspective.

Table 1-5: Summary of Key Financial Metrics

Commodity Price Assumptions	**Unit**
Gold Price	US$1750/oz
Silver Price	US$20/oz
Zinc Price	US$1.15/lb
Lead Price	US$0.94/lb
Key LOM Financial Metric	**US$**
Revenue (LOM)	$1,465 million
EBITDA (LOM)	$738 million
C1 Cost (Real – US$/oz)	$527/oz
ASIC (real – US$/oz)	$830/oz
Free cashflow (Pre-tax) LOM	$527 million
Free cashflow (Average per annum)	$78 million
Pre Tax NPV5	$409 million
Post tax NPV5	$295 million
Payback Period (Pre-Tax)	1.25 years
Payback Period (Post Tax)	1.25 years
Project IRR (Pre-Tax Real)	75.2%
Project IRR (Post Tax Real)	66.0%

1.19 Opportunities

The key opportunities for improvement are listed below.

- A low grade zinc concentration pathway, based on a recent flotation test on a composite grading 0.36% Zn which produced a saleable Zn concentrate grading 48%

Zn. Based on prior flotation test work, an assumption was used in the Study that an economic zinc concentrate was only achievable from at a grade >1.5% Zn. The MRE contains approximately 267,000 t of zinc of which only 70,000 t is recovered in the scoping study mine plan. The ability to economically recover part of the additional 197,000 t of zinc in the MRE could significantly enhance economics, given the recovered portion of the ~70,000 t of zinc generates US$132 M revenue based on the study forecasts.

- Further improvement to the underground stope optimisation, development sequence and production scheduling. The underground stope optimisation was undertaken using an assumption of US$1700/oz gold. Additionally, some improvements in production and development unit costs in the order of 10-20% have already been identified in the intervening period since running the stope optimisation. These improvements in production and development costs are yet to be incorporated into the optimisation, and are likely to result in additional stopes being included in the mine plan. Additionally, optimisation included a Pseudoflow analysis on the underground design to remove uneconomic areas that sit above the stope cut-off grade. Pseudoflow removed 832 kt containing 72,000 oz AuEq from the underground mine plan that may be profitable at current spot prices and revised operating costs.
- The improvement in underground optimisation includes reviewing the staging of development during the pre-production period to optimise CAPEX while trading off against ensuring access to the highest value stopes in early phases of the UG mine.
- Recovery of the 30 metre crown pillar design which has been left between the base of the open pits and the underground workings. This crown pillar design contains approximately 15,000 oz AuEq. The study currently assumes no recovery of this crown pillar, however additional geotechnical information may support the recovery of this crown pillar.
- Inclusion of a heap leaching option which provides a process path for a significant proportion (~50%) of the MRE that was excluded in the high grade/low-tonnage SS production model. Preliminary column testing on a low grade composite yielded promising results. As a result of this, a panel of column tests were initiated to test the three material types separately at a range of different head grades. Results from this current panel of column tests are in progress and a positive outcome has potential to add significant value to the project.
- Reduction in open pit mining unit cost through owner-operator and bulk mining efficiencies. A unit cost of US$3.00/t was assumed for the study, initially as a conservative estimate based on the predicted reduced scale of the open pit operation, and later to account for contractor premiums. However, preliminary first-principles cost modelling by the Company, and discussions with equipment vendors around collaboration and operating partnerships, indicates that an owner operated unit cost around US$2.00/t may be achievable at scale. This impact of a reduced mining unit

cost is even more pronounced in a high-volume mining scenario that incorporates a low grade heap leach. This cost estimate is supported by localised benchmarking at other owner-operator open pit mines in Argentina.

- Potential processing of the Au-Ag concentrate on site to produce gold and silver dore. The project is forecast to produce 412 kt of Au-Ag concentrate containing 634 koz Au and 1.9 Moz Ag over the Life of Mine. The treatment of this concentrate on site to produce gold and silver dore rather than its transport and sale to off-takers as a concentrate could result in combined cost savings and additional revenue net to the project of over $165 million based on the SS production forecast.

1.20 Conclusions and Next Steps

The study provides justification that the Hualilan Project is commercially viable and accordingly the Board of the Company has approved progression of the project to the next stage of development.

Next steps to add to the robustness of the current project and provide a pathway for future development for the project are listed below. This work will be done in parallel with ongoing exploration and studies to evaluate the viability of a staged large open pit development.

- Taking receipt of the final results for the suite of Column Leach tests currently underway, which will allow for an assessment of the viability of Heap Leach as a potential processing pathway for the low grade mineralisation.
- Completion of additional flotation testing on the potential low grade zinc concentration pathway.
- Completion of additional flotation testing, including locked-cycle and variability test work, which will be required to provide sufficient data for the PFS.
- Testing to determine the liberation of the gold and silver in Au-Ag concentrate and evaluate options to produce dore on site.
- Development of a detailed first-principles open pit mining cost model, in collaboration with equipment vendors, to evaluate the potential owner operated bulk mining efficiencies.
- Completion of a suite of CIL test work (with dual-laboratory verification) to allow Au and Ag recoveries and NACN consumption to be modelled for both the high grade and low grade mineralisation, thereby allowing for a definitive evaluation of the CIL processing option.
- Update the first-principle cost models for the processing and general and administrative areas such that they can be utilised to assess the cost impact of variable process throughputs.
- Update the processing cost model to be inclusive of heap leaching, should the Column Test results be positive.

- Complete geotechnical data gathering, including: additional core logging; collection of Point Load Test data from existing drill core; gathering of televiewer data from existing drill holes; and, any drilling of additional geotechnical test holes.
- Updating the underground stope optimisation for final underground mining and development cost forecasts.
- Further optimisation of the open pit/ underground interface and which components of the orebody should be included in each.
- Additional drilling of some of the drill targets identified in the Hualilan regional exploration programme.

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2 INTRODUCTION

2.1 Purpose of the Technical Report

Challenger Gold Limited (ASX:CEL) is a gold, silver, zinc, copper and molybdenum exploration and development company with projects in San Juan Province, Argentina (Hualilan) and El Oro Province, Ecuador (El Guayabo).

The flagship Hualilan Project is a zinc-enriched skarn with a late-stage gold overprint. A Mineral Resource Estimate (MRE) dated 29 March 2023 included 60.6 Mt at 1.1 g/t Au, 6.0 g/t Ag, 0.44% Zn and 0.06% Pb containing 2.8 Moz AuEq at an average AuEq grade of 1.4 g/t, and has a high grade core of 1.6 Moz AuEq at 5.0 g/t AuEq. The MRE forms the basis of this scoping study.

This scoping study was undertaken to determine the viability of a development of Challenger Gold Limited's (CEL) Hualilan Gold Project, and confirm the business case to progress more definitive studies on the project as the next step towards production. The study examined a range of mining and processing options for Hualilan, and includes a preliminary economic assessment which was conducted to identify the highest-return pathway for the project as it moves towards more detailed studies. It is an order of magnitude (± 35%) technical and economic study of the potential viability of the MRE, and the options identified in this study will be explored and optimised further in later study phases.

Gold equivalent (AuEq) values used in the Study were taken from the 2023 MRE. This gold equivalent calculation is:

AuEq = Au_ppm + (Ag_ppm*0.01209916) + (Zn_%*0.46224835) + (Pb_%*0.19938772)

This gold equivalent was revised for underground MSO (Mine Stope Optimiser) work to the following:

Type A: (Au<1.5 g/t Zn < 1.5% Zn) **AuEq = Au_ppm + (Ag_ppm* 0.009922041)**

Type B: (Au>1.5 g/t; Zn < 1.5% Zn) **AuEq = Au_ppm + (Ag_ppm* 0.009922041)**

Type C: (Au<1.5 g/t; Zn > 1.5% Zn) **AuEq = Au_ppm + (Ag_ppm* 0.009922041) + (Zn % * 0.429482636)**

CEL confirms that it is the Company's opinion that all the elements included in the metal equivalents calculation have reasonable potential to be recovered and sold.

2.2 Project Background

The Hualilan project is in the Department of Ullum, approximately 120 km from the city of San Juan. Access to the deposit from San Juan is by sealed road on National Route 40 from San Juan to Talacasto Station (52 km), then by Provincial Route 436 from Talacasto Station to

National Route 149 junction (23 km), then by National Route 149 (45 km). The sealed road extends to within 1 kilometre of the deposit.

The deposit is in the Central Precordillera, which is part of the La Rioja - San Juan - Mendoza Precordillera. Mineralisation was discovered in 1751 and was worked on a small scale from 1790 to the early 1960's, predominantly from oxidised near surface high grade 'manto-style' deposits in the San Juan Limestone. Mining was predominantly undertaken through narrow underground workings which are partially surveyed where access is possible. There is no reliable past production data, however there is approximately 20 kt of tailings and re-treated tailings evident at surface which indicates that low tonnage, but high grade mineralised material was recovered. Gold and silver recovery was by various processes including washing, roasting and cyanidation.

Exploration recommenced periodically from 1984 to 2005. Various explorers completed mapping underground and at surface, stream sediment, rock chip and channel sampling, development of a 300-metre exploration decline, geophysical surveys, RC drilling, and diamond core drilling. Data or partial data has been recovered for 156 drill holes (total of 17,283 metres) completed during this period. 75 drill holes (total of 8,030 metres) were found to have data of sufficient reliability to use in the most recent MRE. No resource estimate consistent with JORC reporting standard was completed during this time.

From 2018 until the date of the MRE, CEL, via Golden Mining S.A. (100% owned by CEL) has completed ground magnetic and Induced Polarisation (IP) geophysical surveys, additional mapping, underground and surface channel sampling (1,767 samples averaging 1.5 metres length), 37 RC drill holes (2,923 metres) and 762 diamond core drill holes (224,180 metres). These data and the reliable historic data have been used to estimate the current Resource.

The current MRE strikes 2.5 km and is contained with 31 domains that are grouped into 3 mineralisation styles corresponding to the host rock and structural controls of the mineralisation. Estimation was by Ordinary Kriging of 2 metre composites within the hard boundaries of the mineralised domains. The MRE is approximately 75% Indicated and 25% Inferred. The entire resource, including the Inferred material, has been used as the basis for this study.

This scoping study presents a combined open pit and underground operation, delivering an average throughput of 1.05 Mtpa for processing and recovery of gold, silver and zinc by gravity, sequential flotation and cyanidation.

Power is supplied via a renewable energy offset agreement. Initial capital, sustaining capital and mine operating costs have been estimate from first principles and/ or benchmarked from similar projects in South America. Waste from open pit and underground will be utilised in the construction of the TSF embankment and in initial construction activities, and where there is additional waste it will be stacked in conventional terraced waste dumps. Water will be recycled for processing at every stage where possible, while a nearby bore field will produce approximately 30.4 litres/second for processing, mine dust suppression, and camp usage.

2.3 Competent Persons and Other Participants

The following Qualified Persons (QPs) are responsible for the information provided in the indicated sections in Table 2-1, as defined in National Instrument 43-101 – Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1 Technical Report.

Table 2-1: Report Responsibility

Company	Qualified Person	Position	Report Section Responsibility
Mining Plus	Erick Ponce	Surface Area Manager	Sections 2, 3, 4, 5, 13, 14, 15, 18, 19, 20, 21, 22 (partial), 23 (partial), 24, 25, and corresponding subsections of 1 and 27.
Mining Plus	Shaida Miranda	Senior Mining Consultant in Surface	Section 24 (partial) and corresponding subsections of 1 and 27.
Challenger Gold	Stuart Munroe	Manager - Exploration	Sections 6, 7, 8, 9, 10, 12, 25 and corresponding subsections of 1 and 27.
	Jeremy Ison	Contract Metallurgist	Sections 11, 16, 22 (partial), 23 (partial) and corresponding subsections of 1 and 27.
Land & Marine Geological Services Pty Ltd	Chris Lane	Director and Principal Consultant	Section 17 and corresponding subsections of 1 and 27.

2.4 Effective Dates

The report has the following effective dates:

- Mineral Resources Estimation: 29 March, 2023
- Financial analysis that supports the Scoping Study:
- The effective date of this report matches with financial analysis completed on October 30th 2023.

2.5 Frequently Used Acronyms, Abbreviations, Definitions, and Units of Measure

Table 2-2: Technical terms and abbreviation

Abbreviation	Description
Ag	Silver
AISC	All in Sustaining Costs
As	Arsenic
Au	Gold
BM	Block Model
CAPEX	Capital Expenditure
Cu	Copper
DDH	Diamond Drill Hole

Abbreviation	Description
EIA	Environmental Impact Assessment
ft	Foot
g	Gram
ha	Hectare
Hg	Mercury
hr	Hour
IRR	Internal Rate of Return
kg	Kilogram
km	Kilometre
koz	Thousands of ounces
ktpd	kilotonnes per day
l	Litres
lb	Pound
LOM	Life of mine
m	Metre
m^2	Square metre
m^3	Cubic metre
Ma	Mega-annum (1,000,000 years)
masl	Metres Above Sea Level
Max	Maximum Value
Mg	Magnesium
Min	Minimum Value
mm	Millimetre
Moz	Million ounces
Mt	Million tonnes
Mtpa	Million tonnes per annum
NPV	Net Present Value
NSR	Net Smelter Return
OK	Ordinary Kriging
OPEX	Operational Expenditure
oz	Ounce
Pb	Lead
PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PLS	Pregnant Leach Solution
ppb	Parts per billion
ppm	Parts per million
PMI	Potential Mining Inventory
QA	Quality Assurance
QC	Quality Control
QP	Quality Person
RF	Revenue Factor
ROM	Run of Mine
RQD	Rock-Quality Designation
t	Tonne
tpd	Tonnes per day
TSF	Tailings Storage Facility

Abbreviation	Description
US$	United States Dollars
wt%	Weight percentage
yr	Year
Zn	Zinc

3 RELIANCE ON OTHER EXPERTS

The authors of this Technical Report relied upon the following other expert reports which provided information regarding mineral rights, surface rights, royalties and taxation, property agreements, environmental liabilities, the mineral resource, metallurgical testing and process route, geotechnical, hydrogeology and hydrology, tailings requirement and management, and market conditions and contracts.

The authors have taken appropriate steps in their professional judgment to confirm that all information outlined above as having been supplied by non-Qualified Persons to prepare this Technical Report is reliable, but it must be recognized that the authors are relying on the accuracy of the above.

3.1 Land Tenure

The QPs have not reviewed the mineral tenure, surface rights, property ownership, nor independently verified the legal status of the project area, underlying property agreements, or permits. The QPs have fully relied upon, and disclaim responsibility for information derived from experts retained by Challenger through the following document:

- BASTIAS YACANTE ABOGADOS, Lawyers 2022: Legal Opinion of Legal Opinion regarding Challenger Exploration Ltd.´s subsidiaries in Argentina and mining concessions included in the core of the Hualilan project, located in San Juan Argentina

Prepared for Challenger Gold, 5 September 2022. This information is used in Section 4 of the Report.

3.2 Environmental Liabilities and Permits Required

The QPs have not reviewed the Environmental Liabilities and permits required. The QPs have fully relied upon, and disclaim responsibility for, information derived from experts retained by Challenger through the following documents:

- Consultores Mineros de San Juan: Hualilan Social and Economic Impacts

Prepared for Challenger Gold, 1 August, 2021, 24 pages. This information is used in Section 20 of the Report.

- Consultores Mineros de San Juan: Environmental Studies, Permitting and Social or Community Impact

Prepared for Challenger Gold, 1 August, 2021, 24 pages. This information is used in Section 19 of the Report.

3.3 Royalties, Depreciation and Taxes

The QPs have not reviewed the Argentinian mineral taxation requirements or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from experts retained by Challenger through the following document:

- Price Waterhouse Coopers (Argentina): SUMMARY OF KEY TAX GUIDELINES IN ARGENTINA FOR THE MINING ACTIVITY

Prepared for Challenger Gold Resources, 16 April 2023, 24 pages. This information is used in Section 4 of the Report. It is also used in support of the Chapter 23, Economic Analysis. A copy of this report has been included as Appendix 1 – Price Waterhouse Coopers Report on Mining Tax.

3.4 Other Technical Reports and Experts Relied Upon

Other technical Reports and experts relied upon by the QP's are listed below. The authors have taken appropriate steps in their professional judgment to confirm that all information outlined below as having been supplied by non-Qualified Persons to prepare this Technical Report is reliable, but it must be recognised that the authors are relying on the accuracy of the below.

- GEOWiZ Consulting: Hualilan Project Mineral Resource Estimate
  Prepared for Challenger Gold, 29 March 2023 This information is used in Section 12 of the Report.

- IG GMSA GEOTECHNICAL: SLOPE STABILITY STUDY Conceptual Engineering Stage
  Prepared for Challenger Gold, 1 April 2023 This information is used in Section 13 of the Report.

- Consultores Mineros de San Juan: Hualilan Project Hydrology and Hydrogeology
  Prepared for Challenger Gold, 29 March 2023 This information is used in Section 14 of the Report.

- Instituto de Investigaciones Mineras: PROYECTO HUALILAN, SAN JUAN. GEOLOGIA – ESTRATIGRAFIA -PALEONTOLOGIA
  Prepared for Challenger Gold, 29 November 2022 This information is used in Section 19 of the Report.

- Grupo Induser S.R.L.: GROUNDWATER MONITORING (WELL WATER)
  Prepared for Challenger Gold, February 2023 This information is used in Section 14 of the Report.

- INGENIERIA SISMICA Y GEOTECNICA S.R.L: SEISMIC THREAT ANALYSIS FOR THE PROJECT HUALILAN PROJECT (Province of San Juan, Argentina)

Prepared for Challenger Gold, March 2023 This information is used in Section 13 of the Report.

- Dr. Laura P. Perucca: HUALILAN PROJECT NEOTECTONIC REPORT
  Prepared for Challenger Gold, December 2022 This information is used in Section 13 of the Report.

- Dr Catalina Teresa Carlos: Archaeological Monitoring of Historical-Cultural Sites of the HUALILAN Influence Area
  Prepared for Challenger Gold, December 2022 This information is used in Section 19 of the Report.

4 PROPERTY, DESCRIPTION AND LOCATION

4.1 Location

The Hualilan property is situated in northwestern Argentina in the province of San Juan. It is located within the Department of Ullum, at a distance of approximately 120 km from the provincial capital, San Juan city. The province has a population of approximately 800 thousand and contains the satellite offices of many of the leading mining service providers.

The deposit is located in the Hualilan Hills, at the NW end of the Gualilán Basin, in the area of the Central Precordillera, which is part of the morpho-structural unit of the La Rioja, San Juan and Mendoza Precordillera. Figure 4-1 below shows the general location of the project.

The mining concessions are part of Mining District No. 7 of the mining cadastre of the Ministry of Mining. Its centre is located at the intersection of the coordinates GK – POSGAR X = 2,504,610; Y = 6,603,140

The area is included in the topographic sheets of the National Geographic Institute (ex IGM) No. 3169-10 “Niquivil”, 3169-15 “Sierra de la Crucecita” and 3169-16 “Sierra de Talacasto”, all at scale 1: 100,000.

Figure 4-1: Location of the Hualilan Project.

4.2 Land Tenure

Mineral rights in Argentina are separate from landowners and are owned and administered by the provincial governments. The following summarises some of the relevant provisions of the Federal Argentina Mining Code (AMC), and Argentina mining law terminology, to aid in understanding the land holdings in Argentina.

The provinces are the owners of the natural resources located within their territories, and each province retains the power to administer and regulate mineral rights according to the AMC and supplemental provincial laws and regulations.

The AMC establishes that mining is in the public interest and therefore landowners cannot prevent the granting of mining rights or commencement and continuity of mining activities on their property. However, landowners have a right to collect an indemnity due to the use of the land by the miner and the consequences arising from mining activities. Land over which a mining concession has been granted is legally subject to different types of easements provided that an indemnity is paid to the owner of the land.

Mineral rights are considered forms of real property and can be sold, leased, or assigned to third parties on a commercial basis. Cateos (exploration permits) and minas (mining concessions) can be forfeited if minimum work requirements are not performed, or if annual payments are not made.

Grants of mining rights, including water rights, are subject to the rights of prior users. Furthermore, the mining code contains environmental and safety provisions administered by the provinces.

Prior to conducting operations, applicants must submit an Environmental Impact Assessment (EIA) to the provincial mining authority describing the proposed operation and the methods to be used, and how the operator will prevent undue environmental damage. When the provincial mining authority approves the EIA it issues a permit in the form of an official declaration (DIA).

The EIA must be updated every two years, with a report on the results of the protection measures taken. If protection measures are deemed inadequate, additional environmental protection may be required. Mine operators are liable for environmental damage. Violations of environmental standards may cause exploration or mining operations to be shut down, but without prejudice to mining title.

Exploration permits do not allow commercial mining but give the owner a preferential right to obtain a mining concession for the property. Exploration permits have finite terms depending on their area, and the permit holder is required to reduce its holding as time progresses. During exploration, land can be converted to one or more Manifestaciones de

Descubrimiento (MD). Time extensions may be granted to allow for bad weather and difficult or seasonally restricted access.

A fee of ARS$400 per unit must be paid upon application for the exploration permit. This is paid only once. In addition, the tax act for the province of San Juan requires a fee to be paid upon application for exploration. The actual value is ARS$1,600 for each unit of 500 ha. This fee is only paid one time.

To convert an exploration permit to a mining concession, some or all the area must be declared as MD and then converted to a mining concession, which permit mining on a commercial basis. Once granted, the mines have an indefinite term, as long as exploration, development or mining activities are ongoing and the investment conditions according to the AMC are met. An annual canon fee of ARS$3,200 per tenement is payable to the province.

Challenger Exploration holds a variety of mineral tenements and surface rights as shown in Figure 4-2.

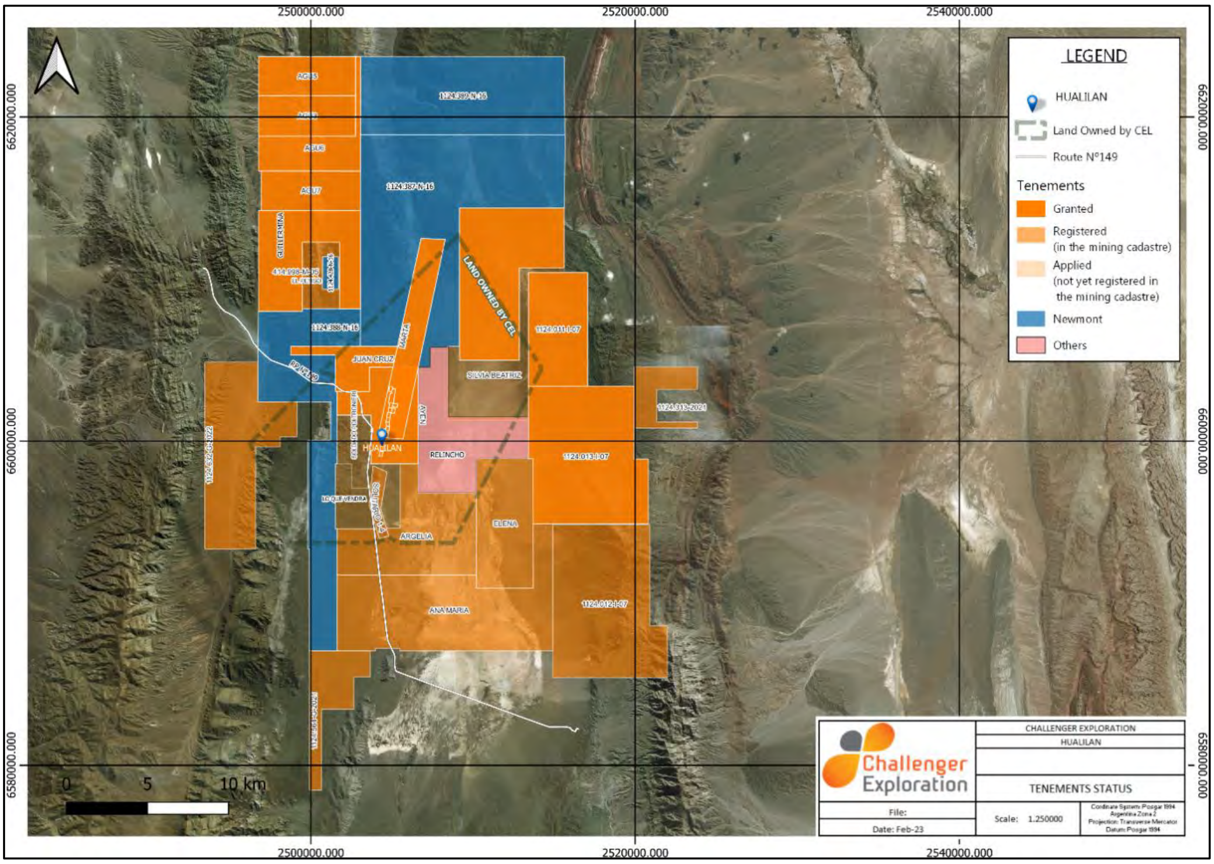


Figure 4-2: Map showing Mineral Tenements and 20,000 Ha surface rights

4.3 Permits Acquired

The Hualilan Property is comprised of 53 mining concessions listed in Table 4-1 and shown on Figure 4-2.

Table 4-1: Mineral Tenement Details

Nº	Name	File	Type	Current Owner	Legal Status
1		1124.188-G-2020	Exploration License	GMSRL	Applied
2		1124.248-G-2020	Exploration License	GMSA	Applied
3		1124313-2021	Exploration License	GMSA	Applied
4		295.122-R-1989	Exploration License	GMSA	Applied
5		338.441-R-1993	Exploration License	GMSA	Granted
6		1124564-G-2021	Exploration License	GMSA	Applied
7		1124.632-G-2022	Exploration License	GMSA	Applied
8		545.880-O-1994	Exploration License	GMSA	Registered
9		414.998-2005	Exploration License	Armando Jesus Sanchez	Applied
10		1124.011-I-07	Exploration License	Hugo Bosque	Granted
11		1124.012-I-07	Exploration License	Hugo Bosque	Registered
12		1124.013-I-07	Exploration License	Hugo Bosque	Granted
13		1124.074-I-07	Exploration License	Hugo Bosque	Granted
14	DIVISADERO	5448-M-1960	Mine	Golden Mining SRL	Granted
15	FLOR DE HUALILÁN	5448-M-1960	Mine	Golden Mining SRL	Granted
16	PEREEYRA Y ACIAR	5448-M-1960	Mine	Golden Mining SRL	Granted
17	BICOLOR	5448-M-1960	Mine	Golden Mining SRL	Granted
18	SENTAZÓN	5448-M-1960	Mine	Golden Mining SRL	Granted
19	MUCHILERA	5448-M-1960	Mine	Golden Mining SRL	Granted
20	MAGNATA	5448-M-1960	Mine	Golden Mining SRL	Granted
21	PIZARRO	5448-M-1960	Mine	Golden Mining SRL	Granted
22	ANDACOLLO	5448-M-1960	Mine	CIA GPL SRL	Granted
23	LA TORO	5448-M-1960	Mine	CIA GPL SRL	Granted
24	LA PUNTILLA	5448-M-1960	Mine	CIA GPL SRL	Granted
25	PIQUE DE ORTEGA	5448-M-1960	Mine	CIA GPL SRL	Granted
26	DESCRUBRIDORA	5448-M-1960	Mine	CIA GPL SRL	Granted
27	PARDO	5448-M-1960	Mine	CIA GPL SRL	Granted
28	SANCHEZ	5448-M-1960	Mine	CIA GPL SRL	Granted
29	MARTA ALICIA	2260-S-58	Manifestation of Discovery	GMSA	Granted
30	MARTA	339.154-R-92	Manifestation of Discovery	GMSA	Granted
31	AYEN	1124.495-I-2020	Manifestation of Discovery	IPEEM	Granted
32	AGU 3	11240114-2014	Manifestation of Discovery	Armando Jesus Sanchez	Granted
33	AGU 5	1124.0343-2014	Manifestation of Discovery	Armando Jesus Sanchez	Granted
34	AGU 6	1124.0623-2017	Manifestation of Discovery	Armando Jesus Sanchez	Granted
35	AGU 7	1124.0622-S-17	Manifestation of Discovery	Armando Jesus Sanchez	Granted
36	Guillermina	1124.045-S-2019	Manifestation of Discovery	Armando Jesus Sánchez	Granted
37	EL PETISO	1124.2478-71	Manifestation of Discovery	Armando Jesus Sanchez, y Cristina Mabel Siviski	Granted
38	SOLITARIO 1-4	545.605-C-94	Manifestation of Discovery	GMSA	Granted
39	ELENA (MARIA VICTORIA)	1124.328-G-2021	Manifestation of Discovery	GMSA	Registered
40	JUAN CRUZ	1124.329-G-2021	Manifestation of Discovery	GMSA	Granted
41	GABRIELA	1124.327-G-2021	Manifestation of Discovery	GMSA	Application
42	ARGELIA	1124.486-G-2021	Manifestation of Discovery	GMSA	Registered
43	ANA MARIA	1124.487-G-2021	Manifestation of Discovery	GMSA	Registered
44	ERICA	1124.541-G-2021	Manifestation of Discovery	GMSA	Application
45	SILVIA BEATRIZ	1124.572-G-2021	Manifestation of Discovery	GMSA	Application
46	SOLDADO POLTRONIERI	1124.108-2022	Manifestation of Discovery	GMSA	Registered
47	LO QUE VENDRÁ	1124.331-G-2011	Manifestation of Discovery	GMSA	Granted
48	SOLITARIO 1-1	545.608-C-94	Manifestation of Discovery	GMSA	Application
49	SOLITARIO 1-5	545.604-C-94	Manifestation of Discovery	GMSA	Application
50	SOLITARIO 6-1	545.788-C-94	Manifestation of Discovery	GMSA	Application
51	SENTAZON D1	546.516-C-94	Manifestation of Discovery	GMSA	Application
52	PEREYRA Y ACIAR D1	546.520-C-94	Manifestation of Discovery	GMSA	Application
53	MAGNATA D1	546.521-C-94	Manifestation of Discovery	GMSA	Application

The 53 mining concessions are in good standing, and none are presently subject to liens or encumbrances which are registered in the Mining Registry of San Juan Province. They can be of perpetual duration, subject to the owner’s compliance with the Argentine Mining Code provisions.

4.3.1 Surface Rights

In 2022, Challenger finalised an agreement to purchase 20,000 ha of land which contains the Hualilan project and the area surrounding the current mineral resources (Figure 4-2).

The size of the land acquisition was designed to:

- cover possible extensions within 5 kilometres in all directions; and,
- provide a sufficient footprint such that all mine infrastructure including the process plant, camp, office, waste dumps and TSF could be located on land owned by the Company.

The acquisition will greatly simplify the permitting process as the Company progresses towards production by removing the need to reach compensation and access agreements with landowners for mining, the provision of site infrastructure, and private contracts for the supply of water.

Under the Land Acquisition agreement, the Company will make total payments of US$1.2 million over 2 years. The first tranche payment of US$533,333 has been completed with two additional payments of US$333,333 payable in December 2023 and December 2024.

4.4 Ownership, Royalties and Other Payments

4.4.1 Ownership

The Hualilan Property is owned by Golden Mining S.A., controlling 536 ha of mining rights covering the Hualilan project site.

Golden Mining SA is 100% owned by Challenger Gold (Argentina) Limited which is a wholly-owned subsidiary of Challenger.

4.4.2 Royalties

The majority of the Hualilan Property is subject to a three percent royalty charged by the province of San Juan, based on the value of the recovered metal. Additionally, a 1.5% royalty, payable to a community development fund managed by the province of San Juan, applies to the concessions.

The Ayen concession was awarded to Challenger by the Instituto Provincial de Exploraciones y Explotaciones Mineras (“IPEEM”). The IPEEM is wholly owned by the province of San Juan and exists for the purpose of registering and administering mining rights as determined by the Mining Code. The royalty structure in tenements awarded by the IPEEM includes an additional 1%. At this stage none of the Potential Mining Inventory (PMI) included in the Scoping Study is located on the Ayen concession.

The costs of certain infrastructure, such as power lines and roads, may be recovered from the royalty payments in the event that the infrastructure is made available for public use.

4.4.3 Other Payments

Until Challenger completes the acquisition of the surface rights, the following payments are due under access and water supply agreements with the current holder of the surface rights.

4.4.3.1 ***Land Access Agreement Summary***

Camp

Following CEL's acquisition of the land around Hualilan, the actual camp use agreement (1124.163-G-21) is under evaluation to cancel upon the completion of the acquisition of the surface rights (Table 4-2).

Access

Following CEL's acquisition of the land around Hualilan, the land access agreement (1124.209-G-21) was terminated. The access agreement (1124.221-2023) was required for study the "El Peñon" area (Table 4-3).

Table 4-2: Camp land access agreements

File	Legal Status	Current Owner	Application Date	Ha Required	RC Report	Ha Registered	DGvC Report	RPI Report	Preliminary Registration	Stamp Tax	Publication Official Gazette	Landowner Acquisition	Concession
1124.163-G-21	Application	GMSA	30/4/2021	30.69	04/03/2022	30.69	09/06/2022	29/09/2022	NEXT	----	----	----	---
1124.585-2022	Application	GMSA	30/11/2022	457.45	NEXT	----	----	----	----	----	----	----	---

Table 4-3: Access agreements

File	Legal Status	Current Owner	Application Date	Ha Required	RC Report	Ha Registered	DGvC Report	RPI Report	Preliminary Registration	Stamp Tax	Publication Official Gazette	Landowner Acquisition	Concession
1124.221-2023	Application	GMSA	11/5/2023	1,64 Km	NEXT	----	----	----	----	----	----	----	---
1124.209-G-21	Desisted	GMSA	2/6/2021	30.69	NEXT	----	----	----	----	----	----	----	---

4.4.3.2 Water Supply Agreement Summary

Water use Permit Exploration

The current cost per cubic metre (m^3) of water for use in drilling, camp and mining roads, is ARS$200 (US$0.57 at the official exchange rate). The rate is based on an agreed usage of 1 m^3 for every metre of drilling. The payment is regulated by Hydraulic Department, which reports to the Minister of Works and Public Services of the provincial government.

Water Supply Agreements During Production

During the production stage a water usage fee applies under Article 26, including A1 law 2485-I of the San Juan Water Code. The remuneration rates for water services under Article 26, in accordance with articles 258; 259; 262 and 263, of the Water Code, Law No. 190-L, are:

1. Water Usage Fee:

Under Subsection A1, as a fee for all water use, the sum of ARS$200.00 per registered hectare, or per litre per second, as appropriate, is payable. The water usage of the project during production, after water recycling, will be based on 20,000 Ha which equates to an annual fee of approximately US$11,400.

2. Remuneration Rate for Water Services:

Under Article 26 Inc. 4 Law 2485-I of the San Juan Water Code, as remuneration for water services for each litre per second for mining use concessions, the sum of ARS$21,546 is payable for each litre per second usage annually. This equates to an annual fee of approximately US$1,850.

3. Special Concession Right:

Under Article 257 of the San Juan Water Code (Law 190-L) Special concession rights, the Department of Hydraulics will receive an annual special concession right that will be determined annually by that department when producing its budget. The special concession right will be at least equal to 75% of the difference in value between a hectare of arable land free of all improvements and without water rights, and that corresponding to an equal hectare but with water provision by legal concession. Based on the consultation with the Hydraulic Department, this payment is estimated at an amount is ARS$25,522 per litre/second which equates to US$2,200 per annum.

Drilling Construction Permit

The current cost of every hole is about ARS$28,000 (US$79.97 at official exchange rate). The payment is regulated by Hydraulic Department, which reports to the Minister of Works and Public Services of the provincial government.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The access to the deposit, from the city of San Juan, is a 120 km drive along double lane sealed highway which takes approximately 1.5 hours. The route takes the following itinerary:

- National Route No. 40: San Juan – Talacasto Station. Distance: 52km.
- Provincial Route No. 436: Talacasto Station – Junction of National Route No. 149. Distance: 23 km.
- National Route No. 149: Junction Provincial Route No. 436 – Deposit. Distance: 45 km.

This road corridor is enabled for the transit of all types of cargo and substances, hazardous and non-hazardous, and is used by numerous mining operations and projects in the Iglesia Department, including Barrick Golds Veladero Mine.

The highway passes within 500 metres of the property, which may be accessed internally by a network of dirt roads and drill pad access tracks (Figure 5-1). This includes roads to the top of the Hualilan Hills at Cerro Sur.

Figure 5-1: Main Highway (left) and network of access roads and drill pad access roads

5.2 Climate

The area is classified as desert. It receives an average annual rainfall ranging between 100 and 200 millimetres, falling primarily during the months of December and January. Annual average temperatures range between 16°C and 18°C, with minimums of –10°C and maximums to 40°C during December and January. Average humidity is 43.2%.

The property may be worked year-round. No permanent streams traverse the project area. There are numerous proximate groundwater aquifers, with water bore drilling undertaken by Challenger confirming a nearby source that will meet operational requirements for production (Chapter 14). A series of springs eight kilometres to the south presently provide water sufficient for operational requirements, including exploration drilling.

Wind directions are from the north and the south. The wind from the north, known locally as the "Zonda", is hot and characterised by low pressures and large, dense dust clouds. Reaching its maximum intensity from September through December, it remains benign the rest of the year. The southern wind, though cold, does not carry a dust load like that of the zonda.

Soils are characterised as infertile, desiccated, and generally alkaline. They support a sparse growth of grasses, cactus, thorn bush and other hardy species. Population is even more sparse, with less than 50 people living along the entire 120 km stretch of highway from San Juan to the property.

5.3 Topography and Drainage

The topography of the project area is dominated by the Hualilan Hills which are north-south trending hills approximately 200 metres in elevation above the plain. These hills are shown looking north from the main highway in Figure 5-2 with Cerro Sur in the foreground. Figure 5-3 shows the hills at Cerro Norte (looking north) with Figure 5-4 showing the hills at Cerro Sur looking south.

Figure 5-2: Hualilan Hills, looking north from the main highway with Cerro Sur in foreground

Figure 5-3: Hualilan Hills (Cerro Norte looking north) via drone

Figure 5-4: Hualilan Hills (Cerro Sur looking south) via drone

The hills are formed by thrusting associated with the compressive tectonics responsible for the formation of the Andes. The hills comprise 45-65 degree dip slopes on their western side. There are gaps between the hills associated with east-west lateral faults. The surrounding plain has a gentle slope toward the east.

The Hualilan project area is located in a closed drainage basin; this basin covers an area of 317 km^2 and has watercourses of a non-permanent nature (Figure 5-5). Drainage in the project area is from west to east, with the cap area between the Cerro Norte and Cerro Sur hills the site of one of the non-permanent watercourses. The site plan includes flood barriers to divert any floodwaters associated with short term flash flooding.

Figure 5-5: View to the West, from the north central area, of the Pampa de Hualilan where the development of the Alluvial Fan Unit can be seen, as well as rocky outcrops of the Hualilan Mountains.

The centre of the enclosed drainage basin is located south-east of the project area (Figure 5-6) and is known as the Pampa de Hualilan. The northern half of the Pampa de Hualilan comprises a sandy and muddy plain, which in the southeastern portion can form a saline muddy plain. The tailings storage facility (TSF) has been located to the north-west of the Pampa de Hualilan to provide an additional layer of safety, with any spill from the TSF draining into the four-way dip enclosed by the Pampa de Hualilan.

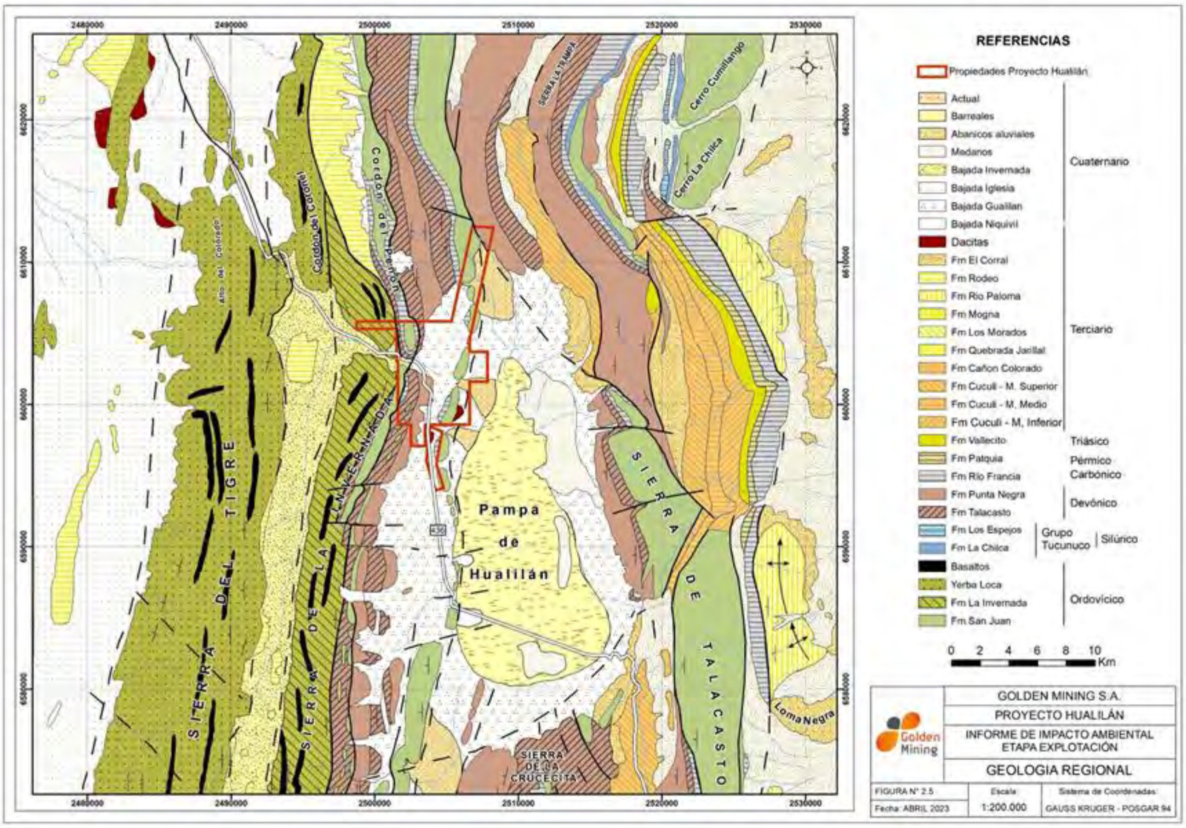


Figure 5-6: Pampa de Hualilan in relation to the project area

In the area of direct influence and in the footprint of the project, no cryogenic geoforms - glaciers and/ or periglacials – have been identified according to the existing information in: Phase I of the National Inventory of Glaciers prepared by IANIGLA (www.glaciaresargentinos.gob.ar), within the framework of the provisions of National Law No. 26,639/10; or, in the Provincial Inventory of Glaciers prepared by the National University of San Juan (Institute of Geology and Institute of Hydraulic Research), within the framework of the provisions of the Law Provincial Nº 1.076-L respectively.

5.4 Local Resources and Infrastructure

Presently, the Hualilan base of operation for the project is located on site. Resources include a modular camp capable of accommodating 100 people, with full kitchen facilities to provide meals for up to 100 people, as well offices and modular living facilities.

Communications are provided by high-speed satellite internet. There is a Security Guardhouse including a checkpoint with boom gates located at camp entry.

There is a core storage facility capable of holding 300,000 metres of drill core, and separate core logging, cutting and sampling areas.

Electrical power is provided by a combination of solar and diesel generators. Challenger has rented a portable, modular sewage treatment plant and hazardous waste storage tanks for the camp, which are emptied bi-weekly.

Other infrastructure on site includes three locations for drillers camps and a storage area for drill supplies.

In San Juan city, the company leases a small office which serves as a base of operation for its social and community relations personnel.

5.5 Physiography

Lying within foothills of the Andes Precordillera, the broader project area exhibits a basin and range typography characterised by a series of steep northerly-trending hills and small mountains within extensive flat valley floors. Average elevation of the lower portions of the Hualilan property is approximately 1,720 metres ASL, with relief in the order of 200 metres. A range of steep northerly trending hills (Figure 5-2) up to 500 metres in width and flanked by extensive plains (‘pampas’) make up much of the property area where the current mineral resource is located. Outcrops of positive relief diabase porphyries form small hills, which break up the monotony of the plains. East-west faulting across the trend of the hills forms a series of sharp, eroded gulches or quebradas marking the fault zones.

Due to its location at less than 2,000 m ASL, the Hualilan property is not impacted by Argentina’s glacial law. Passed in 2010, this law declares all of Argentina's glaciers to be “strategic reserves of water” and “public property.” It bans mining and oil extraction in the glaciers' watersheds, and places strict restrictions on industrial activity in surrounding lands, thereby expanding an earlier definition of what constituted periglacial area.

6 HISTORY

6.1 Exploration History of the Project

Modern Exploration at Hualilan started in 1984. Prior to that time there is an incomplete history of discovery, exploration and mining. The following is an amalgamation of information from internal company reports, (Jenks, The Hualilan Property Geological Appraisal, 2003) , (Rios, 2016) and (Mallea, 2020).

The discovery of mineralisation exposed at surface at Hualilan is attributed to a mule driver in 1751. The discovery was subsequently assigned as church property.

6.2 Prior Ownership

The deposits were worked from 1790 until 1815 by at least 14 owners/operators. In 1802 there was a family conflict over mining, suggesting that the mining was profitable and there was competition between operators.

Mining restarted in 1826 after Argentinian Independence in 1816, and during the 'Period of Organisation'. There are 19 different excavations that were worked during this period. From 1846 there were a series of sales to mining companies including Anglo Argentina Company, Anglo Argentina de Gualilan Company, and the Company Argentina Limited, without significant work proceeding.

A company formed by "El Tontal and Castano" raised capital in 1870 and began work in Hualilan in 1872. An amalgamation circuit was installed, but the inability to treat sulphide mineralisation resulted in mine closure.

In 1875, a new English company "La Argentina" resumed work, installing two roasting ovens with capacity of 80 tons per day, working with a cut-off grade of 21 g/t gold. During its peak approximately 160 miners worked in about 30 mines at Hualilan. Many of the underground workings, buildings and infrastructure at the Hualilan Project date from this time. The mine did not achieve expected results and the operation was abandoned. On 19 May 1883, the projects were granted to "Lloyd and MacKenzie Gold Mines Property".

In 1914, a cyanidation plant was installed to treat tailings that came from the mining carried out by the English (mining of El Tontal-Castano and La Argentina).

In 1936, a 'reserve' estimate and geological evaluation was completed by la Direccion de Minas y Geologia de la Nacion.

In 1947, the project was leased from la Sociedad Anonima Guanizuil Rural and Comercial and some tailings were processed by cyanidation.

La Direccion Nacional de Geologia completed an additional Government sponsored survey in 1951. Detailed surveys of all accessible underground workings were made, geological mapping was done of many of the tunnels and 193 samples were taken and analysed for gold and silver. An additional ‘reserve’ estimate was also completed.

Starting in 1955, the company “Los Marayes Mining Company” installed cyanidation with a Merrill-Crowe circuit. This operation ran for approximately three years treating 6-7 kt of mineralised material from underground, 2-3 kt of stockpiled mineralised material and 1 kt of tailings. Gold grades of 2-4 g/t are estimated. Later the Company was dedicated to the manufacture of sulfuric acid from the sulphide mineralised material and continued with exploration at "Pirguen" (precise location uncertain).

Ownership was transferred to Meteor SAIC in 1959. Tailings were processed and some high grade pillars were removed from the upper levels for processing. The processing plant was decommissioned in 1959. In the 1960’s, Aluvion SRL worked Level 3 of the Pique Sur shaft with limited success.

More recently modern exploration on the Hualilan Project prior to the Company extends to over 150 drill holes. The key more recent historical exploration drilling and sampling periods are listed below with more detail on modern exploration provided in Section 8.

In 1984 Lixivia SA channel sampling and 16 RC holes (AG1-AG16) for 2040m.

In 1995 Plata Mining Limited undertook 33 RC holes (Hua- 1 to 33) plus 1500 samples.

In 1998 Chilean consulting firm EPROM (on behalf of Plata Mining) undertook systematic underground mapping and channel sampling.

From 1999-2001 Compania Mineral El Colorado SA (“CMEC”) drilled 59 core holes (DDH-20 to 79) plus a 1700m RC program.

From 2003 - 2006 TSX listed company La Mancha undertook 7447m of DDH core drilling (HD-01 to HD-48).

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

Hualilan is in the Argentinian pre-cordillera, east of an active continental convergence zone, where the Nazca oceanic plate is being subducted below the South American craton. The subduction zone below Hualilan is characterised by the Nazca Plate having a shallow dip less than 30 degrees from horizontal (Figure 7-1).

The shallower heating below Hualilan results in increased crustal contribution to the magma which is forming at depth above the subducting plate, and intrusions further east in the upper crust than would normally be anticipated. Whole rock geochemistry of Dacite in the Calingasta Valley (including in samples from Hualilan) indicate calc-alkaline to alkaline compositions (6-8 wt% $Na2O3 + K2O$) and high Na/K (2.0 - 3.0) for unaltered intrusions (Kay et al.,1998) which are fertile for porphyry-style mineralisation and associated metalliferous deposits.

The significant host sedimentary units at Hualilan are the Ordovician aged San Juan Limestone, which is overlain by the Silurian age La Chilca and Los Espejos Formations. The upper part of the Ordovician limestone contains bedding parallel faults and reverse faults created during arc convergence. The host rocks strike approximately north and dip west at 25°–70°. The folding and thrusting are interpreted to have begun in the Silurian to mid-Devonian age and was again folded from late-Devonian through late-Permian age.

The host rocks are intruded, post-folding, by dacitic stocks, sills and dykes. Magmatic activity during the time of subducting slab flattening occurred between 18 to 7 Ma. The precise age of the intrusions at Hualilan is unknown.

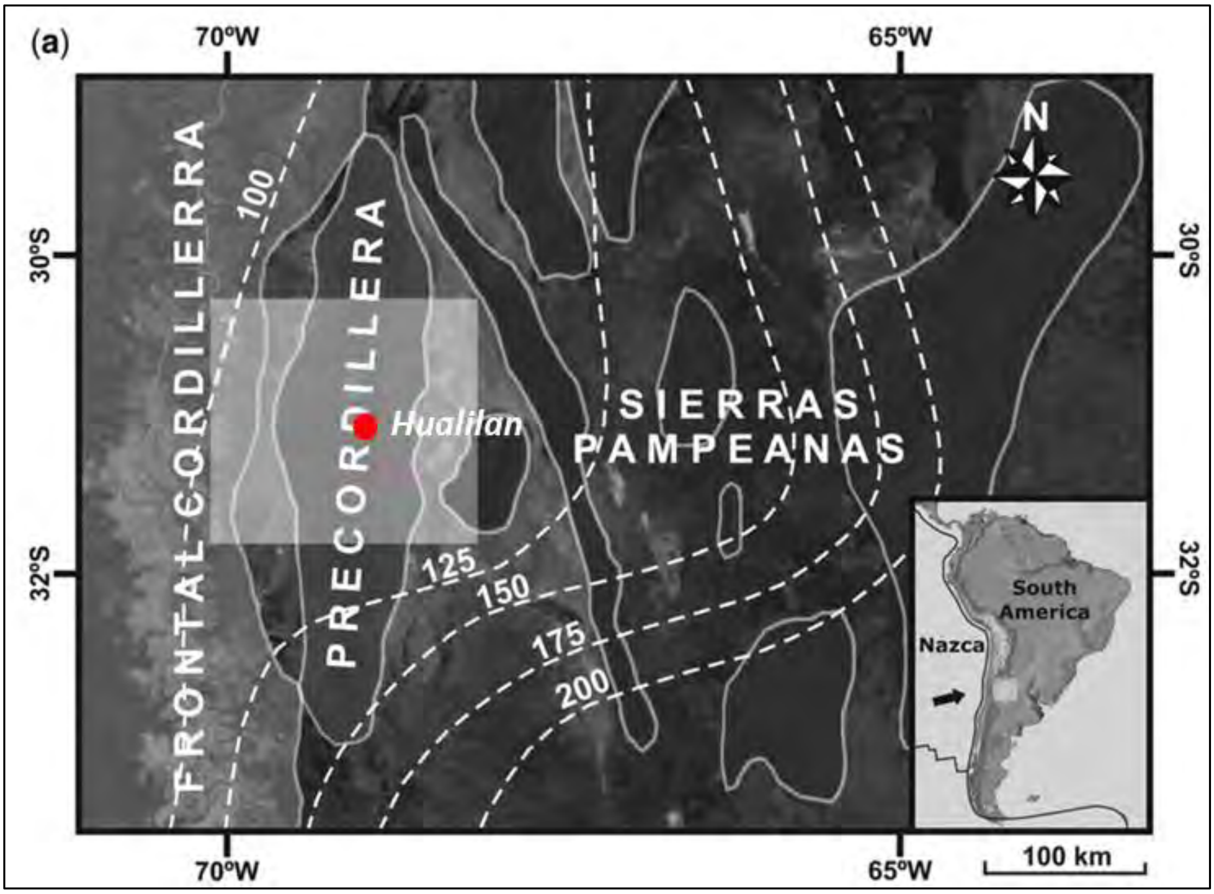


Figure 7-1: Plan showing contours of depth (kilometres) of the subducting Nazca plate below the South American craton illustrating the shallower slab below Hualian influencing magmatism and crustal fluid movement.

Source: Oriolo, 2014.

7.2 Local Geology

The San Juan Formation is a massive detrital limestone sequence that is responsible for the distinctive, north-South striking Hualilan Hills. The upper 240 m of the San Juan Formation is exposed at Hualilan (Figure 7-2). A west-dipping thrust fault on the east side of the hills marks the contact of the San Juan Formation with Miocene age sedimentary rocks to the east. Additional faults, sub-parallel to bedding, are evident in the limestone. Conformably overlying the limestone are Silurian age siltstone and fine-grained sandstone. During mineralisation, bedding parallel faults, cross faults, steep reverse faults and lithology contacts have provided pathways for intrusive porphyry sills and hydrothermal fluids.

The dacitic intrusions occur as small stocks, dykes and sills which may be recessive in the valley areas west and east of the Hualilan Hills and within the San Juan Formation. Ground magnetic data clearly indicates the location of the intrusive complex under cover (Figure 7-3). The intrusions have fractionated porphyry texture and are dominated by plagioclase feldspar, potassium feldspar, quartz, hornblende and biotite phenocrysts.

The depth of surficial partial oxidation (weathering) ranges from 5 m to 50 m below surface and is dependent on fault and fracture density, being deeper around the fault zones. There is very little complete oxidation of the bedrock. Complete oxidation, where present, extends only a few metres below the surface.

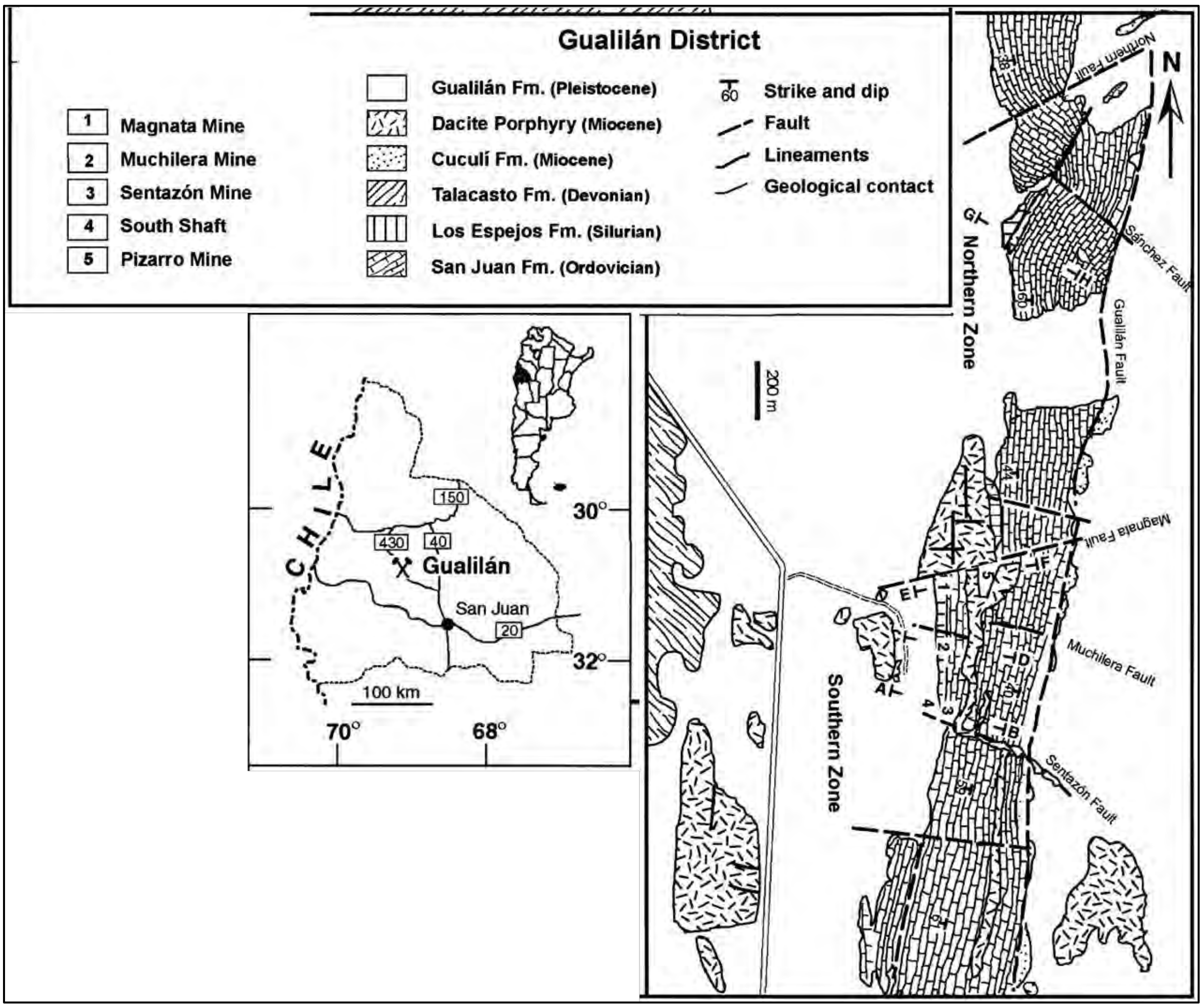


Figure 7-2: Location and Hualilan (Gualilan) Project geology.

Source: Logan, 2000.

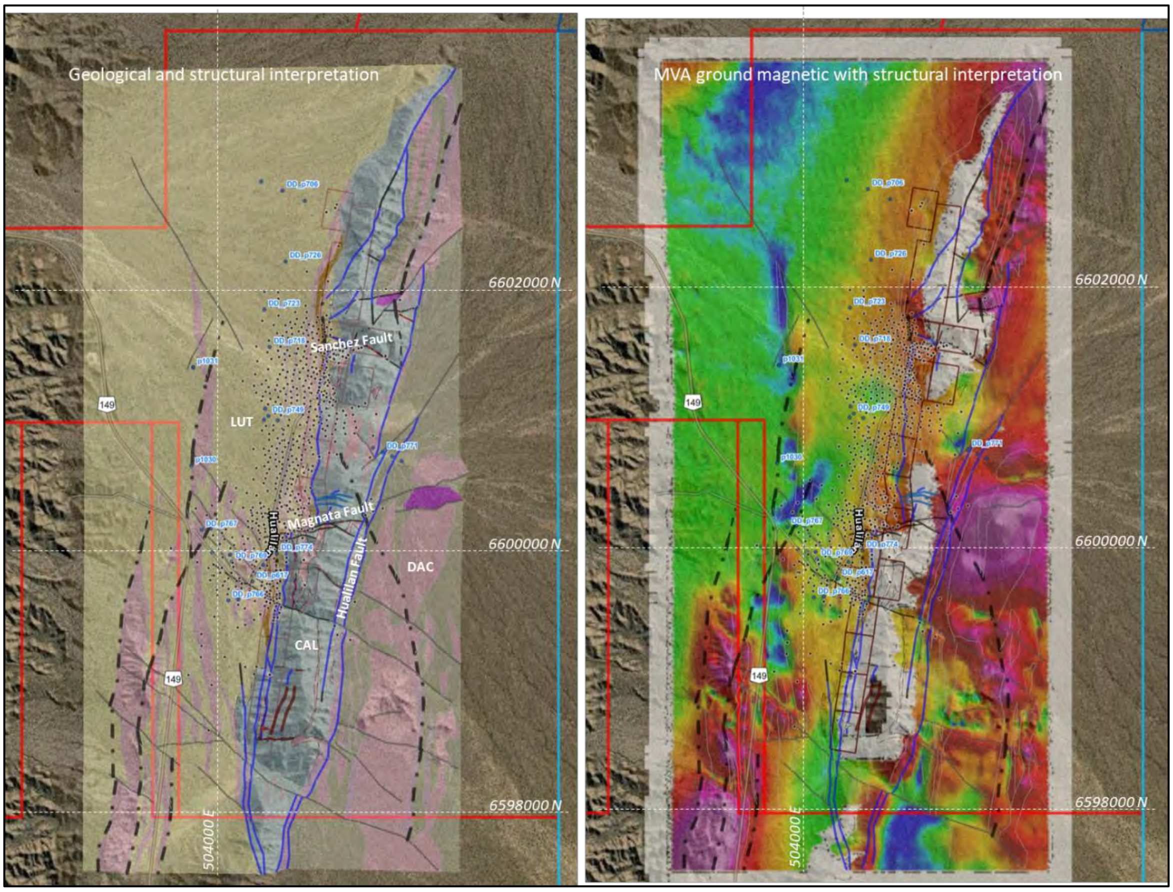


Figure 7-3: Left: Hualilan geology and structure at surface from detailed mapping and drill hole data Right: interpreted from ground magnetic data over covered areas and where there is no drilling.

7.3 Deposit Type (s)

Hualilan is a zinc skarn (containing Zn, Ag and Pb with only minor Cu), overprinted by epithermal to mesothermal low sulphidation gold (Au-Ag). Both stages of mineralisation have utilised the same structures. Zinc skarns typically occur near convergent plate subduction margins, such as seen throughout western South America.

The unusual characteristic of Hualilan is that the zinc skarn is overprinted by the later stage epithermal-to-mesothermal gold mineralisation. This later style of mineralisation typically occurs in similar geological settings to zinc skarn deposits, but is commonly formed either closer to the surface from cooler hydrothermal fluids, or from fluids circulating at a later stage of the hydrothermal system. At Hualilan, the same structures that controlled the zinc skarn also control the epithermal mineralisation, although end-member examples of both styles of mineralisation are present.

Both the zinc skarn and the epithermal styles of mineralisation develop distal to the causative intrusive centre. The intrusive centre may or may not be altered and mineralised. Where the causative intrusive centre is mineralised, porphyry-style (Cu-Au-Ag-Mo) is common. Figure 7-4 illustrates genetic models for distal skarn and for epithermal styles of mineralisation at approximately the same scale. Both styles of mineralisation commonly occur at least 1 kilometre away from a porphyry intrusive centre but may be several kilometres depending on the efficiency of the hydrothermal fluid pathways.

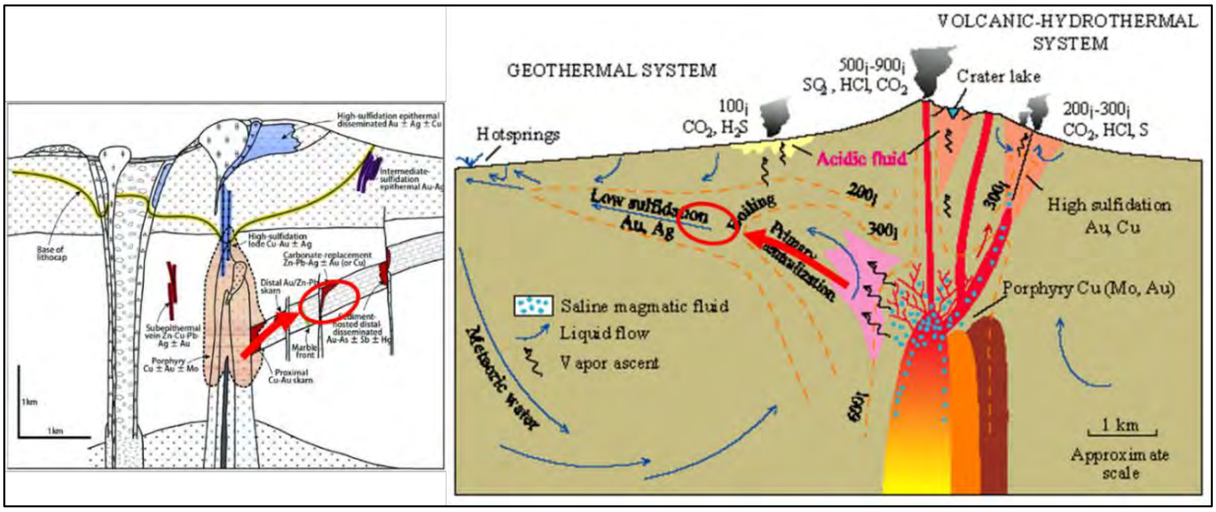


Figure 7-4: Schematic diagrams illustrating the spatial relationship between intrusive source and zinc-lead-silver skarn and low sulphidation epithermal gold-silver mineralisation.

7.4 Lithology

The sedimentary host rocks in the project area are described in detail in the report for Challenger Gold (Heredia & Mestre, 2022). A summary of the sedimentary rock sequence is shown in Figure 7-5.

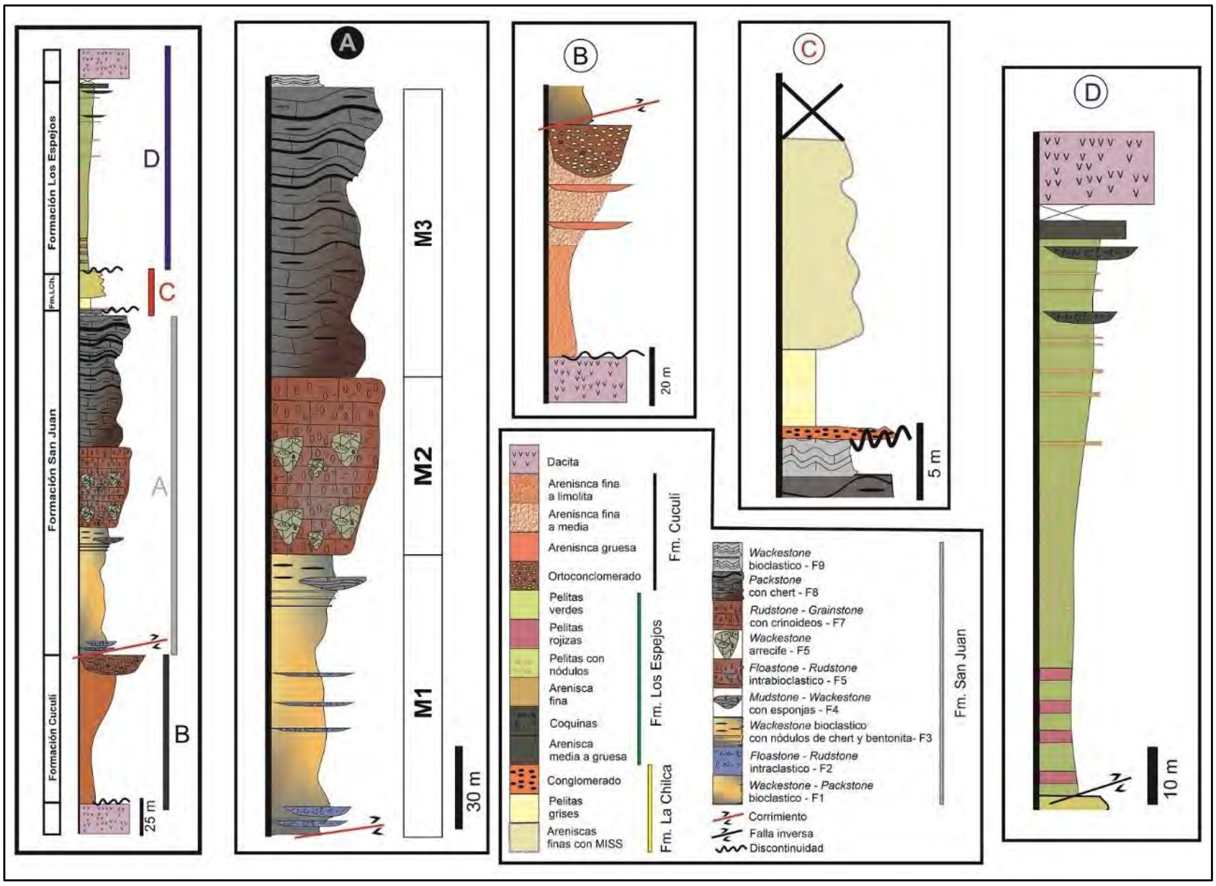


Figure 7-5: Summary of the host sequence stratigraphy at Hualilan.

Source: Heredia S and Mestre A, 2022

7.4.1 San Juan Formation (Middle Ordovician)

The oldest rocks are a sequence of bioturbated detrital limestone with minor dolomite and reef limestone which is approximately 240 m thick. The rocks form part of the San Juan Formation which strikes approximately north-south and repeats throughout the pre-cordillera of San Juan province. The original sequence commonly has a basal detachment thrust fault, such that the limestone exposed at surface in the pre-cordillera is structurally thickened and repeated against north-south striking thrust faults. In the project area, a reactivated thrust fault at the base of the Limestone is known as the Hualilan Fault. The limestone is relatively resistant to erosion and so forms a series of prominent north-south trending hills and ridges throughout the pre-cordillera (Figure 7-6).

Figure 7-6: View north from Cerro Sur of the prominent hills at Hualilan that are composed of San Juan Limestone.

7.4.2 La Chilca Formation (Late Ordovician – Early Silurian)

The La Chilca Formation is a sequence of siltstone (phosphorite) and breccia which is 2 – 10 m in thickness which is paraconformable on the San Juan Formation (Figure 7-7). Phosphate content averages approximately 0.4 wt% P_2O_5 and 500 ppm total rare earth oxide (TREO), weighted towards light REO with elevated Cerium (Ce) common. The La Chilca Formation has been deposited in a shallow marine platform environment.

Figure 7-7: La Chilca Formation siltstone dipping west at the contact with San Juan Limestone at the portal to the Sentazon underground workings.

7.4.3 Los Espejos Formation (Silurian)

The Los Espejos Formation is paraconformable on the La Chilca Formation. The base is generally fine-grained sandstone, grading upwards to well bedded siltstone with repetition of the coarser and finer units (graded beds). In the western part of the project, the siltstones are commonly well bedded and sandstone units are less common. The thickness of the unit is not known as it is open to the west.

7.4.4 The Cuculi Formation (Miocene)

The Cuculi Formation is part of a synorogenic basin sequence formed in response to deformation and uplift of the pre-cordillera. Coarse grained, red coloured sandstone and conglomerate exposed east of the San Juan Formation at Hualilan is typical of the upper part of the Cuculi sequence (Figure 7-8). The conglomerates are poorly sorted and polymictic including igneous and sedimentary material. The upper part of the Cuculi Formation is known from regional paleontography to have an age of 10.8 to 10.4 Ma.

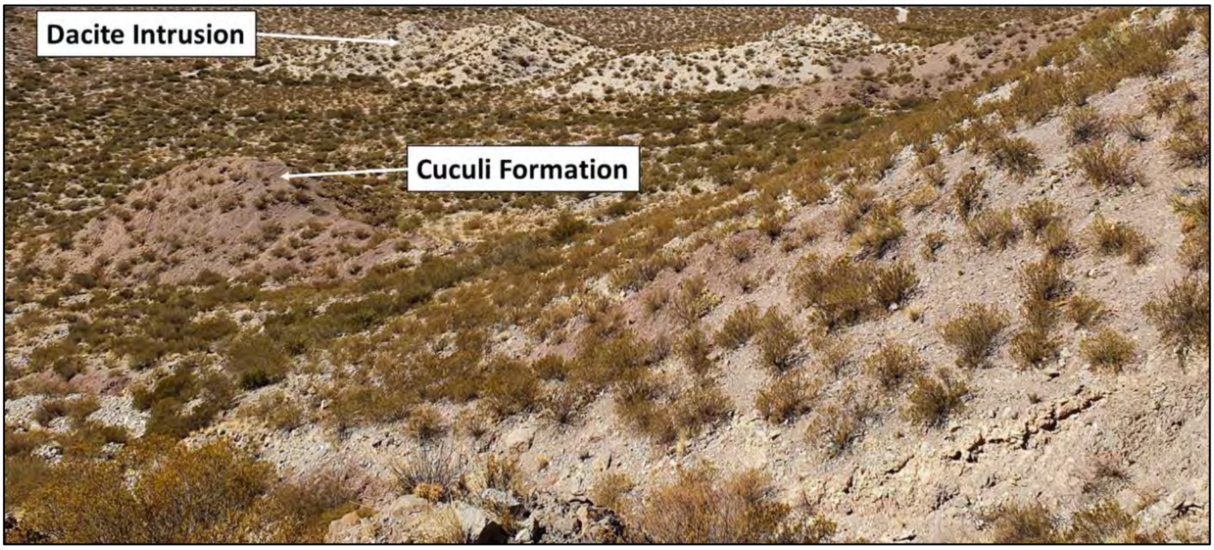


Figure 7-8: Red coloured sandstone and conglomerate of the Cuculi Formation exposed on the eastern side of the Hualilan Hills, intruded by dacite.

7.4.5 Dacitic intrusions (likely Miocene)

The intrusions at Hualilan are porphyritic (quartz-plagioclase-biotite/hornblende phenocrysts) with a dacitic composition (Figure 7-9). The age of the intrusions is unknown; however, they are hosted by all rock types including the Cuculi Formation and so are likely to be less than 10 Ma (Miocene – Pliocene). The least altered intrusions have Na>K with total Na+K of approximately 5 wt%.

Dacite with abundant hornblende phenocrysts (up to 5%) are intruded into the central part of the project area, are commonly altered so that the hornblende is partially or completely replaced by chlorite, epidote or sericite, and are closely associated with skarn and gold mineralisation. Dacite with biotite phenocrysts (up to 5%) and rare hornblende occur on the eastern and western flanks of the mineralisation. Biotite phyric dacite has been observed with skarn alteration, although these intrusions are commonly the least altered, and rarely host significant skarn or gold mineralisation. No overprinting relationships have been observed between the biotite and hornblende phyric dacites.

Dykes of dacite composition commonly have flow banding parallel to and near contacts with host sedimentary rocks. The mineralisation in the dacite is fractured but otherwise undeformed and so appears to be post-orogenic.

Figure 7-9: Weakly altered biotite phyric dacite porphyry in GNDD374 at 340m (Na+K is 5.8%, Al is 8.5%), drilled east of the Hualilan hills.

The dacite geochemistry indicates fractionation from more mafic andesitic, through dacite to rhyodacitic compositions (Figure 7-10).

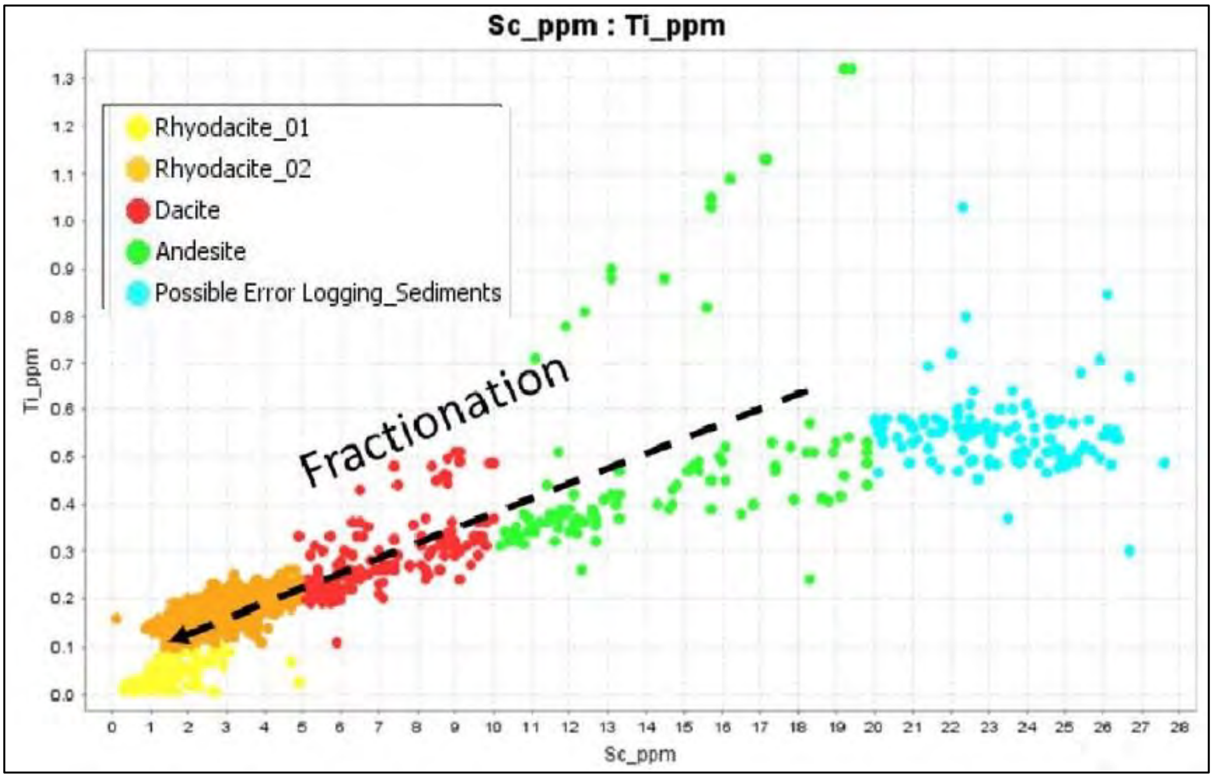


Figure 7-10: Scandium-Titanium for 18,013 drill core samples logged as dacite.

Source: Via S M, 2021.

Rarely observed in drill core are mafic dykes, which may represent least the fractionated magmatic components seen above in Figure 7-10. These dykes are usually finer grained, contain abundant plagioclase and mafic minerals (pyroxene, hornblende) and are usually un mineralised.

7.5 Structure

The Precordillera where Hualilan is located is part of a thin-skinned fold and thrust belt underlain by Cambrian and Proterozoic sequences. The edge of the continental shelf is located approximately 2 kilometres to the west of Hualilan.

Surface mapping and interpretation of ground magnetic data at Hualilan has identified the main NNE striking faults such as the Hualilan Fault with cross faults striking 070° (Sanchez & Magnata Faults) and 120° (Sentazon Fault) as shown in Figure 7-11. Work is continuing to build models of the second order faults and detailed structural controls on mineralisation. Most of the mineralised domains are controlled by fault-fracture zones which are west dipping, commonly parallel to bedding. Many of these fault zones are reactivated thrust faults that were initially formed during convergence from the Carboniferous through to the Tertiary, including during mineralisation in the Miocene to Pliocene. A second set of controlling faults with the same strike, dip steeply east. Additionally, the cross-faults striking 070° are also mineralised.

A 3D model of the geology, highlighting key structures has been developed and is periodically updated by the exploration geologists. Structural information is being added to this model, highlighting bedding orientations, faults and fracture zones. Observed throughout the project is a fault zone at and parallel to the upper contact of the San Juan Formation. This fault zone is commonly mineralised (Figure 7-11, Figure 7-12 and Figure 7-13) and also ramps down through the limestone where manto-style replacement mineralisation may form in the limestone as it does at Sentazon, Muchilera, Magnata, Norte and Verde. Similar faults are interpreted ramping through the Los Espejos Formation siltstone. Thrust inversions on these faults are responsible for the formation of ramp anticline folds and east dipping faults. During mineralisation, reactivation of these faults focuses hydrothermal fluid and become the site of concentrated skarn alteration and mineralisation.

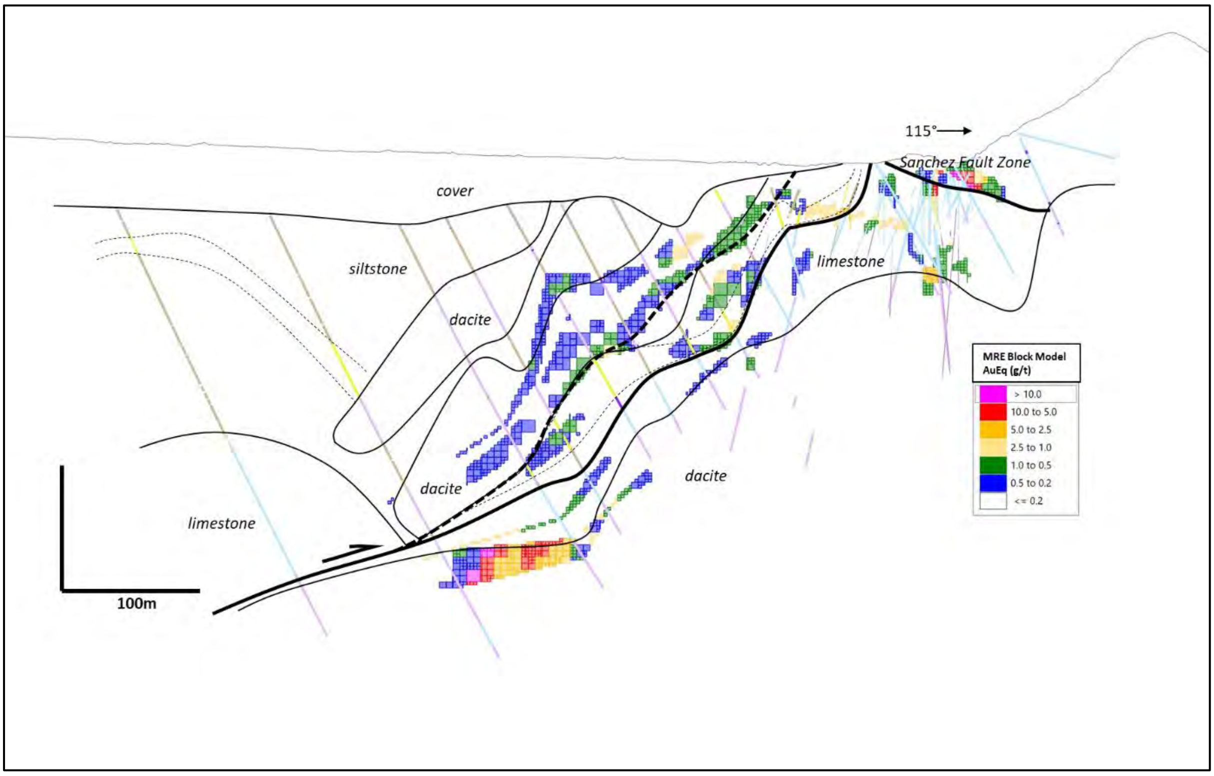


Figure 7-11: Geology and structure section through Sanchez – Verde (Cerro Norte).

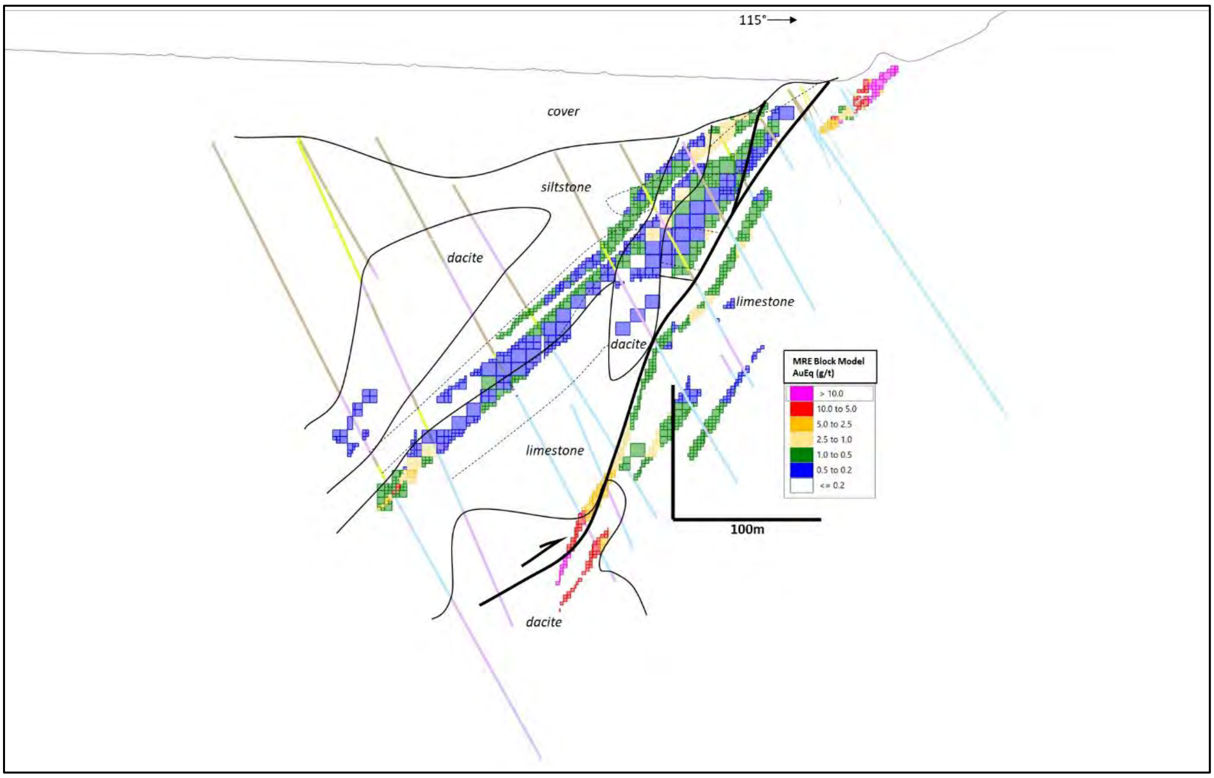


Figure 7-12: Geology and structure section through Verde and Norte Manto (Cerro Norte).

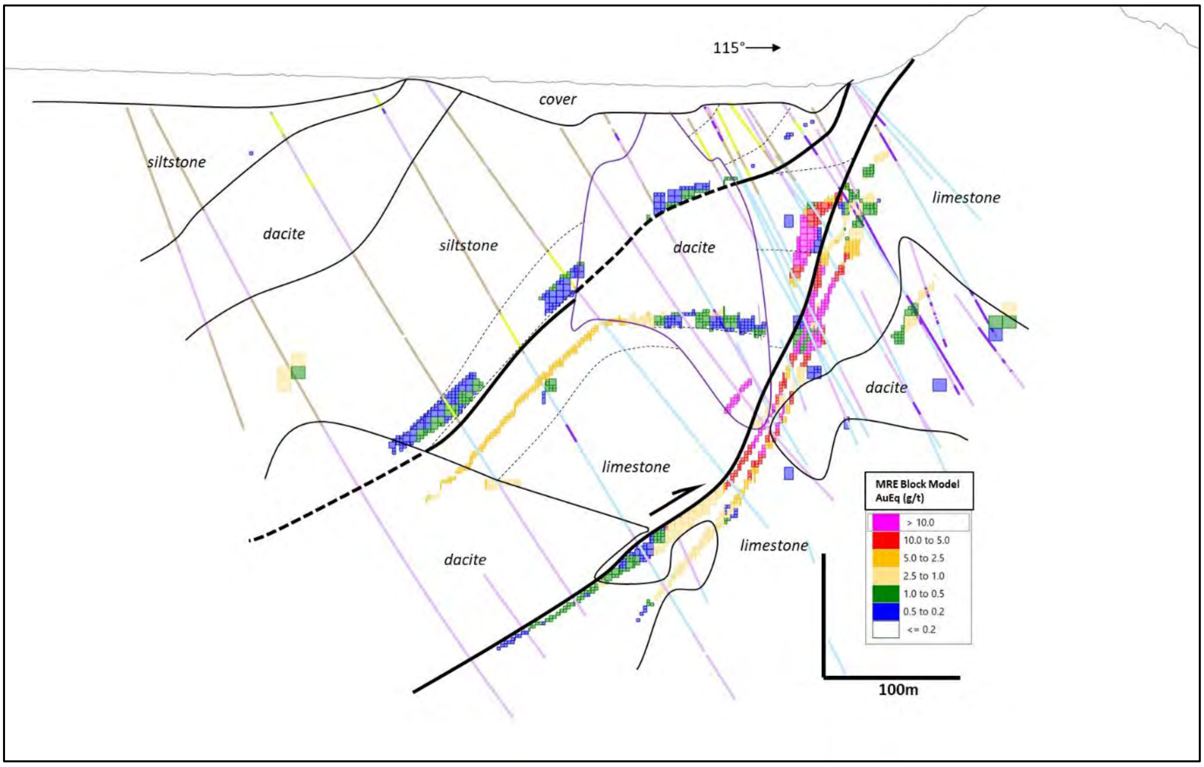


Figure 7-13: Geology and structure section through Sentazon (Cerro Sur).

7.6 Alteration

Alteration mineral assemblages are dependent on host lithology. There are multiple stages of alteration relating to the evolution of the mineralisation. Generally, the alteration can be divided into prograde skarn alteration, followed by stages of retrograde alteration as the activity of the hydrothermal system reduces.

Prograde skarn in siltstone and limestone is dominated by hedenbergite (up to 60%) and smaller amounts of garnet (~5-25%), with minor wollastonite and recrystallisation of the limestone (marble texture). Hedenbergite occurs as fanning to radiating aggregates, in places bladed crystals, which range in size from very fine (0.1-0.2 mm) to large (10 cm). Garnet is yellow andradite and occurs as anhedral to dodecahedral grains, in places intergrown with hedenbergite or including small hedenbergite crystal laths.

Retrograde alteration occurs as a continuum, from weak overprinting of skarn textures to complete obliteration of the original texture and mineral assemblage. Retrograde alteration includes quartz and carbonate, which fills irregular cavities. Fibrous and very small columnar-to-bladed crystal forms may be wollastonite replacement, or bladed carbonate replacement. Pyroxene has been observed in thin section and occurs with secondary assemblage of carbonate, sericite, hematite, epidote and chlorite. Retrograde alteration also includes magnetite, which occur as irregular masses replacing the skarn minerals. Adularia has been rarely observed to be associated with the retrograde alteration. In the siltstone, retrograde alteration is dominated by chlorite with lesser epidote and sericite.

More advanced retrograde alteration is dominated by sericite and carbonate, with some hedenbergite or epidote altered to an extremely fine-grained assemblage of carbonate and clay or is stained by red-brown iron oxides.

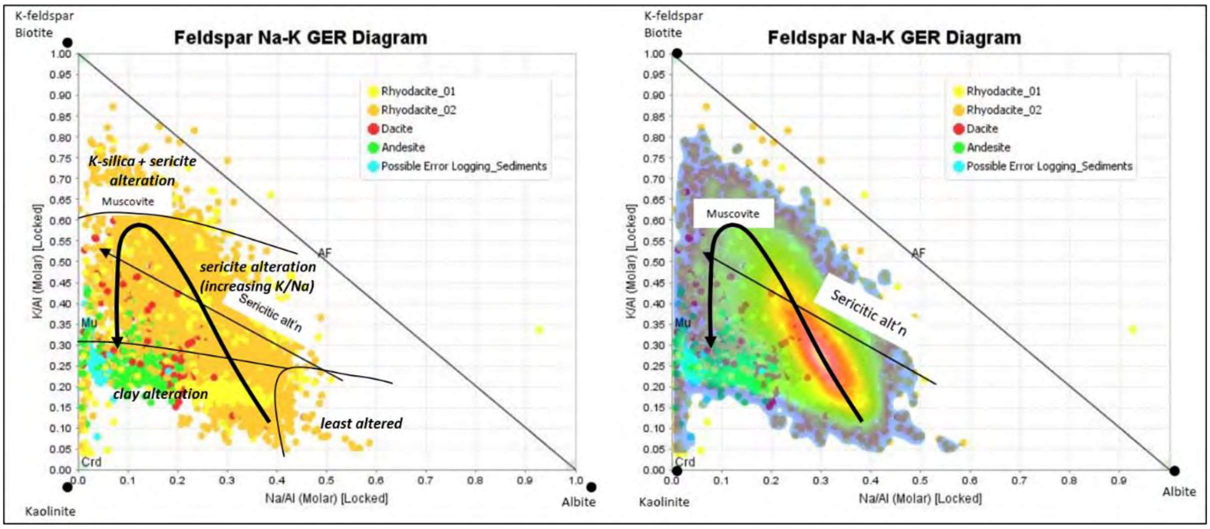


Figure 7-14: Na/Al v K/Al ratios from drill core samples in dacite, illustrating the progression during retrograde alteration towards potassic (K alteration). Analysis of 18,013 samples logged as DAC (from Via, 2021).

Source: (Via, 2021).

The alteration assemblage in the dacite consists of plagioclase altered mainly to sericite and carbonate, with epidote and chlorite alteration of hornblende, overprinted by muscovite and carbonate. Prograde pyroxene and garnet alteration in the dacite are rare and mostly occur near the contact with limestone.

Retrograde alteration results follow a progression towards a more potassic (sericite) alteration before progression to potassic clay alteration late during retrograde alteration (Figure 7-14).

Spatially, the K/Na ratio is an indication of retrograde alteration in dacite. Figure 7-15 shows a plan through K/Na ratio isosurfaces at 1,700 m RL (without directional bias) from samples logged as dacite in all drilling completed at the date of the MRE. Increased K/Na is evident in the dacite around and within the mineralised domains and generally follows controlling structures (faults).

On-strike extensions of higher K/Na ratio in dacite are evident, particularly north of Sanchez and to the west of Verde.

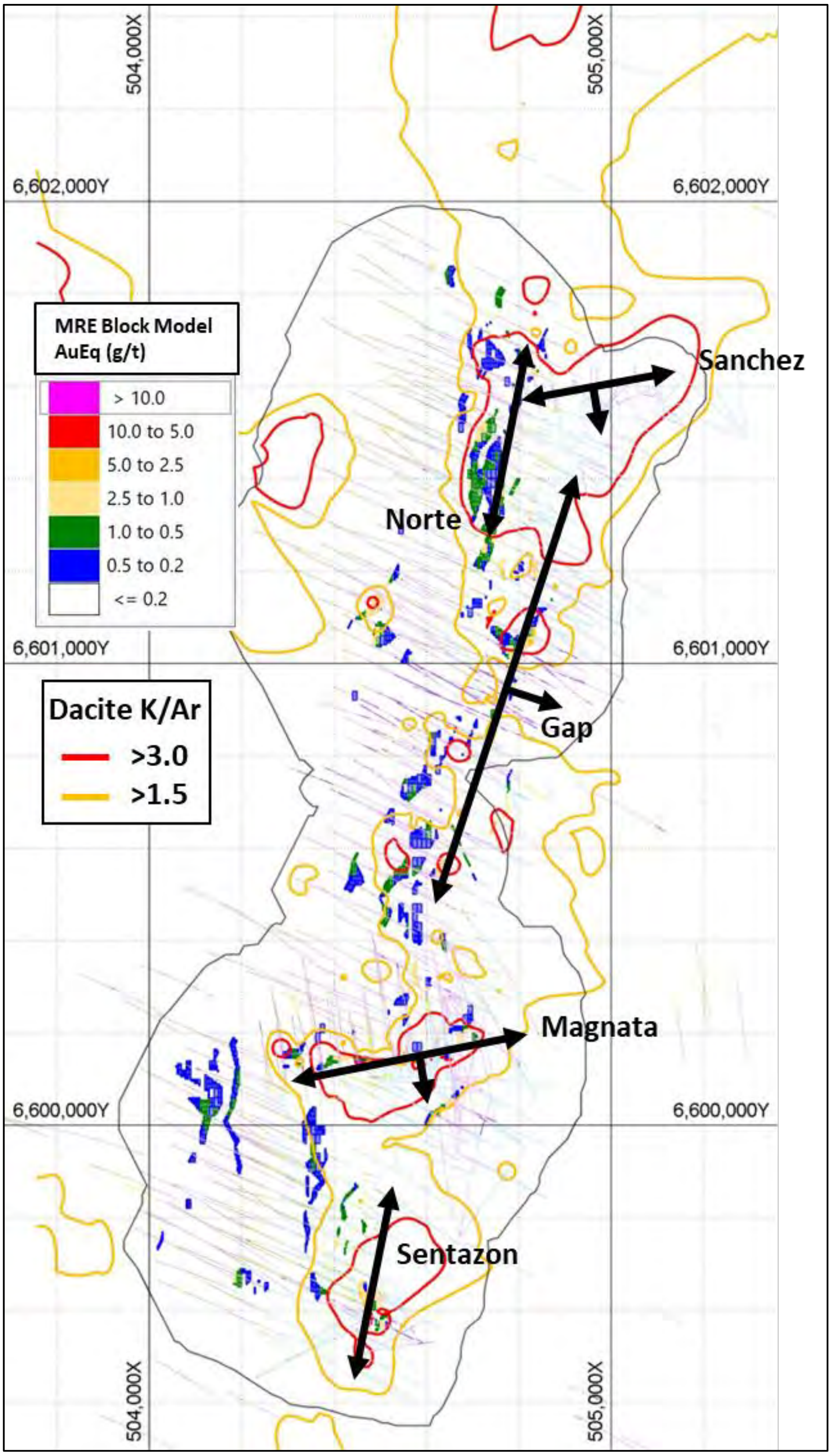


Figure 7-15: Isosurface model (RBF function) for K/Ar alteration in samples logged as DAC at 1,700m RL. Outside of the dacite, the alteration is difficult to trace as it is dependent on lithology. Retrograde potassic alteration has a reasonable correlation with mineralised domains.

7.7 Mineralization

Mineralisation occurs in all rock types at Hualilan (Figure 7-16, Figure 7-17 and Figure 7-18), principally in limestone, siltstone to fine grained sandstone and Dacite intrusions. Sulphide minerals are undeformed, temporally consistent with textures in the Dacite and therefore mineralisation is interpreted to postdate the deposition of the youngest sedimentary unit (Cuculi Formation) and intrusion of the dacite.

The mineralisation generally follows the alteration. Earlier zinc skarn mineralisation occurs with the prograde skarn and later mineralisation occurs with retrograde alteration.

Mineralisation occurs in the following geological settings:

- Steeply dipping, fault-hosted, striking east-northeast (approx. 070° strike);
- Fault hosted and bedding-parallel replacement deposits, striking south to south-west and commonly dipping at 30°–70° to the northwest and less commonly dipping southeast;
- At the contact of intrusions – particularly where the intrusion contacts limestone.

The mineralisation has true thicknesses of up to 60 m containing prograde skarn alteration, skarn mineralisation and commonly retrograde alteration and mineralisation. Intersections between faults and intersections between the bedding-parallel mineralisation and the east-striking cross veins are important for localising thicker, higher-grade zones of mineralisation.

Prograde mineralisation consists of sphalerite, lesser galena, rare chalcopyrite, pyrrhotite and magnetite which replaces hedenbergite and garnet and so is paragenetically slightly later that the initial skarn alteration. Sphalerite with red/brown colour is relatively iron-rich and typically higher temperature and evolves to the later yellow coloured sphalerite. Growth rims of lighter coloured sphalerite have been observed around earlier formed darker sphalerite. Pyrite may also be deposited with the prograde mineralisation, although it is difficult to distinguish with the later, more dominant (retrograde) pyrite.

Figure 7-16: Zinc skarn (manto) in limestone (CAL) with overprinting retrograde alteration.

Source: Challenger Exploration, 2022.

Figure 7-17: Skarn alteration in siltstone with overprinting narrow vein and disseminated mineralisation.

Source: Challenger Exploration, 2021.

Figure 7-18: Low grade mineralisation formed in narrow veins and disseminated pyrite in altered dacite.

Source: Challenger Exploration, 2022.

Pyrite is the dominant retrograde sulphide mineralisation. Pyrite replaces hedenbergite, garnet and sphalerite or occurs at silicate grain boundaries forming irregular masses. Small (10-40 mm diameter) inclusions of gold are observed in thin section, both as tiny inclusions within pyrite and as free gold within hedenbergite (Figure 7-19). Gold is very rarely observed with the naked eye. Visible gold has only been observed in one drill core sample in fresh limestone and one drill core sample in fracture oxidised dacite. Sphalerite does not appear to contain gold but does contain tiny inclusions of chalcopyrite and pyrrhotite.

Figure 7-19: Photomicrograph of Sentazon manto from GNDD009 (116.2 m), (plane polarised reflected light). Two gold grains are evident in pyrite at the contact with hedenbergite. Base of photograph is 0.44 mm.

Source: (Simpson, 2020).

Later retrograde pyrite mineralisation has well-formed cubic crystal faces and is commonly coarser, up to several millimetres. These pyrites are less commonly associated with gold mineralisation.

A detailed study of hydrothermal fluid inclusions trapped in the dacitic intrusions has shown that two hydrothermal fluids were responsible for mineralisation (Bengochea & Mas, 2006):

1. A high salinity (12 wt% NaCl) fluid at temperatures over 300°C is responsible for the primary sulphide mineralisation, calc-silicate alteration, clay (illite) and adularia.
2. A lower temperature, lower salinity fluid (4.5 wt% NaCl), more intense in the northern parts of the deposit is thought responsible for hydrothermal oxidation of the primary mineralisation, formation of silica (jasperoid) veins and retrograde alteration of primary calc-silicate alteration.

The higher salinity fluid may have evolved from a magmatic source distal to the deposit and the lower salinity fluid likely represents evolution to a mixed groundwater and magmatic fluid towards the end of the mineralisation.

Samples of core that have been logged as dacite have been analysed using principal component analysis (PCA) to identify metal associations (metals that have similar principal component orientations). Dacites were analysed separately from other host rock types to reduce influence of host rock on mineralisation. The results of the PCA of dacite samples are summarised in Figure 7-20.

Two groups are observed with geochemical correlations which are interpreted to correspond to two mineral associations.

1) Ag + Cu + Pb + Zn + In. Interpreted as the skarn polymetallic mineralisation; and

2) Au + As + Bi + Sb. Interpreted to represent the later mineralisation with an epithermal affinity.

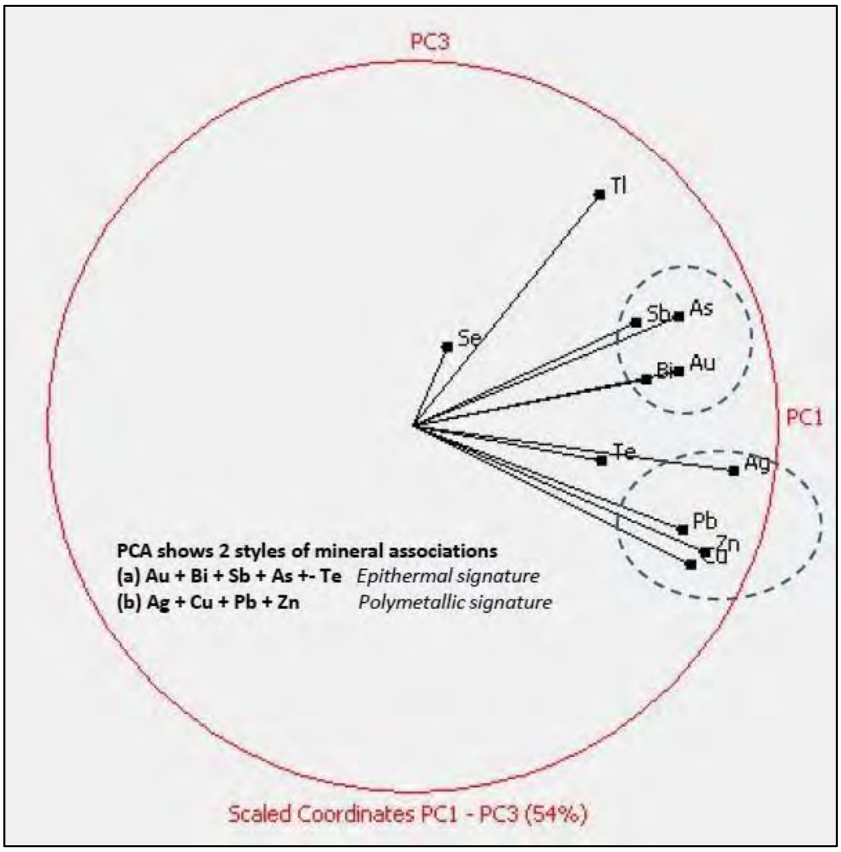


Figure 7-20: Principal Component Analysis (PCA) of the geochemistry of 18,013 samples logged as dacite. Two PCA metal associations are evident which are interpreted as Zinc-Lead-Silver (prograde) and Gold (retrograde).

Source: (Via, 2021).

8 EXPLORATION

8.1 Previous Exploration

Modern exploration started at Hualilan in 1984, when Compañía Minera Aguilar S.A. (Aguilar) completed an exploration program concentrating on Cerro Norte.

From 1984 to 1990, Lixivia S.A. (Lixivia) in joint venture with Aguilar treated some tailings from historical workings and mined mineralised material from the easily accessible areas of Cerro Norte. An unknown volume of material was processed by cyanidation and vat carbon-in-leach. Pre-strip development for open pit mining was undertaken on the Sanchez Fault, along with exploration pitting and trenching. 2,040 metres of RC drilling was completed (16 holes, Table 8-1). Most of the work was contracted to Aguilar.

In 1990, Lixivia formed Alulix S.A. (Alulix) to bring the Cerro Norte mineralisation into production. Work included surveying and geology mapping at surface and underground, channel sampling of mineralised zones (over 200 samples) and geophysical surveying.

Table 8-1: Summary of historic drilling completed at Hualilan from 1984 to 2005. 75 historic drill holes (8,030.4 metres total) have been used for the current MRE. Other holes have proven to have incorrect data or are superseded by CEL drilling.

Date	Company	Drill Type	Number	Total Depth (m)	MRE Number	MRE (03/2023) Total Depth (m)
Jan – 84	Compañía Minera Aguilar S.A.	RC	16	2,040.75	5	608.45
Jan – 95	Plata Mining Ltd (Monarch)	RC	13	1,193.00	0	0.00
1999	Compañía Minera El Colorado S.A.	RC	20*	1,665.00	10	955.00
1999 – 2000	Compañía Minera El Colorado S.A.	DD	60	4,907.30	40	3,477.45
2003 – 2005	La Mancha Resource Inc.	DD	47	7,477.00	20	2,989.50
	Total		**156**	**17,283.05**	**75**	**8,030.40**

*20 drill hole records recovered from a possible total of 32 holes drilled

In 1993, Compañía Minera El Colorado S.A. (CMEC) entered into a purchase option agreement with Alulix. Plata Mining Ltd (Plata), a company listed on the Alberta Stock Exchange, optioned the project from CMEC. In 1995, Plata commissioned an exploration work program at Cerro Norte (Watts, Griffis and McOuat Limited, 1995). Exploration activities included surface mapping, channel sampling of surface trenches and underground workings, 13 RC drill holes for a total of 1,193 metres (Table 8-1) and gold assays of more than 1,500 samples. Also, in the 1990's, Aerodat Inc. conducted an airborne magnetic, resistivity, electromagnetic and radiometric geophysical survey for Monarch Resources Ltd (Monarch), covering an area of 90 km^2 which includes the Hualilan project.

In 1998, a Chilean consulting firm, E-PROM, supervised detailed exploration of the property for Plata. Exploration including surface geological and structural mapping at 1:10,000 and 1:1,000 scales, underground mapping at 1:500 and 1:800 scales, systematic 3 metre interval rock chip channel sampling of many of the known mineralised areas at Cerro Norte, newly discovered structures and adjacent zones and historic tailings stockpiles. A total length of 6 kilometres of underground workings passes through mineralised zones. The development is most extensive at the Cerro Norte between the Pique Ortega shaft and the Dona Justa workings which is a strike length of approximately 1 km. Development extends for approximately 100 m vertically with the deposit dipping northwest at 30°–50°. Other workings at Cerro Sur are less well mapped and sampled. In total, 585 samples were collected and assayed. Seven bulk metallurgical samples were also collected and analysed at the CIMM Tecnologías y Servicios S.A. (CIMM) laboratory in Chile. E-PROM completed resource and reserve estimations using a polygonal estimation method based on the underground samples and available drill hole data. A 320 metre long, 4 × 4 metre production decline was driven by Plata beneath the Main Manto at Cerro Norte with drifts (15 metres and 25 metres) excavated from the main decline.

CMEC assumed active management of the Hualilan project in 1999. CMEC's objective was to better estimate reserves and bring the property into production. To that end, an aggressive program of exploration was completed which included induced polarisation (IP), ground magnetic and electromagnetic geophysical surveys, RC drilling of at least 32 holes, from which records exist for 20 holes (total of 1,665 metres, Table 8-1), metallurgical testing of material at Lakefield Laboratories (cyanidation) and CIMM (flotation), resource and reserve estimation and mining studies. Records are incomplete for some of the exploration and metallurgical testing.

In addition to the RC drilling, 107 diamond holes for total of 12,384 metres were completed between 1999 and 2005. CMEC drilled 60 diamond holes in 1999–2000 for a total of 4,907.3 metres (Table 8-1) and La Mancha Resources Inc. (La Mancha) drilled a further 47 diamond holes (total of 7,477 metres) in 2003–2005 (Table 8-1) (Jenks, 2005). The previous exploration drill data, sampling and assay has been compiled into a drill hole database.

For the historic drilling, it is expected that drilling sample submissions would have included standard reference samples and blanks. No records of these QAQC samples have been recovered and no check assay results have been found in the historic records.

La Mancha contracted Servicio Geológico Minero Argentino (SEGMAR) to complete a surface mapping program which is presented at 1:2,500 scale (Figure 8-1). Review of the earlier mapping by GMSA / CEL has found the mapping to be reliable. GMSA/CEL geologists continue to extend and update the Hualilan Project mapping.

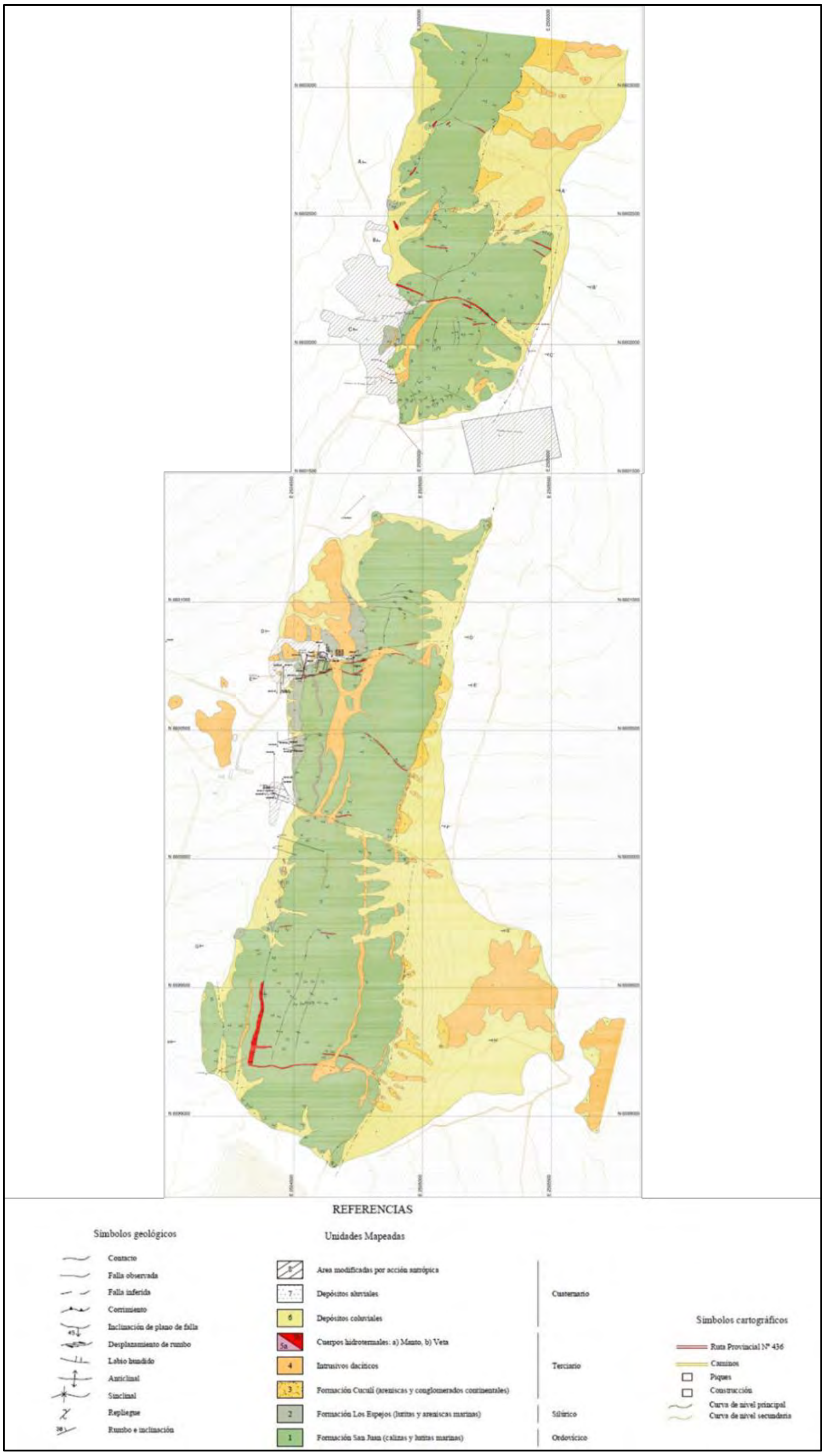


Figure 8-1: Surface mapping

Source: SEGMAR, 2005.

8.2 Challenger Gold Exploration

Challenger Gold Limited (CEL) consolidated the northern and southern mining concessions at Hualilan and started exploration in 2019. Initial check channel sampling and drilling was completed as part of a QAQC program on previous exploration in the absence of documented QAQC from previous explorers. Subsequent consolidation and exploration of the surrounding tenements is ongoing.

8.2.1 Underground and surface channel sampling

In 2019 geological mapping, measurement of structural information and sampling from underground and surface exposures of mineralisation was done to test the grade tenor of mineralisation that was sampled by previous explorers. Sixteen (16) grab samples from mine dumps (both surface and underground), 3 rock chip samples and 51 channel samples (0.4 to 3.0 metres length) were taken. Samples were collected from Bicolor, Norte, Dona Justa, Magnata, Muchilera and Sentazon mantos. The 70 samples average 15.1 g/t Au, 78.5 g/t Ag, 0.06% Pb and 6.5% Zn. The results confirmed the tenor of the underground sampling that was completed previously.

From 2020 – 2022, surface exposures of mineralisation and all accessible underground exposures of mineralisation were safely cleaned, and a more systematic program of channel sampling was completed. Samples were taken by cutting a channel into the rock and chiselling out a continuous representative sample with a weight that is comparable to that which would be collected by ½ cut HQ3 drill core (Figure 8-2). The start and end points of the samples were surveyed so that the locations are precisely known. These channel samples were then able to be compared with results received from nearby drill holes. In total, 1,767 channel samples have been collected across the deposit and are used in the Mineral Resource estimate. The average channel sample is 1.5 metres in length and the average weight of the samples are 5.30 kg.

Figure 8-2: Left: channel sampling limestone in the decline at Norte completed by CMEC in 1999. Right: Channel sampling at the Limestone – siltstone contact at Muchilera..

8.2.2 Historic tailings waste dump sampling

In 2020, 22 samples from historic vat leach tailings waste dumps located at Sentazon, near the processing plant ruins, were systematically collected (Figure 8-3). Samples were analysed by ALS Laboratories using the same technique as was used for the channel sampling. The samples returned an average of 2.1 g/t Au, 22.6 g/t Ag, 0.45% Pb and 3.2% Zn. In 2021 a survey of the dumps was completed using UAV (drone). A topographic surface with a 0.2 m pixel was generated. Assuming a flat base (the dumps are lined), the volume is estimated to be 12,400 m^3. Using an average dry density of 1.6 t/m^3 it is estimated that there are 19,800 t of tailings.

In 2021, a further 9 samples of tailings were collected and analysed at MSA laboratories using the same preparation and assay technique as the tailings samples collected in 2020. These samples averaged 1.8 g/t Au, 21.8 g/t Ag, 0.39% Pb and 3.2% Zn.

The historic tailings do not form part of the Mineral Resource Estimate for Hualilan.

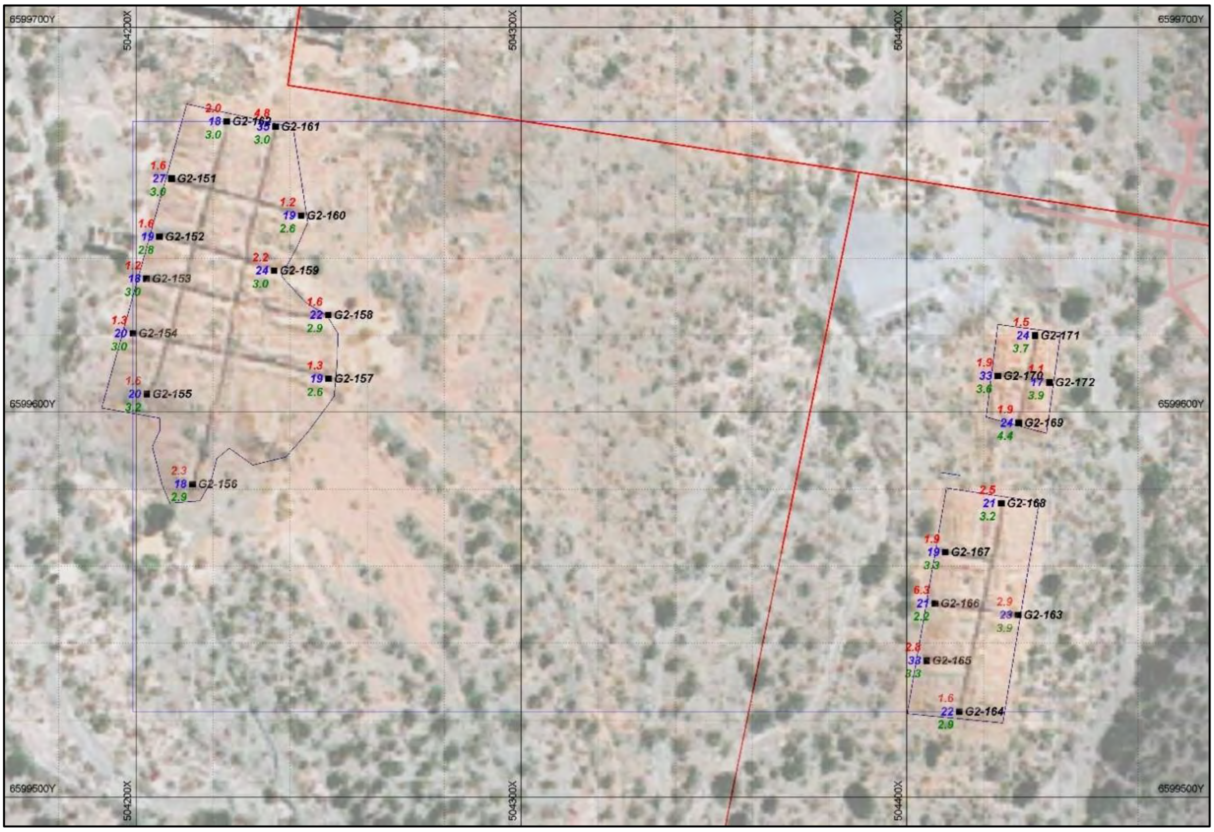


Figure 8-3: Plan of the location of twenty-two samples taken at historic tailings dumps at Hualian in 2020. Results for Au (g/t) are shown in red, Ag (g/t) are shown in blue and Zn (%) are shown in green.

8.2.3 Geophysical surveys

Pole-dipole IP surveys were completed in the northern part of the deposit in 2020 (a total of 8 lines, 115° azimuth, 100 m line separation, 8.35 line kilometres) and in 2021 (a further 8 lines, 090° azimuth, 400 m line separation, 66.3 line kilometres) as shown in Figure 8-4.

The objective of the IP surveys completed in 2020 was to test if the sulphide mineralisation in mantles and disseminated in the siltstones was able to be imaged by the method. The survey used a 50-metre dipole spacing. Lines were separated by 100 m so that the survey could achieve the most detailed response possible. One of the difficulties with the northernmost 6 lines is the limestone hills to the east of the survey lines. The hills were not able to be crossed with the survey lines and so the survey closer to the foot of the hills was not able to image the IP response at depth.

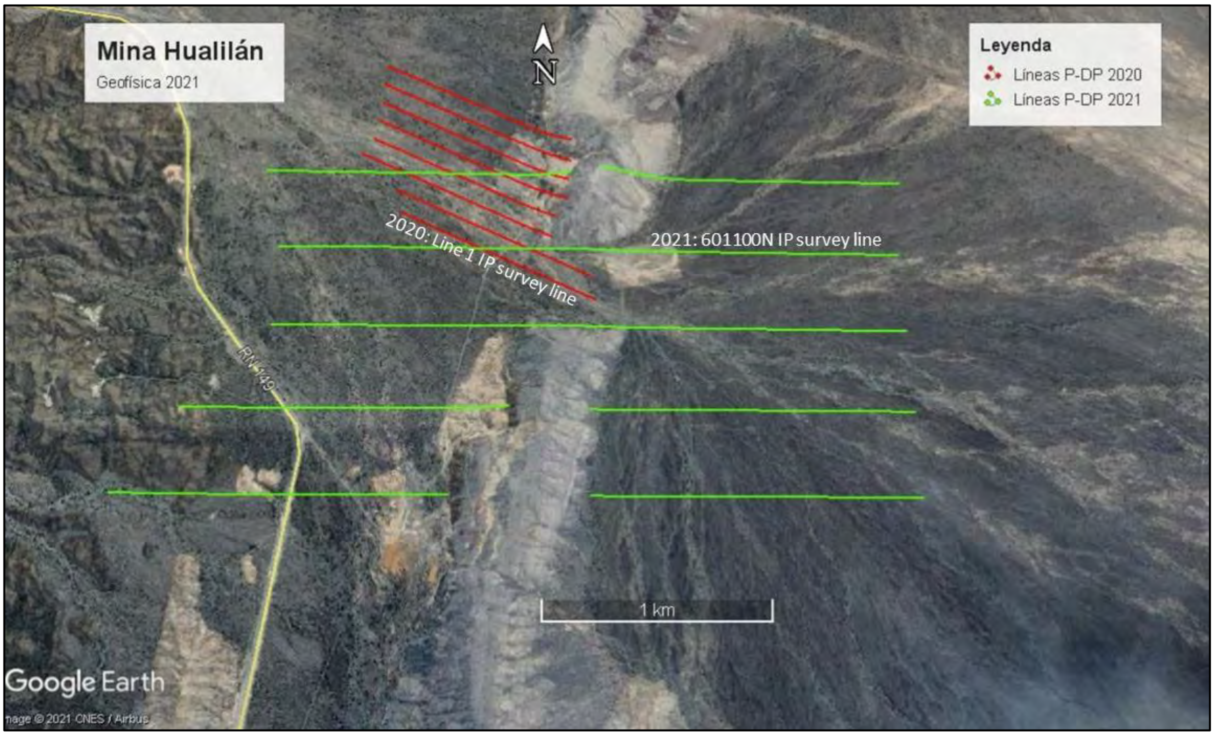


Figure 8-4: Location of IP survey lines.

The two southern-most 2020 lines pass through the gap between the limestone hills and provide the most complete cross section through the deposit (Figure 8-5). The observed resistivity and chargeability response reflects the location of the mineralisation at a deposit scale, but lacks the detail required to make the IP response a useful tool for detailed drill exploration targeting.

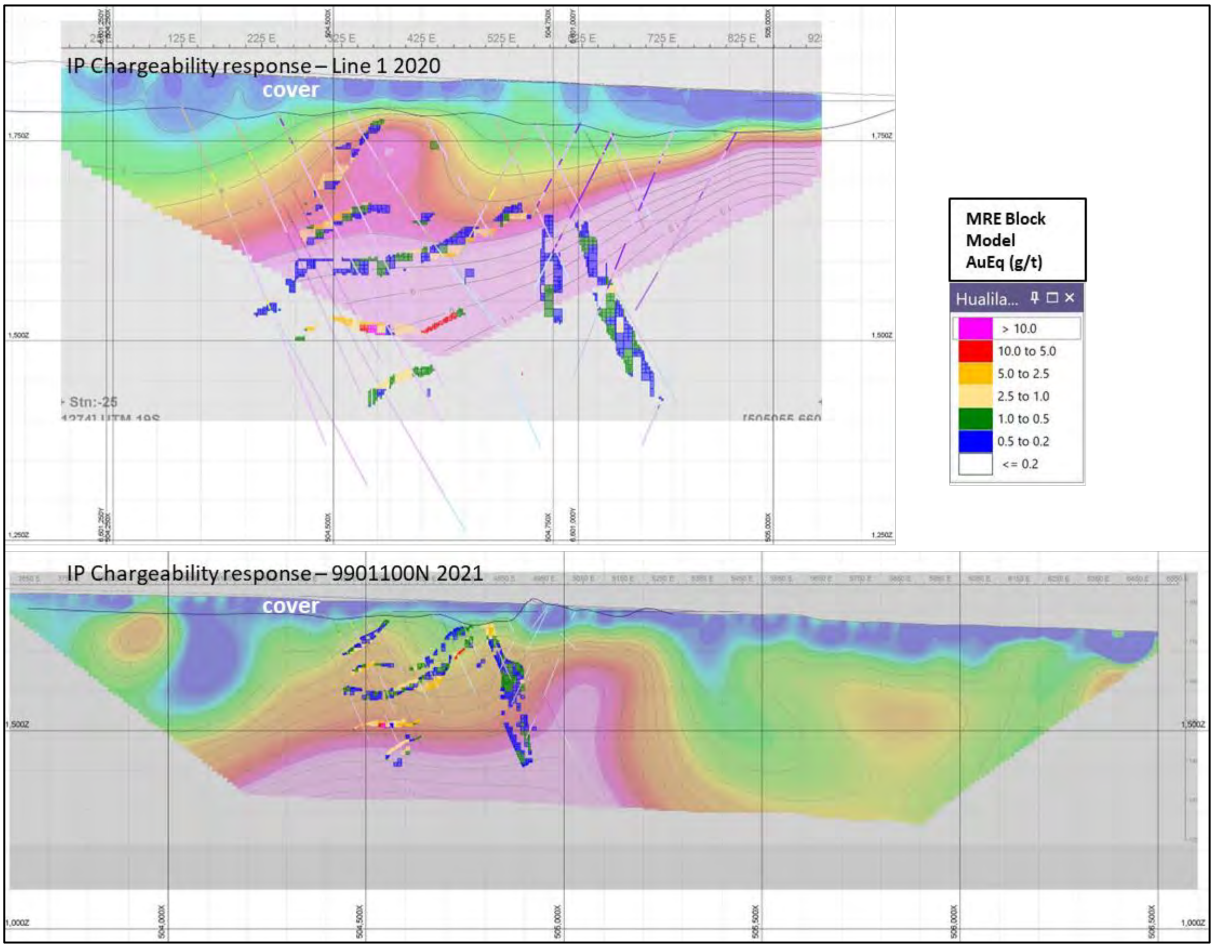


Figure 8-5: IP chargeability sections in the Gap area from 2020 and 2021 surveys with drill hole geology and 2023 MRE block model. The IP method is not able to define the detail of the mineralisation.

Source: Challenger Exploration, 2023.

In 2021, longer lines with a 090° azimuth, separated by 400 metres with a 50 metre pole-dipole spacing were used to test the IP response to larger targets (potentially lower grade, intrusion-hosted, disseminated mineralisation). The IP chargeability response was able to image the deposit at a broad scale to a greater depth with the longer lines (Figure 8-5).

Some of the larger IP chargeability anomalies that were drill tested failed to identify significant mineralisation. It is possible that the cover material at Hualilan is diminishing the IP response from bedrock and reducing the resolution of the survey.

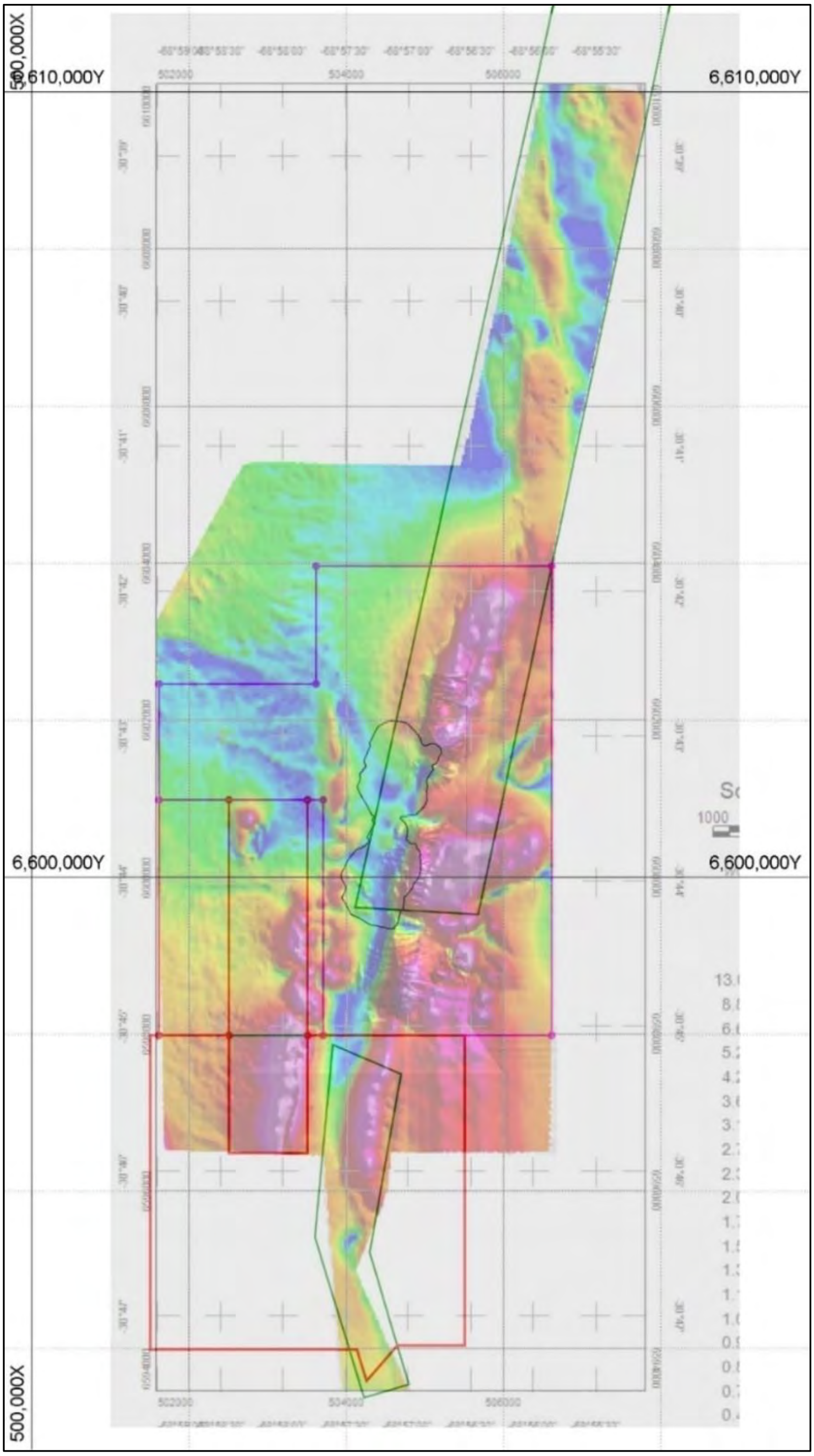


Figure 8-6: MVA from combined ground magnetic surveys with surface limit of the March 2023 MRE surface shell ($1,800) and lease boundaries.

Source: Challenger Exploration, 2023.

A ground magnetic survey over the central part of the Hualilan Project was completed in 2020. The survey consisted of east-west survey lines spaced 40 metre apart for a total of 379.4 line kilometres including tie lines. Surveying across the Hualilan Hills was not possible due to the steep terrain.

The ground magnetic survey was extended in all directions in 2021. The same 40 metre east-west line spacing was used. In addition to the extension, lines were walked along the ridges of the Hualilan hills to cover the areas where the terrain prevented surveying on E-W lines. The extension survey collected an additional 863.2 line kilometres including tie lines.

Magnetic grids of TMI, RTP, 1VD of the RTP, Analytic signal of the RTP and a tilt derivative of the RTP were produced. In addition, CEL contracted a magnetic vector amplitude (MVA) enhancement and derived grids from a newly micro levelled TMI grid (Figure 8-6). The MVA and component grids from this processing are being used to interpret the geology below the cover.

8.3 Drilling

CEL/ GMSA has completed 799 drill holes (total of 227,103.60 metres of drilling) that has been included in the March 2023 MRE (Table 8-2). Five drilling companies were contracted to complete the drilling. Diamond core drilling collects HQ3 core. In rare cases, holes have been completed by reducing to NQ3.

Table 8-2: Summary of drilling completed by CEL/GMSA from October 2019 – January 2022.

Date	Company	Drill Type	Number	Total Depth (m)
Oct – Nov 2019	CEL / GMSA	DD	13	1,471.80
Feb – Nov 2020	CEL / GMSA	DD	131	28,017.70
Mar – Sept 2020	CEL / GMSA	RC	37	2,923.00
Jan – Dec 2021	CEL / GMSA	DD	321	96,481.80
Jan – Dec 2022	CEL / GMSA	DD	293	96,321.30
Jan 2023 *	CEL / GMSA	DD	4	1,888.00
	Total		**799**	**227,103.60**

*to GNDD790. Drilling has continued past the January 2022 cut-off date for the March 2023 MRE

Of the 224,180.60 metres of diamond core drilling, 22,041.30 metres (9.8%) is drilled through cover where no sample is taken. Drilling through cover is done using tricone to resistance and then with HQ until competent bed rock is reached that is capable of seating stable casing. The casing is used to prevent the cover from collapsing into the hole during drilling. At the end of the drilling, the casing is recovered and 1 – 2 metres of PVC tube with a PVC cap is placed into the hole and the collar is cemented around the PVC to mark the location of the hole.

Of the 202,139.30 metres of core drilling below the base of cover, recovery is 195,650.07 metres (96.8%). All core that is recovered is marked up for recovery and Rock

Quality Designation (RQD), logged and marked up for sampling before being photographed and sampled.

RC drilling was completed using a LM90 drill rig with a side-mounted cyclone to collect drill cuttings and to split off a 3-4 kg sample and duplicate sample for every metre of drilling. Drill cuttings were collected for each metre drilled. Cuttings recovered were weighed to check sample recovery and the cuttings then logged. The samples that are logged as being below the base of cover were submitted for preparation and assay.

8.4 Surface Mapping and Geological Model

Surface exposure is concentrated on the limestone that forms the main trend of the Hualilan Hills, and on smaller hills of dacite and limited siltstone and sandstone where exposure is near the contact with the limestone and dacite exposure. Surface mapping completed by SEGMAR for La Mancha Resources Inc. is reliable (Figure 8-1). The mapping is being updated and extended by GMSA/CEL with additional detail and regular rock chip and soil sampling. Detail of minor structures (faults and veins) and some narrow manto zones have been added to the previous maps. Additional near mine drill targets have been identified from recent detailed mapping and rock chip sampling.

The surface mapping is being digitised and added to the 3D Geological Model of the resource area (Figure 8-7).

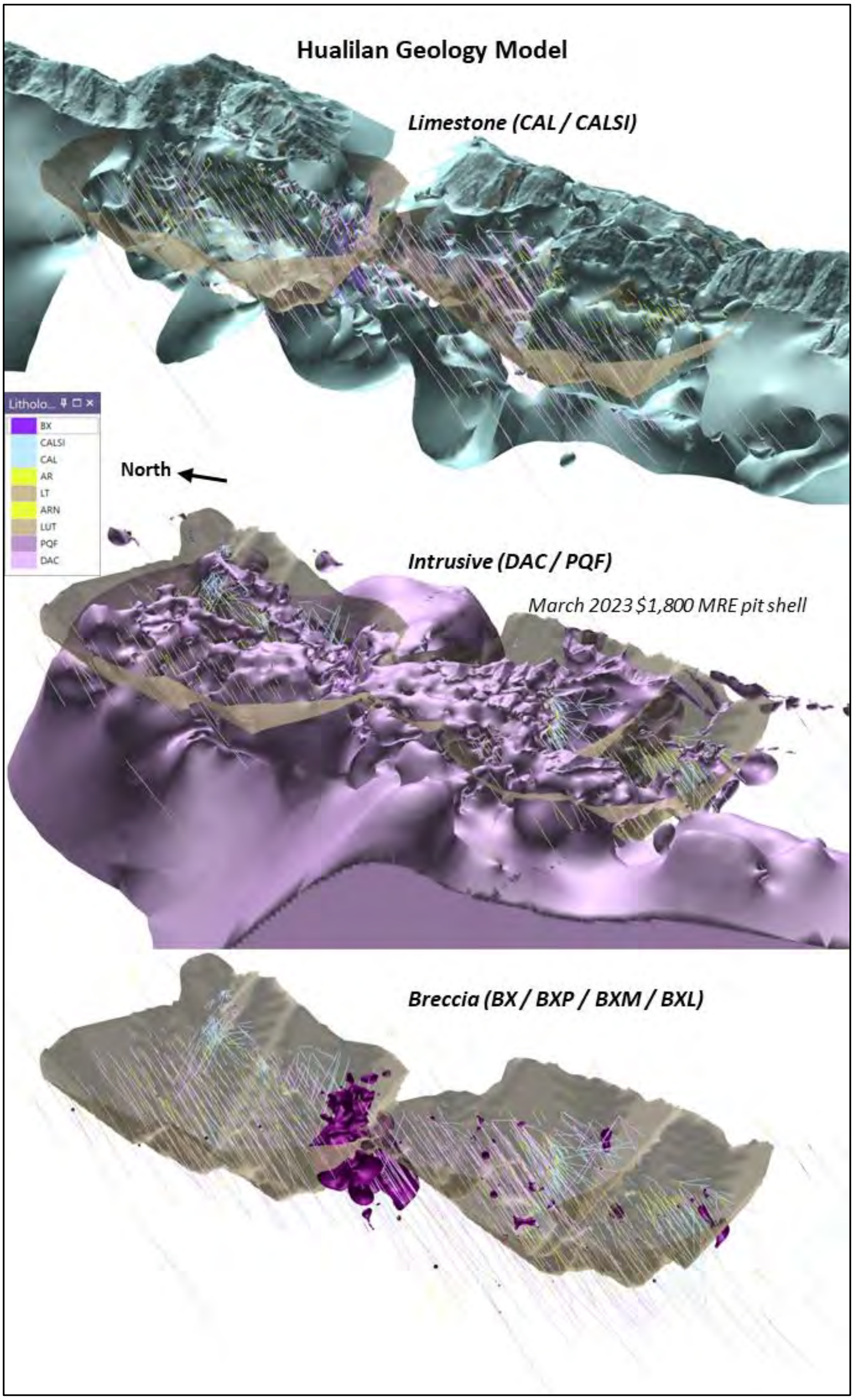


Figure 8-7: Geology model volumes for Hualilan with March 2023 MRE pit shell ($1,800). Geology model has been generated from surface and drill hole data in Leapfrog.

Source: Challenger Exploration, 2023.

8.5 Next Steps

The Priority Exploration Targets (Figure 8-8) are:

1. Along strike to the north where the near surface target is a north-west extension from the Verde-Sanchez area towards Marta Alicia, on the north side of the Sanchez Fault. Current drilling shows there to be a westward fault displacement of the sedimentary sequence.
2. Extending deeper targets that could support underground mining. These include Verde, Magnata Fault Zone and Muchilera – Sentazon.
3. Near-mine targets along strike to the south of Hualilan, including at Solitario. In this area there are surface indications of alteration and mineralisation, similar to Hualilan and there is a similar MVA low magnetic response (possibly alteration).
4. Low magnetic responses (interpreted to be alteration) to the east of Hualilan in the Cucili Formation under cover).
5. Additional targets within 10 kilometres of the Hualilan project, including immediately west of the Ayen Cateo, north-west (Vera Cruz) and NNW (El Peñon).

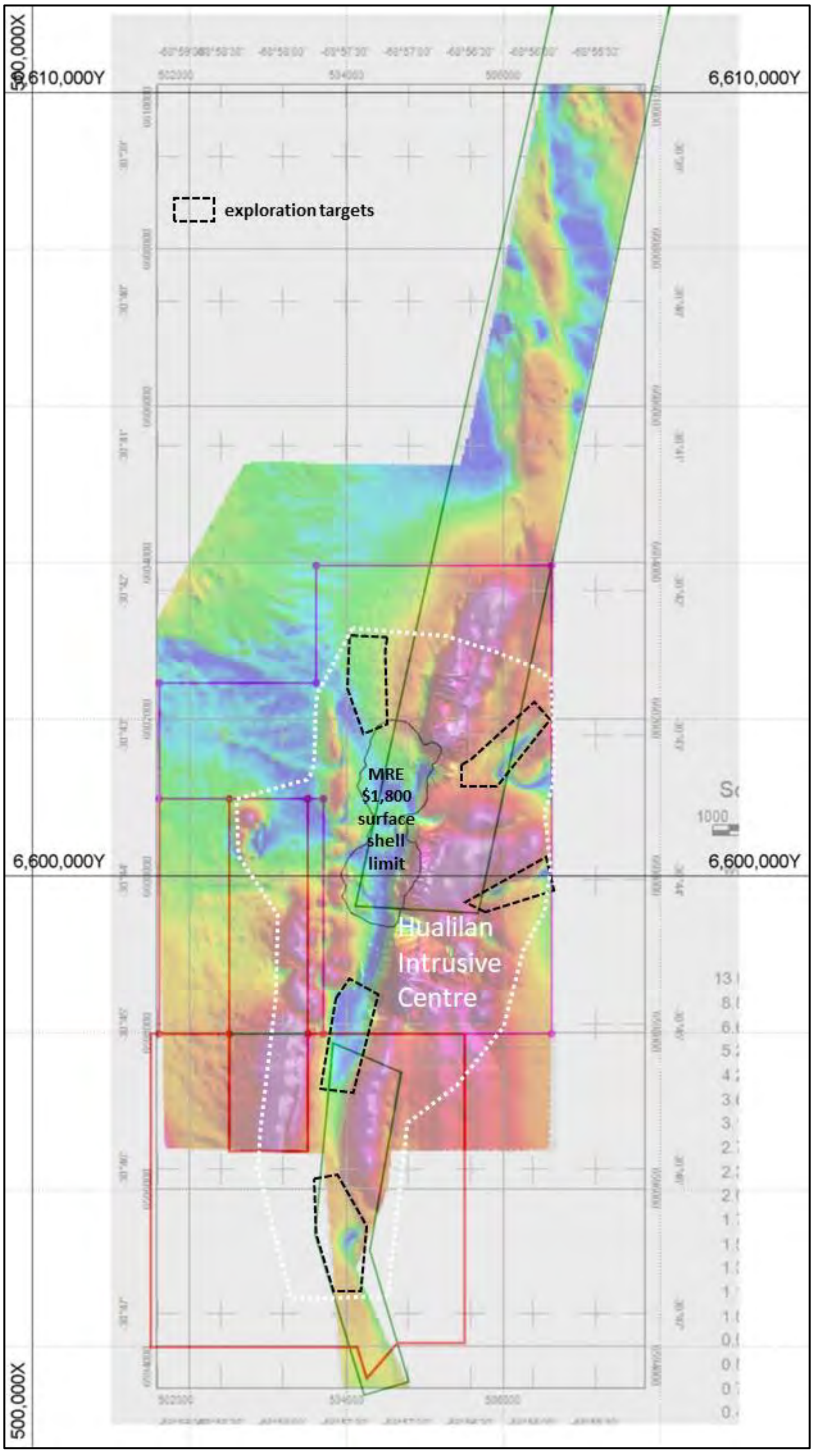


Figure 8-8: MVA from combined magnetic surveys with surface extent of MRE surface shell and near mine exploration targets highlighted by magnetic lows (alteration) within the Hualilan intrusive complex.

Source: Challenger Exploration, 2023.

9 SAMPLE PREPARATION, ANALYSES AND SECURITY

9.1 Sample Preparation

Three sample types have been used in the Hualilan MRE. GMSA/CEL drilled diamond core samples, reverse circulation drilling chips and channel cut samples weighing 2 – 7 kilograms per sample were collected into plastic bags over their respective sample intervals. Each plastic bag is marked with metres from, metres to, and sample ID. A sample ID tag is also attached to the bag and a tag is also inserted into each bag for cross-checking by the laboratory. The bags were sealed with a cable ties. Samples were then placed into polyweave bags at the rate of 4-6 samples per bag. Written on each polyweave bag was the sample IDs of the contained bags. Individual sample preparation in detailed below.

There is little information provided by previous explorers to detail sampling techniques of historic drill core or reverse circulation samples.

9.1.1 Reverse Circulation Drill Holes – RC

Reverse circulation drilling collected samples over 1 metre intervals from the collar to end of hole. Drill chips from each 1 m were collected and the sample weighed to check sample recovery. Sub-samples of weight 2-4 kilograms for each 1 metre interval were collected from a cyclone mounted on the drill machine. The weight of the sub-sample was also recorded. Each 1 m interval was logged for lithology, alteration and mineralisation by the geologist at the drill rig. 1 m interval samples that were below the cover were set aside for assay. Duplicate RC samples have been collected by taking a 2-4 kg split using a riffle splitter. Duplicate samples have been taken at the rate of 1 duplicate for every 25-30 metres drilled. Blank samples were included at the rate of 1 blank for every 30 metres drilled. One Certified Reference Material (CRM) pulp sample was included in the sample sequence for every 30-40 metres drilled.

9.1.2 Diamond Drill Holes – DDH

Diamond core (HQ3 and NQ3) was collected at the drill rig from the triple tube splits and placed into wooden boxes by the drill crew. The drill crew marked the start and end of each core run with a wooden block that has the drill hole number (hole ID) and the depth clearly marked on the block. Sequentially numbered boxes were marked with the hole ID.

The drill core was transported the short distance to the preparation and logging camp at the end of each drilling shift. The core was checked for correct numbering and a quick log of the core was done to highlight any interesting geological features that require immediate attention.

The core was then logged for recovery and RQD. Where recovery was less than 100%, a block was added to the tray to indicate the location metreage of core loss. The boxes were marked with the start and end metreage.

The core was then logged for geology, alteration, mineralisation and structure. The structural log is limited to observations and alpha angle measurement as the core was not oriented. The geologist marked the core with the intervals that were to be sampled. Sample intervals were selected according to lithology, alteration, and mineralisation contacts. All the meterage that was recovered was sampled. The cover was not sampled. Marked on the core boxes were the locations where duplicate samples, certified reference material (CRM) and blank samples were taken. From drill hole GNDD073, duplicate core samples consisted of two ¼ core samples over the same interval. Duplicate or blank samples were taken at the rate of 1 duplicate or blank for approximately 30 samples taken. One CRM pulp sample was included in the sample sequence for every 30-40 samples taken.

The logged core with recovery and sampling intervals was then photographed before sampling.

Drill core was cut longitudinally on site using a diamond saw or split using a hand operated hydraulic core splitter where lower grade is expected. Sample lengths were generally from 0.5 m to 2.0 m in length (average 1.8 m).

9.1.3 Channel Samples

Channel samples from surface exposures (mostly along track cuttings) and from accessible underground exposures have been cut into the exposure using a hand-held diamond edged circular blade cutting tool. Parallel saw cuts 3-5 cm apart were made which were 2-4 cm deep into the rock. This has allowed for the extraction of a representative sample using a hammer and chisel. The sample was collected onto a plastic mat then collected into a sample bag. Sample intervals were determined by the geologist logging the exposure. Sample intervals vary from 0.5 m to 2.0 m. The exposure was logged for lithology, alteration, mineralisation and structure. Duplicate or blank samples were taken at the rate of 1 duplicate or blank for approximately 30 samples taken. One CRM pulp sample was included in the sample sequence for every 30-40 samples taken.

9.2 Analyses

Core, RC and channel samples have been prepared and analysed by three different laboratories using the same preparation and assay techniques.

The three laboratories are:

- MSA Laboratories – preparation by MSA / QLabs in San Juan, Argentina, pulp transport and assay at MSA Laboratories in Langley, BC, Canada.

- ALS Laboratories – preparation by ALS in Mendoza, Argentina, pulp transport and assay at ALS Laboratories in Lima, Peru or at ALS Laboratories in Vancouver, BC, Canada.
- SGS Laboratories – Preparation by SGS Laboratory in San Juan, Argentina, pulp transport and assay at SGS Laboratory in Lima, Peru.

Core, RC and channel samples, including blanks and duplicates were crushed to approximately 85% passing 2 mm. A 500 g or a 1 kg sub-sample was taken and pulverized to 85% passing 75 µm. A 50 g charge was analysed for Au by fire assay with AA determination. Where the fire assay grade is >10 g/t gold, a 50 g charge was analysed for Au by Fire assay with gravimetric determination. A 10 g charge was analysed for at least 48 elements by 4-acid digest and ICP-MS determination. Elements determined were Ag, As, Ba, Be, Bi, Ca, 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 and Zr. For overlimit Ag >100 g/t, Zn, Pb and Cu >10,000 ppm and S >10%, analysis was done by the same method using a different calibration.

Unused pulp has been returned from the laboratories to the Hualilan project and is stored in a dry secure location (shipping containers), so they are available for any further analyses. Remaining drill core is stored under cover for reference and future use as required.

9.3 Security

On-site security at Hualilan was established in 2019 before the initial drilling by CEL and has been continued throughout the program. At the end of each drill shift and at the end of each working day, samples were taken to the exploration camp which is under security control.

At the exploration camp, samples in numbered bags were photographed before they were placed into polyweave bags for transport to the laboratory. Transport to the laboratory was via commercially operated truck or CEL operated light vehicle depending on the total weight of the submission. Each submission was accompanied by a sample submission form completed at Hualilan by the supervising geologist and given to the driver to pass on to the laboratory on arrival in San Juan or Mendoza. An email was sent to the laboratory supervisor with the sample submission sheets attached prior to arrival of the samples. Samples that arrived at the laboratory were checked on delivery and the submission sheet was countersigned by the laboratory. A copy of the countersigned submission sheets is kept at the Hualilan project.

Unsampled ½ HQ Drill Core and representative Reverse Circulation chips are stored at covered core storage sheds near the exploration camp.

On completion of the sample analysis, coarse rejects and pulp rejects were returned to the Hualilan exploration camp. Coarse rejects were held on site for approximately 1 year in case any repeat assay was required and disposed of at Hualilan after that time. Pulp rejects are stored in paper packets numbered with the sample ID and laboratory job number, packed into boxes, labelled and shelved in shipping containers on site for any future repeat analysis or alternative analysis.

9.4 QA/QC

9.4.1 Analysis of Duplicate Samples

Duplicate samples from diamond drill core were taken after GNDD072 in the form of two ¼ core samples over the same interval (Figure 9-1, Table 9-1).

Table 9-1: Duplicate ¼ core sample assay results (2021-2023).

Element	Count	RSQ	Average		Median		Variance	
			Original	Duplicate	Original	Duplicate	Original	Duplicate
Au (ppm)	3,523	0.960	0.076	0.077	0.007	0.006	0.640	0.816
Ag (ppm)	3,523	0.696	0.53	0.48	0.17	0.16	7.99	3.55
Fe (%)	3,523	0.990	1.997	1.996	1.700	1.710	3.74	3.75
Pb (ppm)	3,523	0.940	64.7	62.4	13.7	13.4	1.9E+05	2.7E+05
S (%)	3,523	0.973	0.333	0.330	0.140	0.140	0.346	0.332
Zn (ppm)	3,523	0.976	254	243	73	72	3.8E+06	3.5E+06

RSQ = R squared coefficient

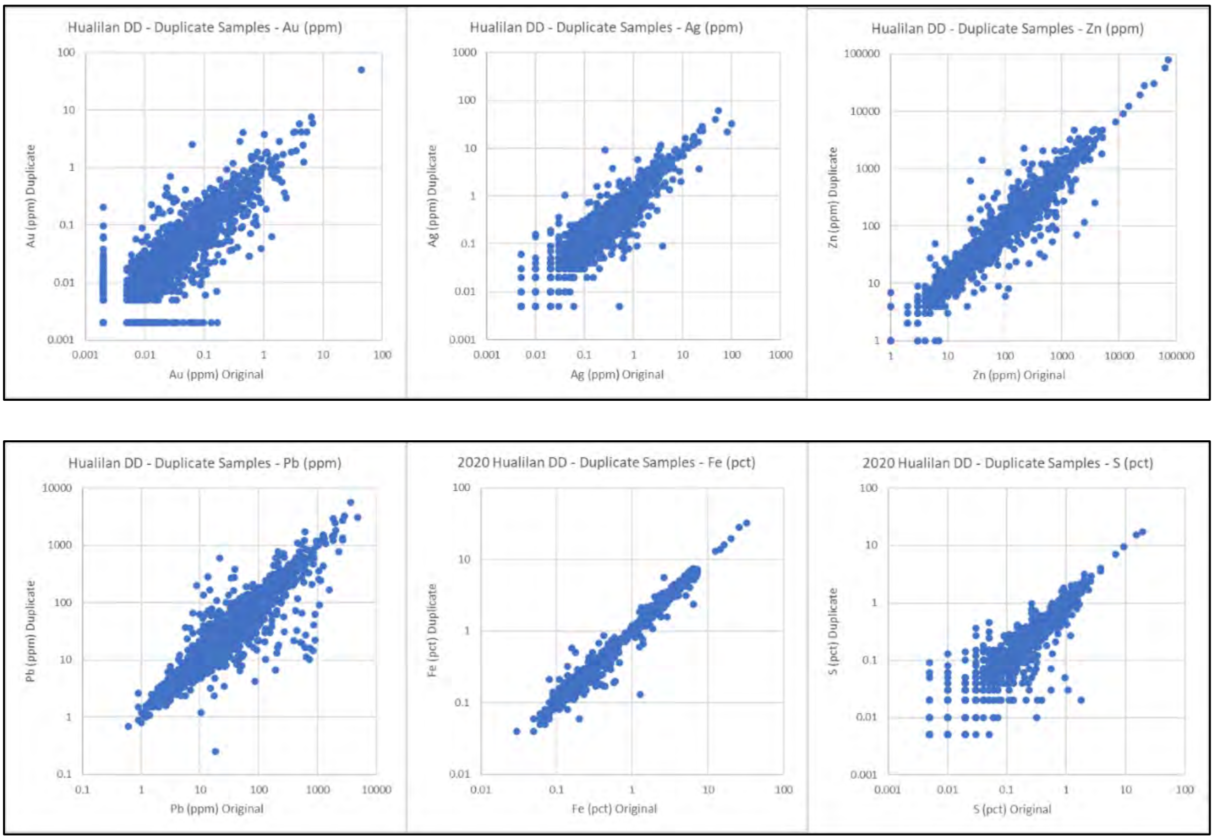


Figure 9-1: Duplicate analyses (log scale for Au, Ag, Zn, Pb, Fe and S) for all laboratories for ¼ core from diamond drill samples (2021-2023).

Duplicate RC samples were taken from all holes completed. Duplicate sample results and correlation plots are shown in Table 9-2, Figure 9-2.

Table 9-2: Field duplicate RC sample assay results

Element	Count	RSQ	Average		Median		Variance	
			Original	Duplicate	Original	Duplicate	Original	Duplicate
Au (ppm)	85	0.799	0.101	0.140	0.017	0.016	0.041	0.115
Ag (ppm)	85	0.691	1.74	2.43	0.59	0.58	13.59	64.29
Fe (%)	85	0.997	1.470	1.503	0.450	0.410	7.6	7.6
Pb (ppm)	85	0.887	296.0	350.6	26.3	32.4	6.0E+05	7.4E+05
S (%)	85	0.972	0.113	0.126	0.020	0.020	0.046	0.062
Zn (ppm)	85	0.977	3399	3234	158	177	2.5.E+08	2.1.E+08

RSQ = R squared coefficient

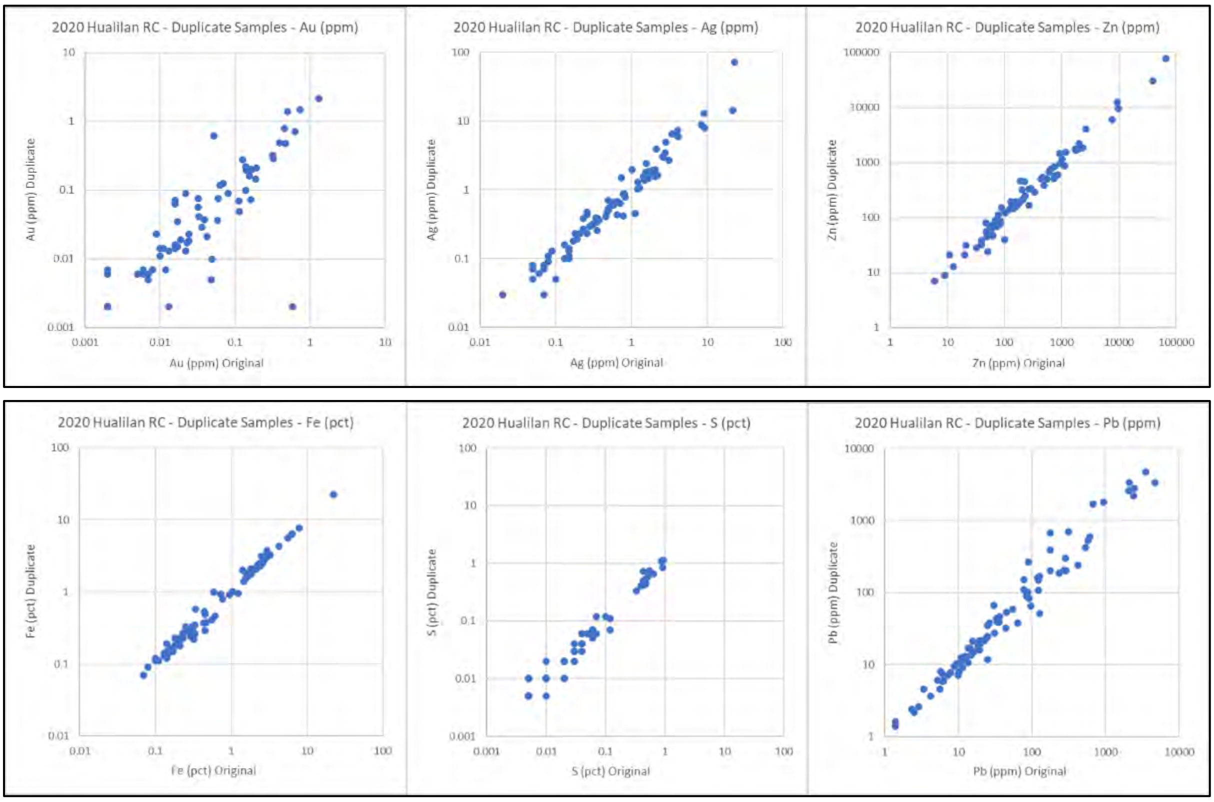


Figure 9-2: Duplicate analyses (log scale for Au, Ag, Zn, Pb, Fe and S) for RC samples.

Duplicate channel sample assays have been collected from the underground and surface sampling program (Table 9-3, Figure 9-3). These data show more scatter due to the impact of weathering.

Table 9-3: Duplicate surface and underground channel sample results

Element	Count	RSQ	Average		Median		Variance	
			Original	Duplicate	Original	Duplicate	Original	Duplicate
Au (ppm)	45	0.296	1.211	2.025	0.042	0.039	8.988	23.498

Element	Count	RSQ	Average		Median		Variance	
Ag (ppm)	45	0.037	8.42	23.25	1.09	1.22	177.31	3990.47
Fe (%)	45	0.373	124.23	77.85	7.54	7.80	61687.10	26171.51
Pb (ppm)	45	0.476	713.23	802.79	46.20	37.40	2.8E+06	3.0E+06
S (%)	45	0.428	4.266	5.745	1.390	1.560	44.4	107.0
Zn (ppm)	45	0.007	955.4	3776.0	75.3	60.7	3.5E+06	3.0E+08

RSQ = R squared coefficient

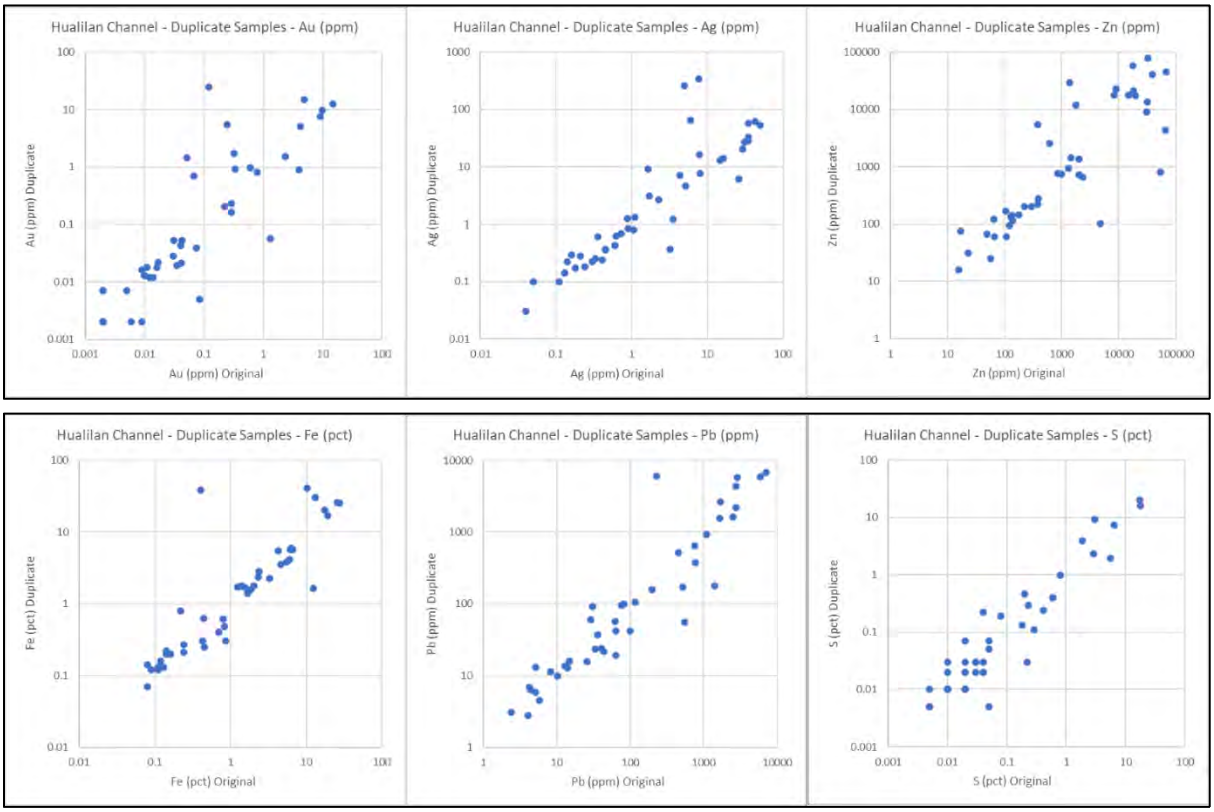


Figure 9-3: Duplicate analyses from all laboratories (log scale for Au, Ag, Zn, Pb, Fe and S) for surface and underground channel samples.

9.4.2 Analysis of Blank Samples

CEL have used two different blank samples (Figure 9-4, Figure 9-5 and Figure 9-6), submitted with drill core, RC chips and channel samples and subjected to the same preparation and assay procedures. The blank samples are sourced from either:

1. Surface gravels in the Las Flores area of San Juan (798 analyses from ALS, 1631 from MSA, 63 from SGS)
2. from a commercial dolomite quarry near San Juan (282 analyses from ALS, 1556 from MSA).

The blank samples were strategically placed in the sample sequence immediately after samples that were suspected of containing higher grades (high sulphide content). This was done to test the lab preparation procedures, particularly regarding cleaning the crushing and grinding equipment between samples. The values received from the blank samples suggest contamination during preparation was rare.

Generally, the gravel from Las Flores has more internal chemical variability than the dolomite from San Juan.

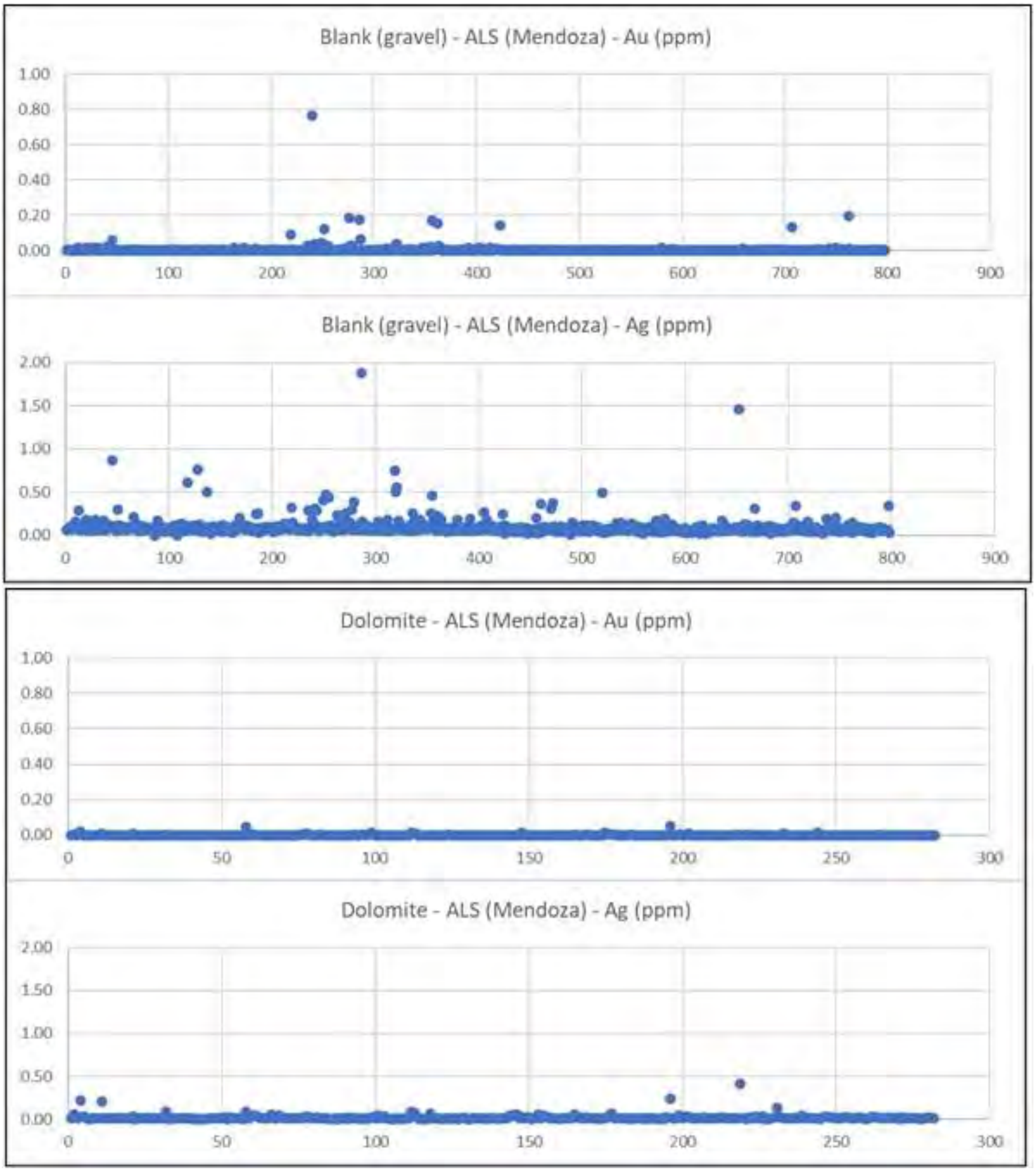


Figure 9-4: Analysis of Au and Ag from blank samples submitted to ALS laboratories.

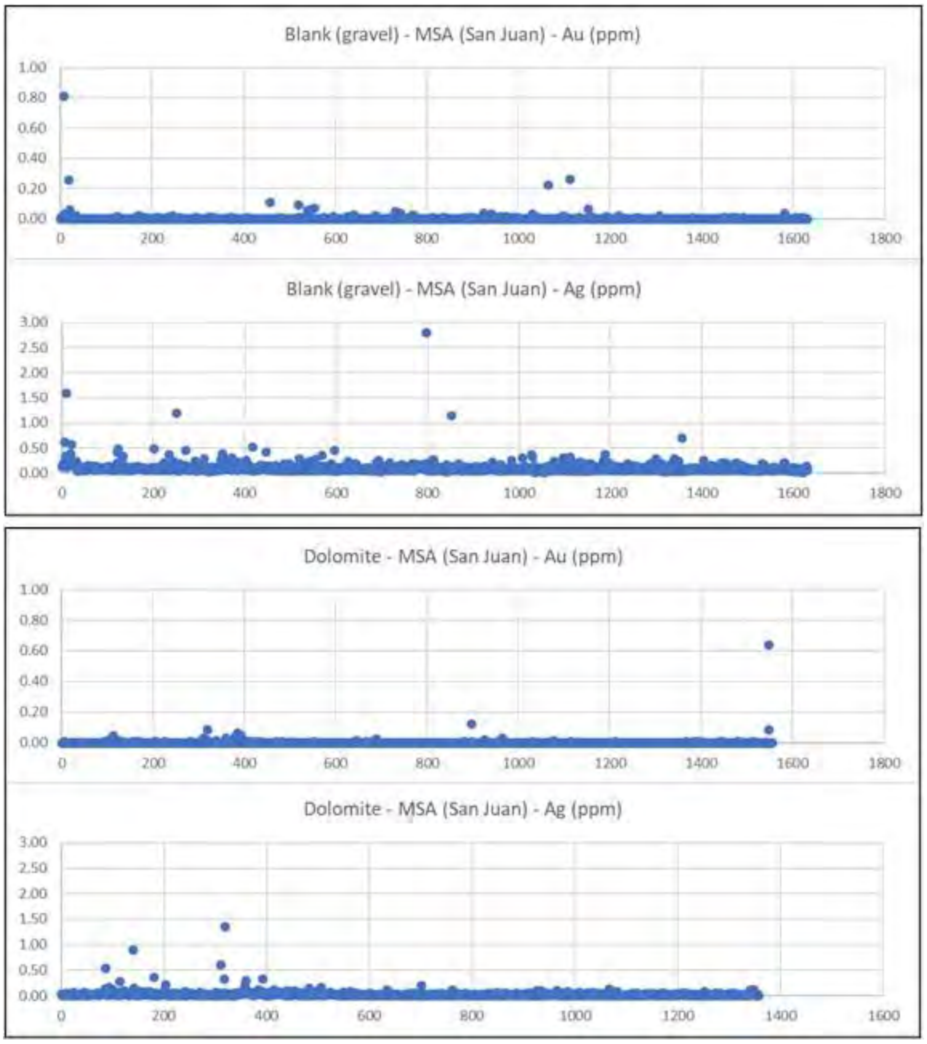


Figure 9-5: Analysis of Au and Ag from blank samples submitted to MSA laboratories.

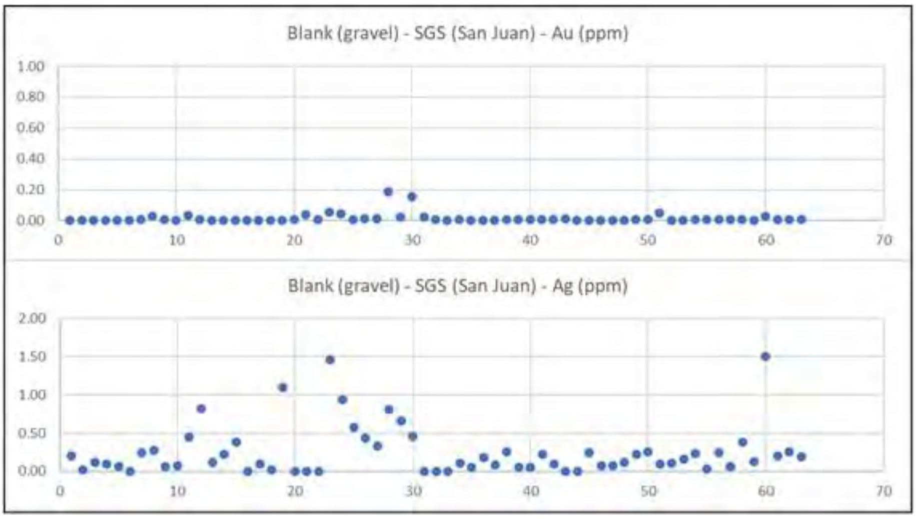


Figure 9-6: Analysis of Au and Ag from blank samples submitted to SGS laboratories.

9.4.3 Analysis of Certified Reference Material (CRM)

Commercially sourced CRM in the form of a 30 gram pulp was submitted in the sample sequences with drill core, RC and channel samples. The CRM has a known, certified quantity of key elements which are tested by the fire assay and/or 4-acid digest ICP MS techniques used to analyse samples from Hualilan.

Selected sample intervals from drill holes GNDD001 to GNDD010, completed in 2019, were analysed by MSA. Three different CRM pulp samples sourced from CDN Resource Laboratories Ltd. with known values for Au, Ag, Pb, Cu and Zn were submitted with samples of drill core to test the precision and accuracy of the analytic procedures at MSA laboratory in Canada. A total of 26 reference analyses were analysed in the samples submitted in 2019 (Table 9-4).

Table 9-4: CRM used for 2019 drill core samples

CRM ID	Code	Count ALS	Fire Assay Certification	4-acid ICPMS Certification
01	CDN-GS-5T	4	Au	Ag
02	CDN-GS-5W	4	Au	-
03	CDN-ME-1704	18	Au	Ag, Cu, Pb, Zn

The results for CRM 03 are shown in Figure 9-7. All results returned were within 2 standard deviations of the expected value for Au, Ag and Zn. The standards demonstrate suitable precision and accuracy of the analytic process. No systematic bias is observed.

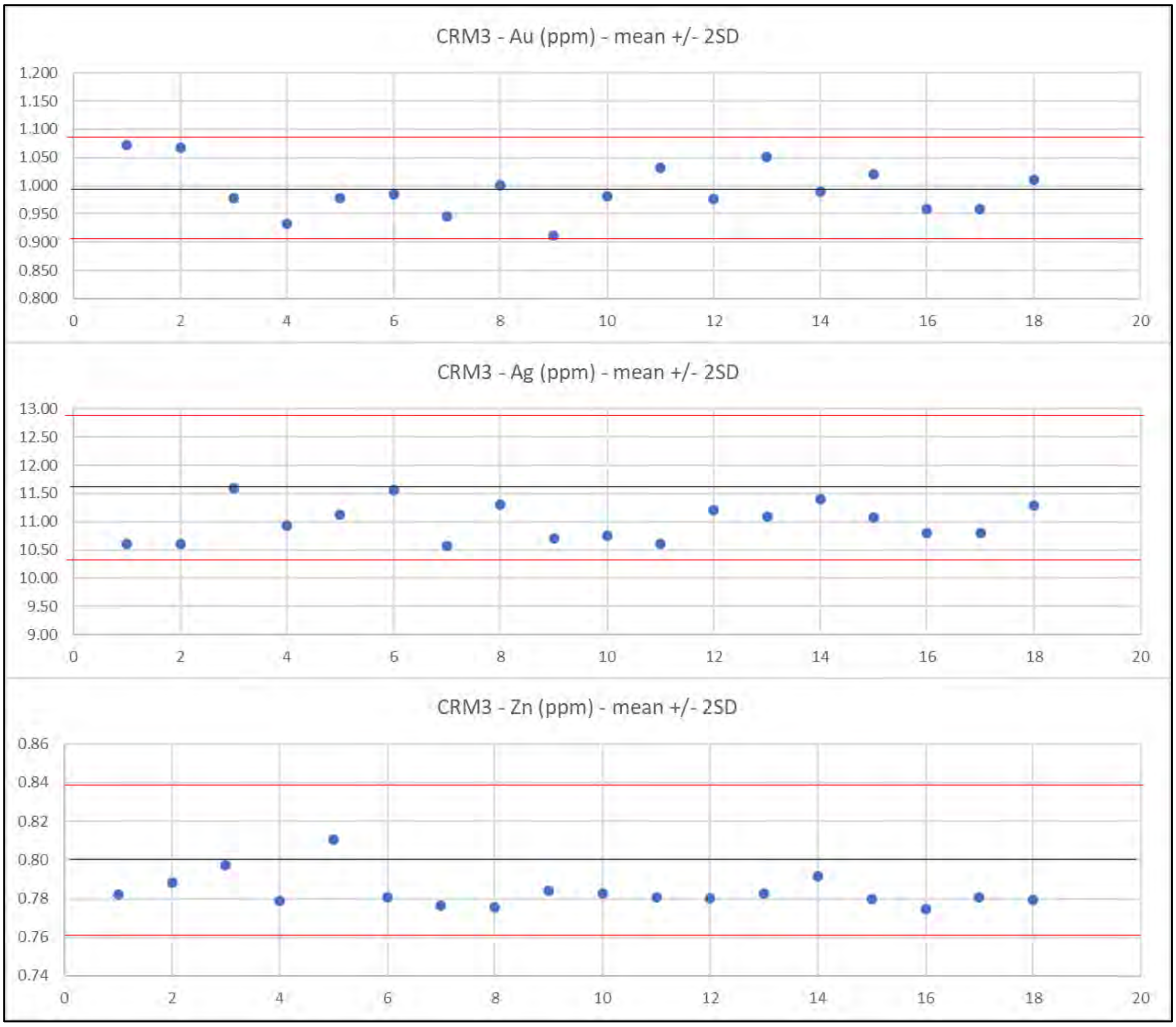


Figure 9-7: Results of analysis of CRM 03 (CDN-ME-1704) at MSA laboratories.

An additional 24 CRM pulps have been submitted with drill core, RC and channel samples for the sampling from 2020 to early 2023 (Table 9-5). These CRM have been sourced from OREAS (n=3,791), INTEM (n=187) and GeoStats (n=342).

Table 9-5: CRM used for 2020-2023 drill core, RC and channel samples

CRM ID	Code	Count ALS	Count MSA	Count SGS	Fire Assay Certification	4-acid ICPMS Certification
04	OREAS603b	57	34	-	Au	All elements
05	OREAS610	50	44	-	Au	All elements
06	OREAS623	52	42	-	Au	All elements
07	OREAS600	41	62	-	Au	All elements
08	OREAS608	78	211	-	Au	All elements
09	OREAS609	90	164	27	Au	All elements
10	IN-M340-171	6	22	-	Au	Ag, Cu, Fe, Pb, Zn
11	IN-M405-187	29	50	-	Au	Ag, Cu, Pb, Zn
12	IN-M425-188	30	50	-	Au	Ag, Cu, Pb, Zn
13	OREAS607	64	98	19	Au	All elements
14	OREAS60d	46	30	21	Au	All elements
15	OREAS600b	-	145	-	Au	All elements

CRM ID	Code	Count ALS	Count MSA	Count SGS	Fire Assay Certification	4-acid ICPMS Certification
16	G307-8	7	34	-	Au	-
17	G310-6	5	52	-	Au	-
18	G907-2	2	33	-	Au	-
19	G909-10	9	49	-	Au	-
20	G909-7	8	48	-	Au	-
21	G910-2	4	43	-	Au	-
22	G910-6	8	40	-	Au	-
23	OREAS240	180	509	-	Au	All elements
24	OREAS254b	172	522	-	Au	All elements
25	OREAS233	152	459	-	Au	All elements
26	OREAS606	-	214	-	Au	All elements
27	OREAS264	81	127	-	Au	All elements
Total		**1,171**	**3,082**	**67**		

The CRM sourced from INTEM was found to be unreliable and so use of this material was discontinued after the first results were received.

Most of the CRM that has been used was sourced from OREAS (Figure 9-8 and Appendix 2 – QAQC Charts). The OREAS CRM is certified for Fire assay for gold and certified for 4-acid digest ICP for all other elements analysed. Failure rates (defined as outside ±2 standard deviations) from ALS and MSA laboratories are low and there is no bias observed in the results. The OREAS CRM that was submitted to SGS laboratories was returned with a high variability and a higher failure rate. As a result, the use of SGS laboratories was discontinued after the first results were received.

The CRM that was sourced from INTEM (Figure 9-9 and Appendix 2 – QAQC Charts) were found to have high variability for some elements and were found to have certified values which were entirely different to the values that were consistently being returned from the laboratories. As a result, the use of CRM sourced from INTEM was discontinued.

The CRM sourced from GeoStats (Figure 9-10) was found to be accurate, although the CRM consistently reported below the certified value at both ALS and MSA laboratories. These CRM are gold standards and so are not certified for other elements. As a result, a limited amount of these CRM was used.

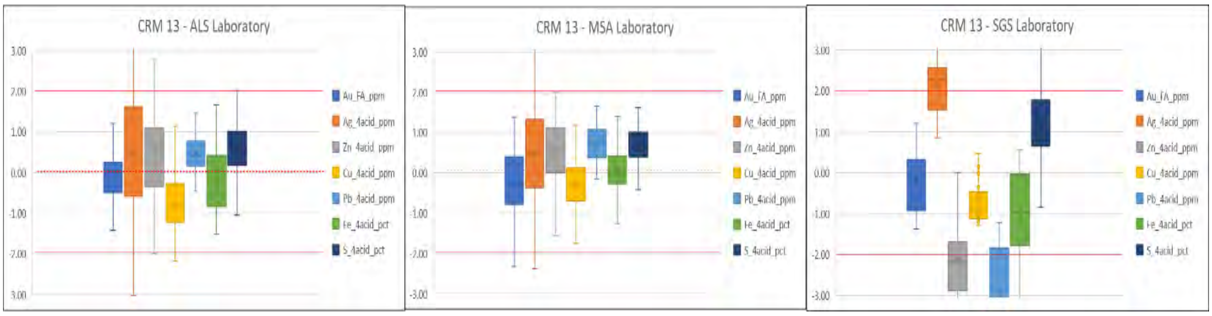


Figure 9-8: Box and whisker plots examples for CRM sourced from OREAS, inserted into sample sequences of drill core, RC and channel samples and analysed at ALS, MSA and SGS laboratories.

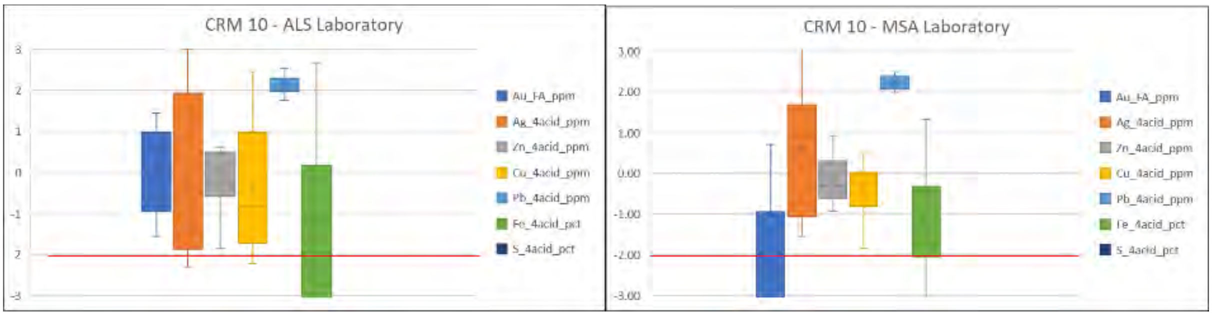


Figure 9-9: Box and whisker plots examples for CRM sourced from INTEM, inserted into sample sequences of drill core, RC and channel samples and analysed at ALS, MSA laboratories.

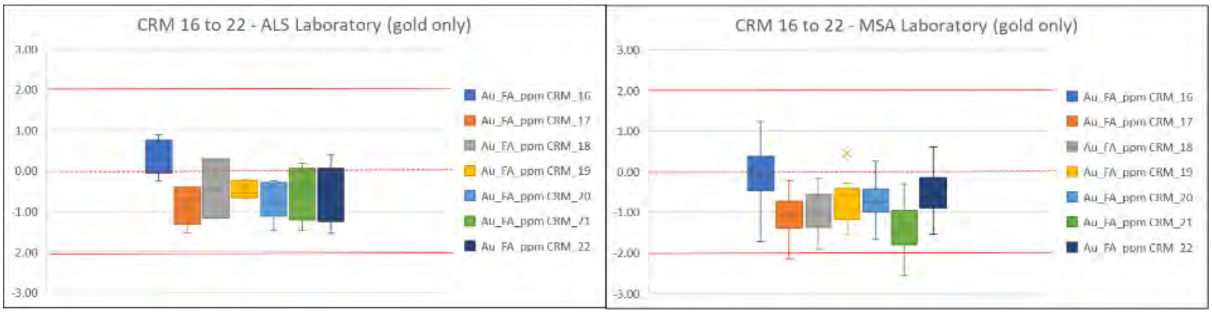


Figure 9-10: Box and whisker plots for CRM sourced from GeoStats (gold only), inserted into sample sequences of drill core, RC and channel samples and analysed at ALS, MSA laboratories.

9.4.4 Replicate Analysis

9.4.4.1 *Coarse Rejects*

Replicate assay of 186 coarse reject samples from 2019 drilling has been done to verify assay precision (Table 9-6). Original core samples were from the 2019 DD drilling which were analysed by MSA (San Juan preparation and Vancouver analysis). Coarse reject samples were analysed by ALS (Mendoza preparation and Vancouver analysis). The repeat analysis technique was identical to the original. The repeat analyses correlate very closely with the original analyses providing high confidence in precision of results between MSA and ALS.

Table 9-6: Summary of analysis of 186 coarse samples that were subject to repeat split, grind and assay

Element	Mean		Median		Std Deviation		Correlation Coefficient
	MSA	ALS	MSA	ALS	MSA	ALS	
Au (FA and GFA ppm)	4.24	4.27	0.50	0.49	11.15	11.00	0.9972
Ag (ICP and ICF ppm)	30.1	31.1	5.8	6.2	72.4	73.9	0.9903
Zn ppm (ICP ppm and ICF %)	12312	12636	2574	2715	32648	33744	0.9997
Cu ppm (ICP ppm and ICF %)	464	474	74	80	1028	1050	0.9994
Pb ppm (ICP ppm and ICF %)	1944	1983	403	427	6626	6704	0.9997
S (ICP and ICF %)	2.05	1.95	0.05	0.06	5.53	5.10	0.9987
Cd (ICP ppm)	68.5	68.8	12.4	12.8	162.4	159.3	0.9988
As (ICP ppm))	76.0	79.5	45.8	47.6	88.1	90.6	0.9983
Fe (ICP %)	4.96	4.91	2.12	2.19	6.87	6.72	0.9994

Cd values>1000 were set at 1000

Replicate assay of 192 coarse reject samples from 2021 drilling has been done to verify assay precision following indications of low accuracy as determined by the CRM data from SGS laboratories (Table 9-7). Original core samples were from the 2021 DD drilling which were analysed by SGS Laboratories (San Juan preparation and Lima analysis). Coarse reject samples were prepared and analysed by ALS (Mendoza preparation and Lima analysis). The repeat analysis technique was identical to the original. The repeat analyses correlate closely with the original analyses providing confidence in precision of results between SGS and ALS.

Table 9-7: Summary of analysis of 192 coarse samples that were subject to repeat split, grind and assay

Element	Mean		Median		Std Deviation		Correlation Coefficient
	MSA	ALS	MSA	ALS	MSA	ALS	
Au (FA and GFA ppm)	1.754	1.680	0.432	0.441	20.8	21.5	0.9837
Ag (ICP and ICF ppm)	12.14	11.57	0.93	1.03	7085	5925	0.9995
Zn ppm (ICP ppm and ICF %)	6829	7052	709	685	4.54E+08	5.34E+08	0.9942
Cu ppm (ICP ppm and ICF %)	203.4	202.9	25.7	24.5	3.30E+05	3.35E+05	0.9967
Pb ppm (ICP ppm and ICF %)	1768	1719	94.7	91.6	5.04E+07	4.39E+07	0.9959
S (ICP and ICF %)	2.23	2.10	0.94	0.87	16.51	15.56	0.9953
Cd (ICP ppm)	43.9	42.4	4.1	4.0	19594	18511	0.9956
As (ICP ppm))	45.4	45.2	16.0	16.9	10823	9893	0.9947
Fe (ICP %)	3.07	3.30	2.38	2.31	4.80	9.28	0.9781

Values below detection were set to half detection limit.

Limit of detection for Fe was exceeded for three samples submitted to SGS with no overlimit analysis.

9.4.4.2 *Pulp Rejects*

Replicate assays of 140 pulp reject samples from the 2022 drilling (parts of drill holes GNDD654 and GNDD666) were done to check assay precision (Table 9-8). The original pulps were analysed by MSA laboratories (San Juan preparation and Vancouver, Canada analysis). Replicate pulps were analysed by ALS (Lima, Peru). The analytic techniques were identical at both laboratories.

Table 9-8: Repeat analysis of pulp rejects from two diamond core drill holes completed in 2020

Element	Count	Mean		Median		Std Deviation		Correlation coefficient
		MSA	ALS	MSA	ALS	MSA	ALS	
Au (FA ppm)	140	0.27	0.30	0.01	0.02	0.98	1.05	0.9829
Ag (ICP ppm)	140	1.16	1.14	0.16	0.16	6.15	6.31	0.9965
Zn (ICP ppm)	140	555	565	50	56	2471	2469	0.9996
Pb (ICP ppm)	140	92.3	95.4	13.6	13.5	338	351	0.9977
S (ICP %)	140	0.64	0.61	0.17	0.17	1.22	1.12	0.9982
Fe (ICP %)	140	1.62	1.59	0.64	0.66	1.91	1.88	0.9991

9.4.5 Twin Hole Analysis

Drill holes have been twinned to check the intercept widths and grades of historic drill holes against CEL drilling.

DDH53 (CMEC diamond core hole, 1999) is twinned by GNRC110 (CEL, 2020). There was no down hole survey for DDH53 so the exact distance between samples is not known but is expected to be less than two metres.

The holes were drilled eastwards into the Magnata manto near the Magnata Fault Zone. For DDH53 only the intervals from 6.60 – 21.80 metres and 22.35 – 44.00 metres were sampled. Analysis for Au, Ag, Cu, Pb and Zn was recorded. GNRC1110 was sampled at 1.0 metre intervals from 1.0 – 61.0 metres (end of hole). DDH53 was not drilled using triple tube, resulting in poor core recovery over some intervals. The mineralised intervals recorded are at similar depths. GNRC110 returned higher weighted average gold grade. This may be a result of core loss in DDH53 in mineralised (sulphide) intervals or it may be natural variability within the deposit. Interval pairs are summarised in Table 9-9 and Figure 9-11.

Table 9-9: Weighted average assay results of twin drill holes DDH53 and GNRC110

Hole Id	From (m)	Int (m)	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm	Zn (ppm)
DDH53	6.6	37.4	1.62	67.6	104	637	245
GNRC110	7.0	37.0	3.22	70.8	97	2,986	220

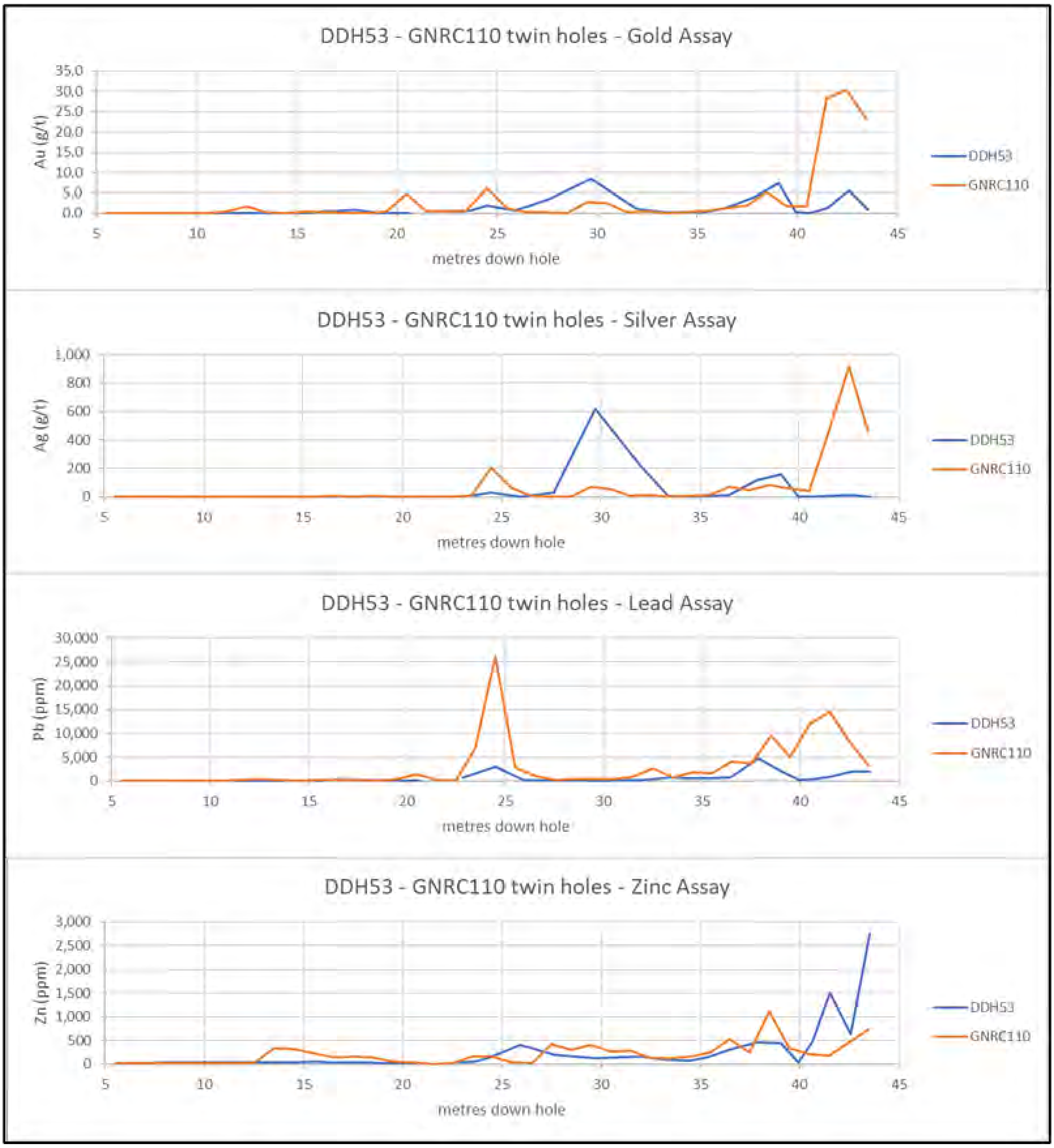


Figure 9-11: Down- hole assay plots for Au, Ag, Pb and Zn from DDH53 and GNRC110.

DDH34 (CMEC diamond core hole, 1999) was twinned by 04HD08 (La Mancha diamond core hole, 2004) and GNDD003 (CEL diamond core hole, 2019). These holes were drilled eastwards in the southern part of the Norte manto. 04HD08 was terminated before it drilled through the full width of the mineralised zone. There is no down-hole survey known for DDH34 so the exact distance between the samples is not known but is expected to be less than two metres. A comparison of drill hole assay results is shown in Table 9-10 and Figure 9-12. For DDH34 only assays for Au, Ag, Cu, Pb and Zn have been recorded. The results of GNDD003 compare reasonably well with DDH34 with the grades recorded int GNDD003 being slightly lower. The slight offset in anomalies is due to the dip of the bedding controlling mineralisation being slightly oblique to the drill dip.

Table 9-10: Weighted average assay results of twin drill holes DDH34, GNDD003 and 04HD08

Hole Id	From (m)	Int (m)	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm	Zn (ppm)
DDH34	42.56	20.54	14.5	8.1	222	1,075	7,181
GNDD003	42.10	20.90	10.2	6.9	230	593	9.420
04HD08	46.65	11.15	2.4	3.0	79	1,085	9,020

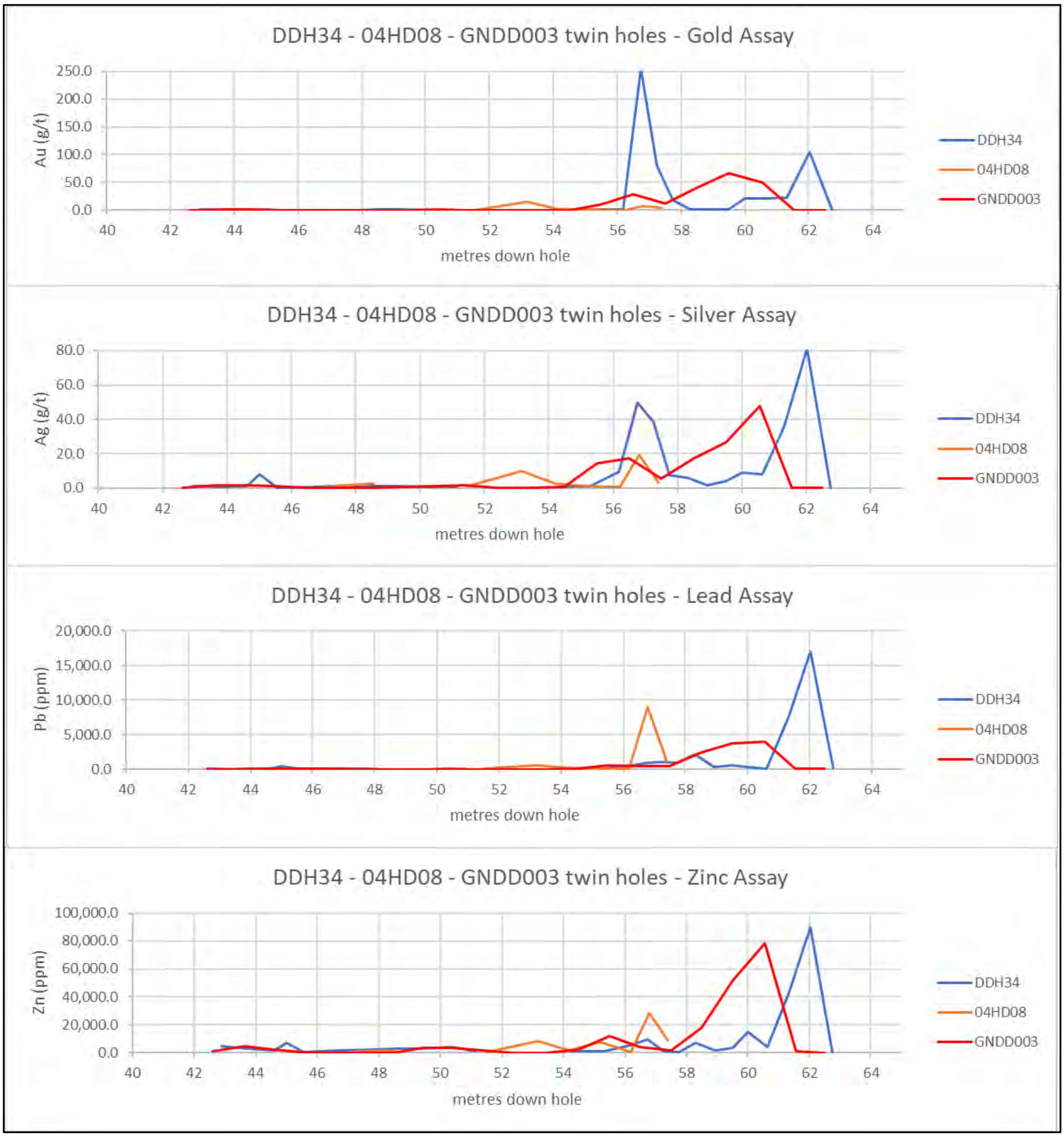


Figure 9-12: Down- hole assay plots for Au, Ag, Pb and Zn from DDH34 and GNDD003 and 04HD08.

Twin hole GNDD008, GNDD008A (CEL, 2019) and GNRC107 (CEL, 2020) were drilled southwards at the Magnata Fault Zone. GNDD008A was a re-drill of GNDD008 after GNDD008 intersected a mined cavity near the fault zone and the drill string could not be withdrawn. GNRC107 steepened at depth but was within two metres of GNDD008A. A comparison of drill hole assay results is shown in Table 9-11 and Figure 9-13. The drill holes passed through two fault zones of the Magnata Fault, the M1 zone (shale, dacite and limestone) and the M2 (limestone). In the M1, GNRC107 intersected significantly higher gold grade and in the M2, GNDD008A intersected significantly higher gold grades.

Table 9-11: Weighted average assay results of twin drill holes GNDD008, GNDD008A and GNRC107

Hole Id	From (m)	Int (m)	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm	Zn (ppm)
M1:							
GNDD008	16.50	36.50	0.32	7.9	52	357	996
GNDD008A	17.50	35.50	0.23	12.8	43	321	778
GNRC107	16.00	37.00	2.67	13.0	57	815	1,904
M2:							
GNDD008A	91.00	24.00	2.78	37.4	582	862	4,587
GNRC107	91.00	24.00	0.01	4.0	24	470	2,078

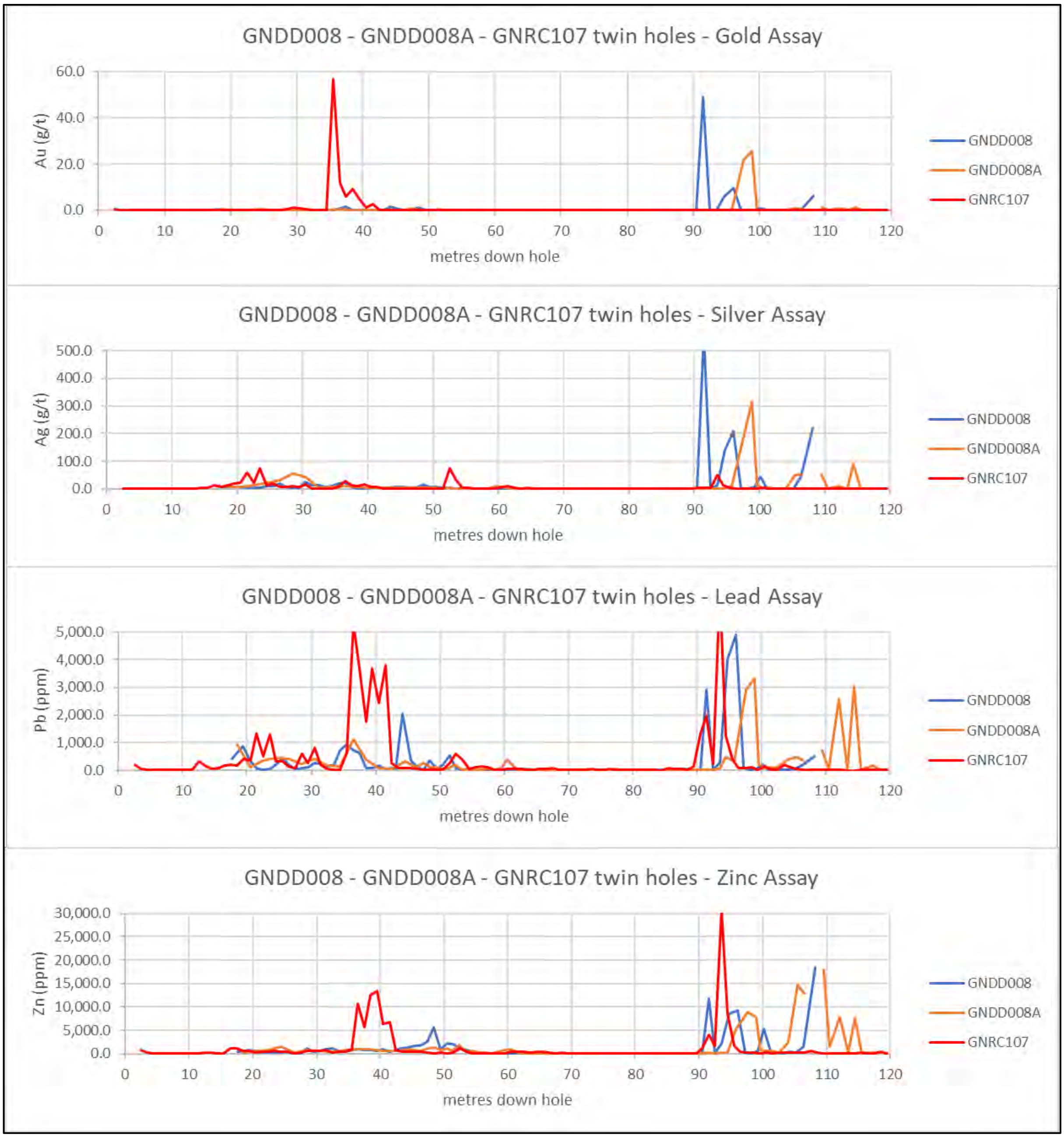


Figure 9-13: Down- hole assay plots for Au, Ag, Pb and Zn from GNDD008, GNDD008A and GNRC107.

Twin holes GNDD421 and GNDD424 intersected mineralisation in siltstone (LUT) above the Norte manto. The two holes are separated by only 3.5 m. The dip of the bedding has resulted in mineralisation at slightly different depths in each hole. At the LUT-CAL contact a difference of 13.25 m was logged. A comparison of the assay over the mineralised interval is shown in Table 9-12 and graphically in Figure 9-14. There is some variability in the weighted average gold grade and less variability in silver and zinc.

Table 9-12: Weighted average assay results of twin drill holes GNDD421 and GNDD424

Hole Id	From (m)	Int (m)	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm	Zn (ppm)
GNDD421	66.00	27.00	1.29	3.5	121	741	3,134
GNDD424	74.00	27.20	0.52	3.3	115	2,012	3,860

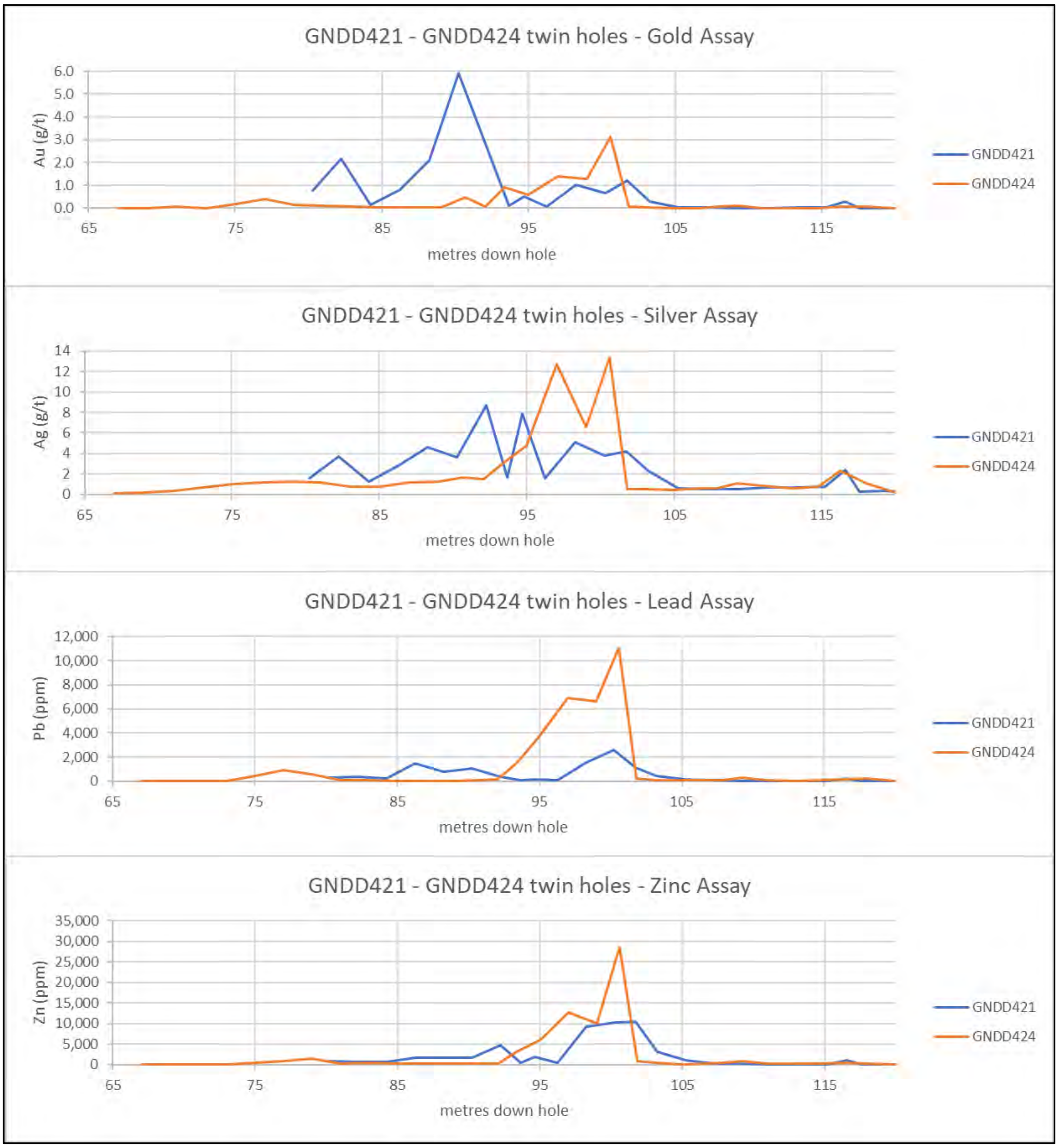


Figure 9-14: Down- hole assay plots for Au, Ag, Pb and Zn from GNDD421 (+13.25m) and GNDD424.

Analysis of the twin hole results indicate there is some close spaced variability in the gold grades in particular with less variability observed for silver, lead and zinc. There is no bias observed between the historic assay results as recorded and the assay results for the diamond core and RC drilling completed by CEL.

9.4.6 QA/QC Results

The following conclusions can be made from the QAQC of the data used for the Hualilan MRE.

- Assays from duplicate samples (¼ diamond drill core, field duplicate RC and channel samples) show a good correlation. Variability observed in channel sample field duplicates can be explained by the low number of sample pairs and mobilisation of elements in near surface (partially oxidised) samples.
- Assay of blank samples strategically placed in the sample sequence after mineralised samples indicate that the laboratories used are cleaning the crushing and grinding equipment between samples. Instances of contamination are rare.
- Analysis of commercially prepared CRM inserted into sample sequences suggest good precision and accuracy of the analytic techniques used at the ALS and MSA laboratories. Results are less accurate at the SGS laboratory. There is no inherent bias observed in the assay results received.
- Check preparation and analysis of coarse rejects initially analysed at the SGS lab have a good correlation with repeat analyses suggesting the SGS lab results have no inherent bias.
- Inter-lab check preparation and analysis of coarse rejects analysed at ALS and MSA laboratories have a strong correlation suggesting good precision and accuracy.
- Analysis of samples from twin drill holes show some variability in interval weighted average assay values (in particular gold) observed in interval pairs separated by up to 5 metres.
- There is no inherent bias observed between historic drill hole assays as recorded and assays received for CEL diamond core or reverse circulation drilling.

10 DATA VERIFICATION

Drill hole and channel logging data was captured by the technical assistants and geologists into a standard Excel workbook with multiple worksheets. Each drill hole has a separate Excel file. The geologist collecting the information had the data peer reviewed and checked by another geologist after the data was entered into the spreadsheet. For diamond core drill holes, the Excel file contains simple formulae that check the validity of the data entered.

For diamond drill holes the following information is collected, entered in the log data which is then uploaded to the Project Data Base and checked for validity:

- Collar (surveyed collar, set-up orientation, drilling equipment, depth, start/end date, logging geologist and reviewing geologist)
- Downhole survey (equipment used, charts of deviation in azimuth and dip to check validity of data collected)
- Core Recovery (drillers blocks used to identify run length and core recovered)
- RQD (recorded for each recovered core run)
- Weathering (down hole depths of cover, oxidised where sulphides are completed oxidised, partial oxidation, fracture oxidation and without oxidation)
- Lithology (down hole depths of key lithology packages)
- Alteration (down hole depths of alteration assemblages and quantification of alteration intensity)
- Mineralisation (down hole depths of mineralisation assemblages and quantification of quantity of mineralisation)
- Structure (down hole depth and core-axis angles of significant structures)
- Sampling (down hole depths of samples, sample ID, including location of blanks and CRM inserted into the sequence)
- SG (down hole depth, SG measurement via weight in air and weight in water method and lithology notes)

For RC drill holes the following information is collected, entered in the log data which is then uploaded to the Project Database and checked for validity:

- Collar (as for diamond drill holes)
- Downhole survey (as for diamond drill holes)
- Weights (weight of sample recovered as a guide to recovery and consistency of sampling)
- Weathering (as for diamond drill holes)
- Lithology (as for diamond drill holes)
- Mineralisation (as for diamond drill holes)
- Sampling (on regular 1 m intervals below the cover sample ID, including location of blanks and CRM inserted into the sequence)

For channel samples the following information is collected, entered in the log data which is then uploaded to the Project Data Base and checked for validity:

- Channel ID, channel survey points coinciding with the start and end of each channel sample collected, date, logging geologist.
- For each channel sample data collected includes lithology, alteration, intensity of alteration, notes on the alteration, mineralisation, quantification of mineralisation, notes of the style of mineralisation, structure and orientation of structure (dip and dip direction).

When the geology team had uploaded the data in each tab of the log sheet, the Geographic Information System (GIS) team was notified to review the uploaded information. The GIS team then proceeded to verify each of the tabs of the logging. The GIS team uses formulae within the standard spreadsheet to validate the data:

- The drill hole or channel co-ordinates against design and update the spreadsheet with surveyed collar coordinates.
- All the depth from (metres) and depth to (metres) records are in sequence, no overlapping intervals, depth not outside the end of the hole.
- Deviation of the drill holes, check of anomalous results.
- Core recovery and RQD checks against interval length, overall hole recovery.
- EOH depth agrees between CEL logs, core photographs and drill company daily drilling logs.
- For weathering, lithology and alteration, structure check that the codes used conform to standard codes, check errors with geologists and check the need for additional lookup codes for newly defined lithology.
- Alteration intensity, mineralisation percentages and core axis angle conform to parameters.
- Log data loaded in Leapfrog GEO to see that the geology is consistent with holes on the same section and adjacent sections. Check logging and interpretations where unexpected results are logged.
- Sampling intervals conform to the length parameters (0.50 metres to 2.00 metres length), blanks, duplicates and standards (CRM) are clearly marked. Check that missing sample intervals conform to logged section of No Core (NC) and that sections logged as NC are not included in sample lengths.

Logs that were validated were recorded in the LOG Validation form and were copied to a folder ready to be uploaded into the database.

Assay data was received from the laboratory via email to multiple accounts within the Company. The final data is:

- Checked using the externally submitted QAQC (Blanks, Duplicates and CRM assays). These data are analysed to check for preparation and assay precision and accuracy.
- Combined with the data recorded in the Sampling Log, loaded into Micromine Origin to check the results against adjacent drill holes.

Final assays that are validated are copied to a folder ready to be loaded into the database.

10.1 Database Management

For the drill hole data, logs and assays are stored in a customised database which is based on a SQL Server. The database is managed by the GIS/Data team in Argentina and Ecuador. Technical expertise is available via SRK Consulting who engineered the database management system. The logging and assay data, in CSV format, is loaded into a cloud-based database hosted on the MS Azure platform and is processed through Microsoft's ADF.

Importing the data using SQL Server involves several data checks made by the software. Where an error is encountered, a message is returned identifying the location of the conflict. The log and assay data will not load into the database until all conflicts have been resolved. If there are any updates to a log, then the data can be re-loaded to replace existing data.

Through Microsoft SQL Server Management Studio, queries can be made, data downloaded or ODBC links can be made.

Historic drill data is kept in separate Excel based data sheets. These data will not be added to and are incomplete. In addition, these data are gradually being phased out as drilling proceeds.

The channel sample data is kept in a separate Excel-based database. These data are treated differently to drill hole data by mining software packages and the SQL Server software has not yet been updated to accept the channel sample data.

10.2 Data Reviews and Checks

All drill hole data and channel data downloaded from the SQL database through MS Azure is periodically imported in Leapfrog, Micromine and Surpac for geological modelling and resource estimation. There have been few reported database errors due to the validation checks made in Excel and the validation checks made when importing the data to the database.

There have been no independent audits of the database.

11 MINERAL PROCESSING AND METALLURGICAL TESTING

11.1 Introduction

A metallurgical testwork program has been conducted on drill core samples originating from the Hualilan project in Argentina at the SGS Lakefield laboratory in Ontario, Canada. Testing started in 2021 and is ongoing, however the cut-off for inclusion in this scoping study was October 2023. A copy of the SGS report is included as an annexure to this Study, Appendix 6 – SGS Report.

SGS's scope was to undertake gravity, flotation, leaching and comminution testwork on composite samples representing the major mineralisation types from the project. Flowsheets for the beneficiation process for these different mineralisation types were developed in conjunction with the testwork program, with the flowsheets evolving as more results were received and evaluated.

The target for the high grade zinc material was to produce a:

- Zinc concentrate containing at least 50% zinc, with minimum gold content as this stream would generate low gold payabilities;
- Lead concentrate containing at least 50% lead; and,
- Precious metal concentrate containing at least 30 g/t gold.

The results from the testwork program at SGS indicated that a sequential flotation route was successful in generating a ≥50% zinc concentrate, a lead concentrate grading ≥50% lead along with a precious metal concentrate with ≥30 g/t gold.

Comminution testwork was conducted on the different mineralisation types for milling circuit selection and design. A small number of variability samples were tested using the sequential flotation route, however these results were inconclusive as the tests were conducted at too coarse a grind.

The SGS testwork included:

- Gravity Recovered Gold (GRG) testwork using high G-force Knelson concentrator, followed by cleaning of concentrate using a Mozley table and return of table tailings to the Knelson tailings for downstream processing. This testwork also allowed the equipment vendor to provide the expected full scale plant gold recovery and to select the most appropriate gravity equipment.
- Bulk sulphide flotation on GRG tails to generate a sulphide concentrate containing lead, copper and zinc mineralisation and iron sulphides that are rich in gold.

- Sequential flotation on GRG tails to generate a high grade lead and copper concentrate with gold rich pyrite, followed by a zinc flotation step to generate a zinc sulphide concentrate.
- Gold leaching of comminuted samples, gravity tailings, and flotation tailings to recover contained gold which is not reporting to either flotation or gravity concentrates.
- Comminution testwork to generate mineralised material properties for optimisation of milling circuit design.
- Flash floation testing, however this technology wasn't deemed to provide a benefit to the circuit.
- Acid base accounting testwork for use in the project enviromental approvals.

The testwork samples were selected from diamond drill hole intervals across the deposit to represent different lithologies and alteration types found at in Hualilan project mineralisation.

A summary of the make-up of these composites is shown in Table 11-1. Weighted average head assays calculated from the interval assays are shown in Table 11-2. Head grades varied between 0.7 g/t Au in sedimentary hosted composite up to 10.4 g/t Au from high grade (limestone hosted) manto-style composite. The zinc grades varied from 0.03% in low grade sediment hosted up to 3.8% in the higher grade composite. A manto-style sample for comminution testing had a higher zinc grade but that sample was not tested for zinc recovery.

A full interval inventory for the variability samples is shown in Table 11-3.

Table 11-1: Summary of Metallurgical Composite Samples

Sample Title	Description	Testwork Type	Number of Drill Holes	Number of Intervals	Total Interval Length m	Depth Min m	Depth Max m
High Grade Manto - High Sulphide	High Grade (HG) manto-style, limestone (CAL) hosted, high sulphide	Gravity, Flotation (Bulk and Sequential)	6	6	23	55	125
			9	9	77	60	301
		SMC, Ai, Bwi	3	3	9	309	367
HG MFZ - high Sulphide	High Grade (HG) Magnata Fault Zone (MFZ), sediment and limestone hosted, high sulphide	SMC, Ai, Bwi	2	2	10	235	360

Sample Title	Description	Testwork Type	Number of Drill Holes	Number of Intervals	Total Interval Length m	Depth Min m	Depth Max m
LG DAC hosted - low sulphide	Low grade (LG) dacite-hosted, low sulphide	Gravity, Flotation (Bulk and Sequential)	5	5	33	154	360
			5	5	33	154	360
		SMC, Ai, Bwi	2	2	101	135	360
LG Sed Hosted - low sulphide	Low Grade (LG) sediment hosted, intermediate to low sulphide	Gravity, Flotation (Bulk and Sequential)	3	3	59	170	228
		SMC, Ai, Bwi	2	2	12	117	282
ROM sample	Blend of LG Dacite-hosted and HG Manto		-	-	-	-	-
			-	-	-	-	-

Table 11-2: Summary of Metallurgical Sample Expected Head Grades

Sample Title	Description	Testwork Type	Au g/t	Ag g/t	Cu %	Pb %	Zn %	S %
High Grade Manto - High Sulphide	High Grade (HG) manto-style, limestone (CAL) hosted, high sulphide	Gravity, Flotation (Bulk and Sequential)	10.41	31.70	0.15	0.46	3.17	-
			9.38	44.26	0.13	0.57	3.83	9.90
		SMC, Ai, BWi	6.22	79.31	0.33	0.27	7.85	15.77
HG MFZ - high Sulphide	High Grade (HG) Magnata Fault Zone (MFZ), sediment and limestone hosted, high sulphide	SMC, Ai, BWi	5.75	22.91	0.07	0.47	3.54	9.76
LG DAC hosted - low sulphide	Low grade (LG) dacite-hosted, low sulphide	Gravity, Flotation (Bulk and Sequential)	1.10	7.03	0.01	0.03	0.09	1.25
			1.08	9.16	0.02	0.04	0.10	1.25
		SMC, Ai, BWi	1.00	7.51	0.01	0.03	0.16	1.68
LG Sed Hosted - low sulphide	Low Grade (LG) sediment hosted, intermediate to low sulphide	Gravity, Flotation (Bulk and Sequential)	0.68	7.60	0.02	0.05	0.34	1.36
		SMC, Ai, BWi	0.68	2.79	0.00	0.01	0.03	0.71
ROM sample		Gravity, Leach	1.68	10.96	0.02	0.08	0.45	-

Sample Title	Description	Testwork Type	Au g/t	Ag g/t	Cu %	Pb %	Zn %	S %
	Blend of LG Dacite-hosted and HG Manto	Gravity Leach	1.53	12.71	0.03	0.07	0.65	-

Table 11-3: Detailed Variability Sample Inventory

Sample ID	Drill Hole	From	To	Length m	Au_ppm	Ag_ppm	Cu_ppm	Pb_ppm	Zn_%
LGLUT229	GNDD229	169.6	197.0	27.4	0.64	8.36	238	804	0.43
HGGN227	GNDD227	222.0	230.0	8.0	4.21	53.57	615	547	1.75
HGGN246	GNDD246	179.5	180.4	0.9	12.68	25.00	1221	353	7.83
HGGN264	GNDD264	104.9	123.2	18.3	1.52	19.99	599	5116	2.04
HGGN280	GNDD280	239.4	248.0	8.6	6.29	65.52	160	697	0.72

11.2 Characterisation of Samples

Plots of the metallurgical sample interval data versus drill hole interval data (bottom cut at 0.3 g/t AuEq) for the following range of parameters are shown in Figure 11-1, including:

- Location of sample compared to the location of drill holes in the database.
- Grade of gold, zinc and sulphur in the samples compared with the grades of drill hole intervals in the database.
- Alteration type, lithology and oxidation of the samples compared with these parameters of drill hole intervals in the database.

Drilling data is represented in orange and sample data is represented in green with the y-axis representing frequency.

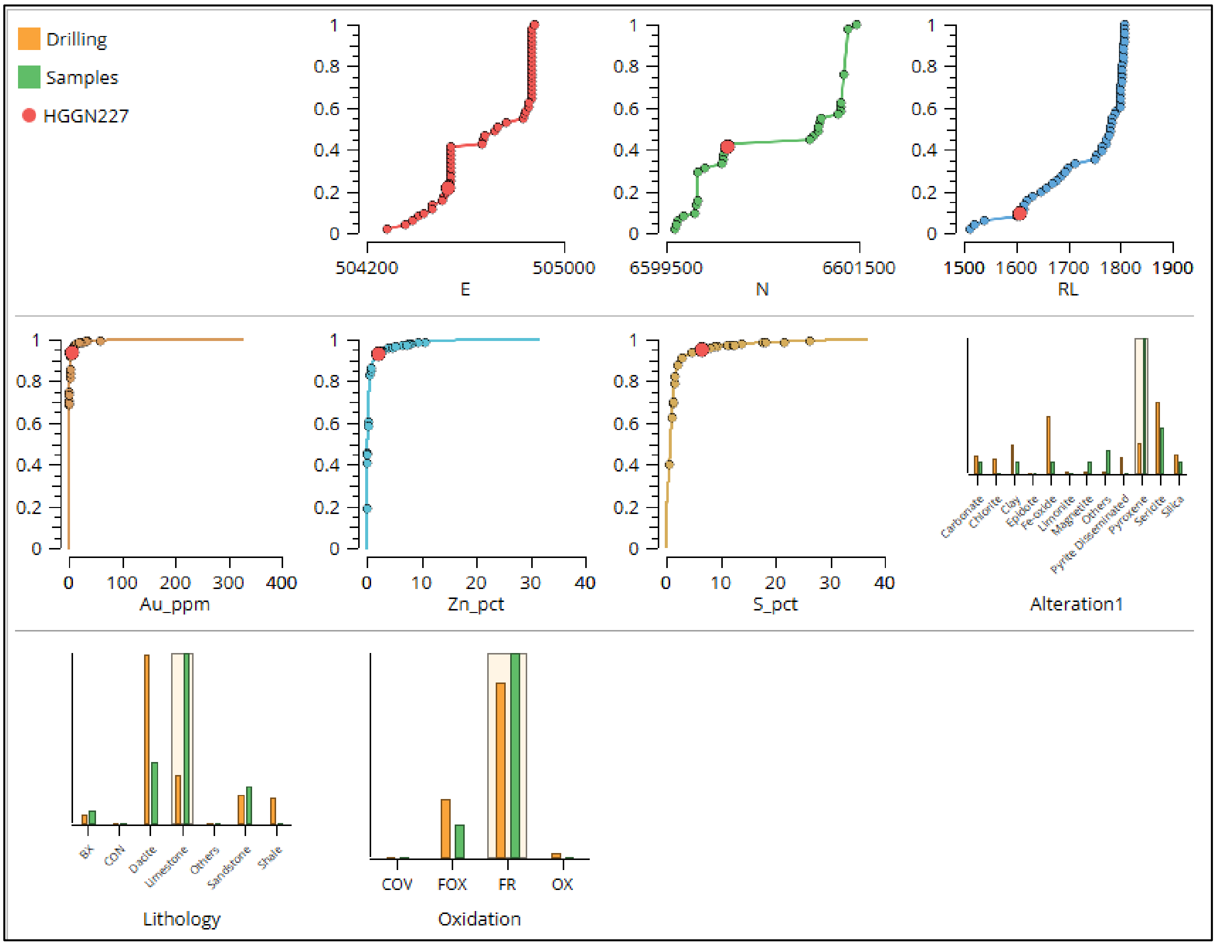


Figure 11-1: Sample Representivity versus Drilling Results with a 0.3 g/t AuEq Cut-off

The key findings of this comparison include:

- The samples tested were predominantly higher-grade skarn (limestone hosted) material.
- A smaller number of samples of siltstone/ sandstone and dacite rock types have been tested to determine their response to the same process.
- In general, samples are biased towards higher grade manto-style mineralisation in limestone which contains mainly pyroxene alteration type. Testing of lower grade samples is expected in the next phase of testing.
- Most of the pyroxene (+garnet +epidote) skarn alteration is associated with the high grade in limestone so the samples are naturally biased to that style of alteration however the majority of mineralisation is hosted in this type of alteration.
- The samples are spatially well represented in both the easterly and northerly directions and are also well represented with RL, except for region between RL1550 – RL1600. Samples will be selected from this depth as deeper drilling progresses.

11.3 Mineralogical Characterisation

Hualilan mineralisation:

- Occurs in all rock types, principally in limestone, siltstone to fine grained sandstone and dacite intrusions.
- Sulphide minerals are undeformed, consistent with textures in the dacite, and accordingly the mineralisation is interpreted to be post-orogenic and postdate all rock types.
- The mineralisation generally follows the alteration. Earlier zinc skarn mineralisation occurs with the prograde skarn, and later mineralisation occurs with retrograde alteration.
- Prograde mineralisation consists of sphalerite, lesser galena, rare chalcopyrite, pyrrhotite and magnetite (which replaces hedenbergite) and garnet. Gold is paragenetically slightly later that the initial skarn alteration.
- Sphalerite with red/ brown colour is relatively iron-rich and evolves to later yellow coloured sphalerite. Growth rims of lighter coloured sphalerite have been observed around earlier formed darker sphalerite.
- Pyrite may also be deposited with the prograde mineralisation, although it is difficult to distinguish with the later, more dominant (retrograde) pyrite.
- Pyrite is the dominant retrograde sulphide mineralisation. Pyrite replaces hedenbergite, garnet and sphalerite, or occurs at silicate grain boundaries forming irregular masses.
- Small (10-40 mm diameter) inclusions of gold are observed in thin section, both as tiny inclusions within pyrite, and as free gold within hedenbergite.
- Sphalerite does not appear to contain gold but does contain tiny inclusions of chalcopyrite and pyrrhotite.
- Later retrograde pyrite mineralisation has well-formed cubic crystal faces and is commonly coarser, up to several millimetres. These pyrites are less commonly associated with gold mineralisation.

11.4 Spatial Distribution of Samples

Figure 11-2 and Figure 11-3 are plan views of the variability and composite metallurgical sample locations. These views illustrate the spread of samples across the north and south drilling areas of the deposit. This is showing that the samples are concentrated in higher grade domains of the south, central and north areas.

Figure 11-4 and Figure 11-5 are 3D views of the variability and composite metallurgical sample locations. This illustrates that the samples provide good coverage over the range of depths of the drill holes.

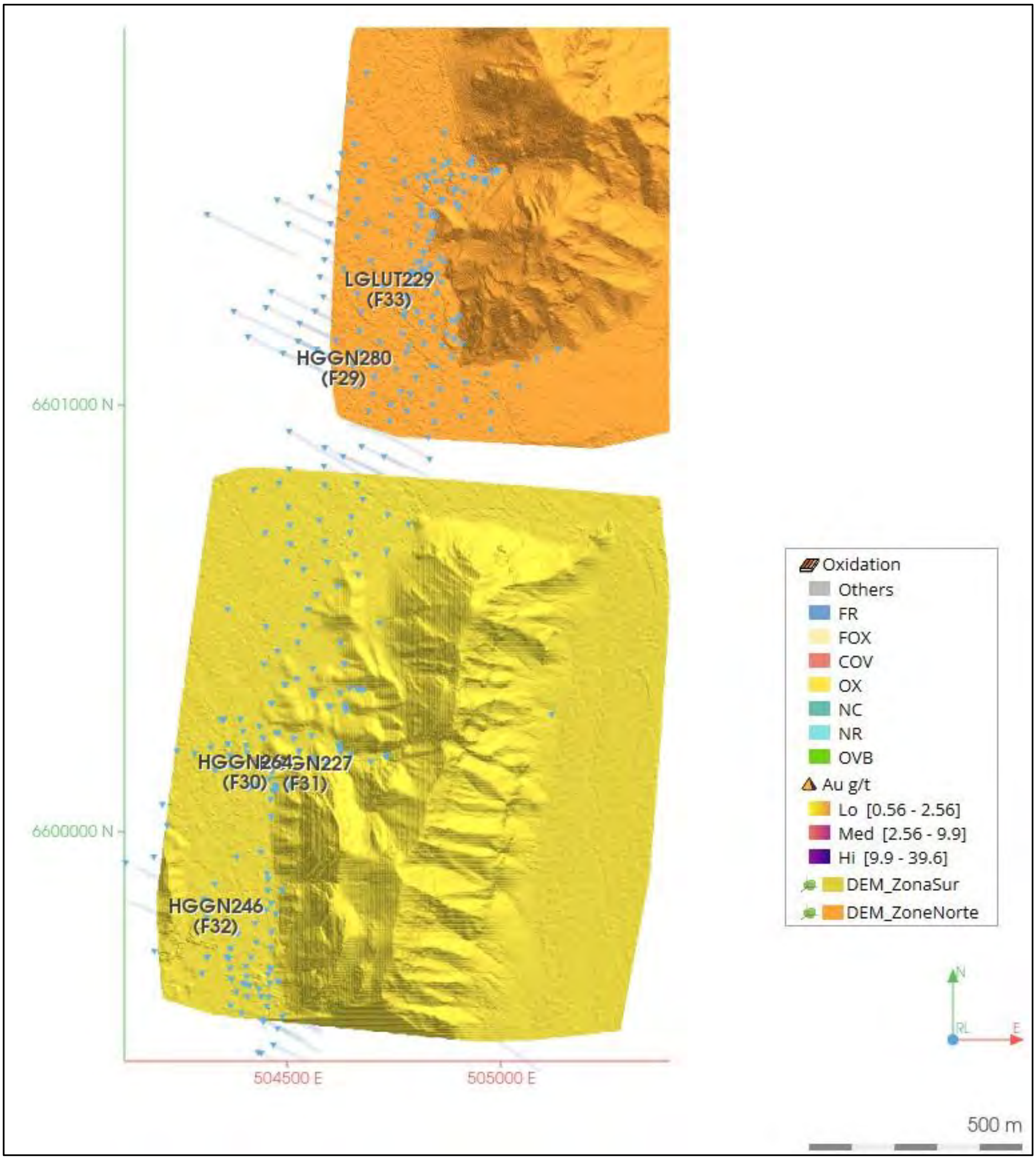


Figure 11-2: Plan View of Metallurgical Composite and Variability Sample Locations Compared with Drill Hole Locations

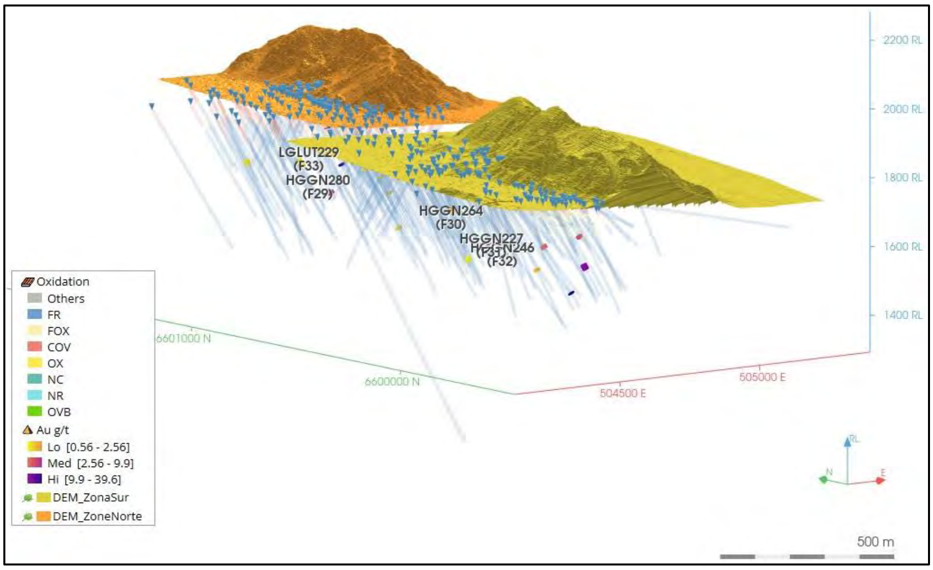


Figure 11-3: 3D View of Metallurgical Variability Sample Locations Compared with Drill Hole Locations

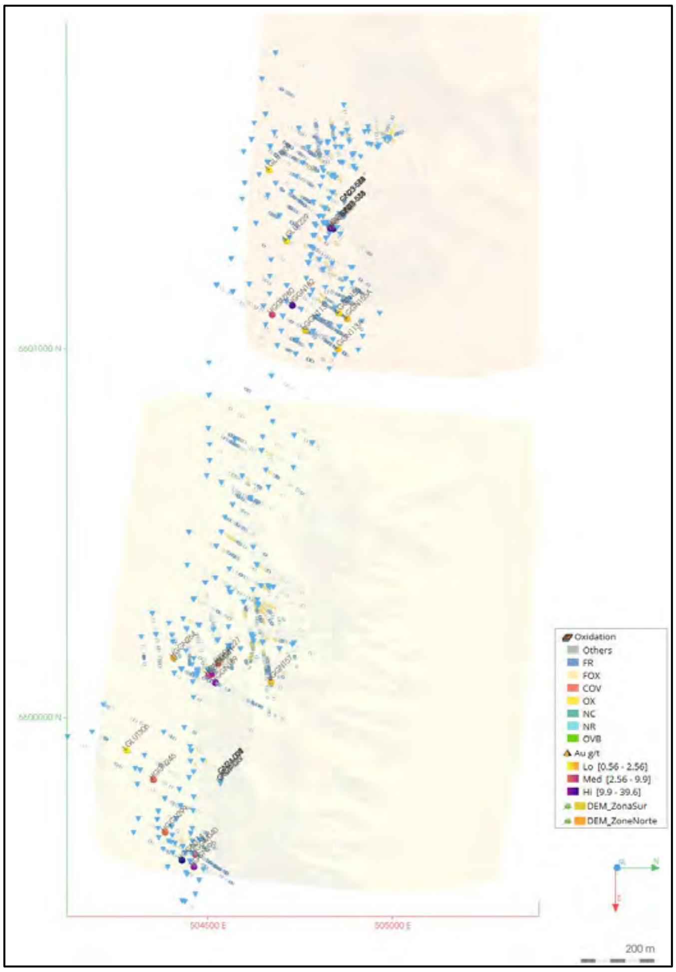


Figure 11-4: Plan View of Metallurgical Composite Sample Locations Compared with Drill Hole Locations

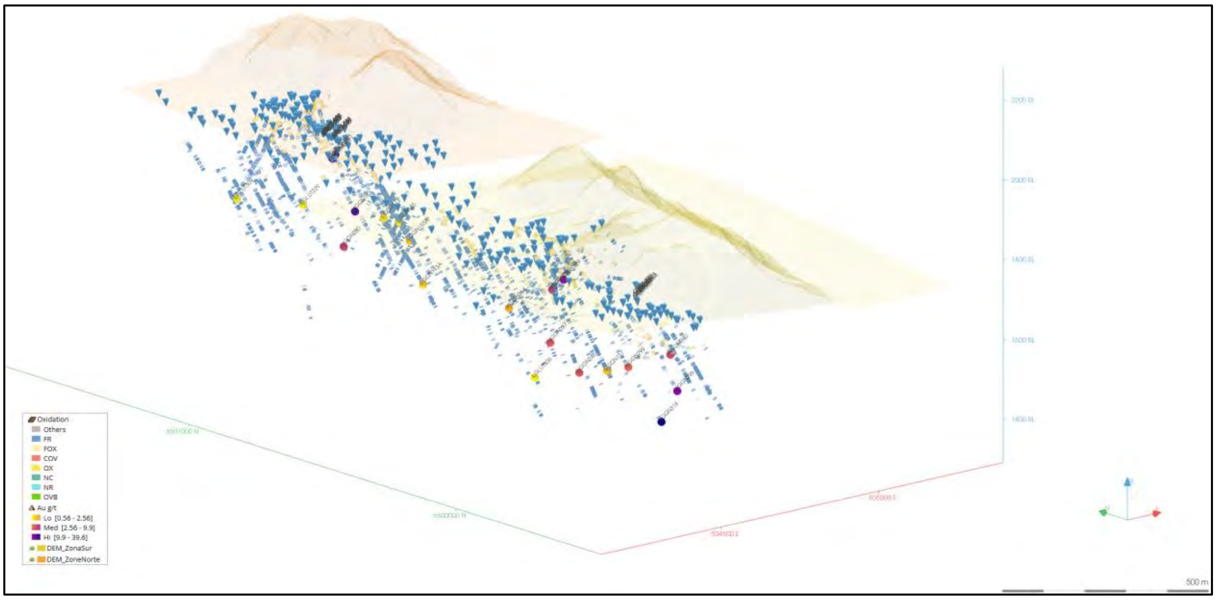


Figure 11-5: 3D View of Metallurgical Composite Sample Locations Compared with Drill Hole Locations

11.5 Test work

This section discusses:

- A summary of the pertinent results of the flotation, leaching and comminution.
- Interpretation of the testwork results to provide a basis for process selection and development of the process design criteria.
- Process opportunities and risks.

11.5.1 Comminution Testwork

Comminution testwork of four composites representing the various geological domains and lithologies making up the Hualilan resource provided the characteristic data for comminution modelling and mill circuit selection. These results are summarised in Table 11-4.

This program resulted in the:

- Determination of SMC indices for the lithology composite samples; and
- Determination of Bonds Ball Mill Work Indices (BWi) and abrasion indices (Ai) for the lithology composite samples.

Table 11-4: Summary of Comminution Testwork Results

ID	Description	RD	Axb	Ta	SCSE kWh/t	BWI kWh/t	Ai
HG COMM1	High grade (HG) manto-style, limestone (CAL) hosted, high sulphide	3.15	53.5	0.44	9.4	15.5	0.226
HG COMM2	High grade (HG) Magnata Fault Zone (MFZ), sediment and limestone hosted, high sulphide	3.13	60.1	0.5	8.9	15	0.108
LG COMM1	Low grade (LG) dacite-hosted, low sulphide	2.53	43.5	0.45	9.3	18.6	0.164
LG COMM2	Low grade (LG) sediment hosted, intermediate to low sulphide	2.66	30.9	0.3	11.1	21.8	0.374
HG Average	Design grind P80=50 µm	3.14	56.8	0.47	9.2	15.3	0.167
LG Average	Design grind P80=75 µm	2.60	37.2	0.38	10.2	20.2	0.269

The comminution results have been averaged separately for the high and low grade material types. This is considered appropriate for this level of study and as there are only two comminution results for each material type.

In the next phase of the project variability comminution testwork will be conducted to allow the uses of the 75% percentile value for design and the ball mill closing screen aperture will be better matched to the primary mill grind size.

11.5.2 Sequential Flotation Testwork

Results from the High grade B (HGB) composite flotation test (F28) have been used as the basis of the process design for the high grade material using the sequential flotation process. This approach is considered appropriate for recovery prediction for this material at Scoping Study level of design. In the next phase of testwork locked cycle testing with a more extensive variability program will be undertaken for a more accurate estimate of process plant recovery.

HGB composite was generated by combining intervals from 9 separate drill holes to represent a high grade Manto composite. A summary of the recipe for this sample is shown in Table 11-5.

Table 11-5: High grade Manto Composite Interval Data (Comp B)

Drillhole	From	To	Length	Au_ppm	Ag_ppm	Cu_ppm	Pb_ppm	Zn_%	Fe_%	S_%	Fe/Zn
GNDD182	148.7	153.0	4.3	31.81	96.52	5508	53125	8.12	8.91	13.65	1.10
GNDD189	60.0	64.9	4.9	16.34	114.44	1662	482	5.13	16.81	14.08	3.28

Drillhole	From	To	Length	Au_ppm	Ag_ppm	Cu_ppm	Pb_ppm	Zn_%	Fe_%	S_%	Fe/Zn
GNDD227[1]	222.0	230.0	8.0	4.21	53.57	615	547	1.75	6.80	5.35	3.89
GNDD246[1]	179.5	180.4	0.9	12.68	25.00	1221	353	7.83	16.00	12.12	2.04
GNDD264[1]	104.9	123.2	18.3	1.52	19.99	599	5116	2.04	4.70	5.48	2.30
GNDD280[1]	239.4	248.0	8.6	6.29	65.52	160	697	0.72	7.00	4.48	9.75
GNDD296	193.0	209.9	16.9	14.11	18.30	1841	42	5.80	21.61	16.27	3.72
GNDD299	147.5	157.4	9.9	3.40	43.95	1069	1981	5.25	17.61	12.34	3.35
GNDD314	296.9	301.0	4.1	39.60	17.34	1825	31	4.80	25.34	17.58	5.28
Composite	**60.0**	**301.0**	**75.8**	**9.71**	**44.90**	**1325**	**5697**	**4.02**	**12.50**	**10.20**	**3.11**

(1) – variability testwork samples

Variability testwork was conducted on 4 of the 9 interval samples from the composite to determine the response of these components to the optimised flotation circuit (Table 11-6). However, the grind duration used in these variability tests was 25-30% shorter than the grind duration for the composite. A shorter grind duration may generate a coarser primary grind which may reduce mineral liberation. As the regrind times were identical between tests, coarser regrind size may reduce product quality.

More variability tests are planned in the next project phase at a similar grind duration and grind size to that used in the F28 sequential flotation test. Milling circuit equipment has been selected to achieve the grind size used in the HG B composite flotation test (F28).

Table 11-6: Summary of Sequential Flotation Grinding Data and Zinc Results

Test	Comp	Title	Primary Grind	Rough Tails	Regrind Grind	Regrind	Head	Zn Clnr Conc		
							%		Assays	Dist %
			min	K80 µm	min	K80 µm	Zn Calc	Mass %	Zn %	Zn
F1	HG A	Exploratory test on Master Composite (Not optimised flowsheet)	35.0	106.0	-	-	4.5	4.5	41.9	42.4
F28	HG B	Repeat test with F26 with adjusted conditions	100.0	51.0	15.0	29.0	3.9	6.9	50.5	89.0
F29	GNDD280 Comp	Repeat of test F28 with new composite	75.0	90.0	15.0	36.0	0.7	2.6	17.0	67.1

Test	Comp	Title	Primary Grind	Rough Tails	Regrind Grind	Regrind	Head	Zn Clnr Conc		
							%		Assays	Dist %
			min	K80 µm	min	K80 µm	Zn Calc	Mass %	Zn %	Zn
F30	GNDD264 Comp	Repeat of test F28 with new composite	75.0	61.0	15.0	32.0	2.5	4.3	46.3	78.9
F31	GNDD-227 Comp	Repeat of test F21 with GNDD-227 composite	70.0	-	15.0	-	3.31	5.2	30.8	86.6
F32	GNDD-246 comp	Repeat of test F21 with GNDD-246 composite	70.0	-	15.0	-	2.12	3.4	48.8	79.0
F33	GNDD-229 comp	Repeat of test F21 with GNDD-229 composite	70.0	-	15.0	-	0.42	1.3	21.5	65.8
GCI-10	**Project Composite**	**(Not optimised flowsheet)**	**-**	**75.0**		**15.0**	**0.36**	**0.4**	**48.2**	**58.4**

Two of the variability samples produced zinc concentrate grades significantly below the target, while the other two variability samples produced zinc concentrate grades in the high 40's, i.e. close to the target grade.

The two samples that produced low zinc concentrate grades had significantly higher Fe/Zn ratios. It is possible that the higher Fe/Zn ratio is an indication of the increased proportion of pyrite to sphalerite. A finer grind and a more aggressive pyrite depression test in the next project phase is planned for samples that have high Fe/Zn ratio.

Sequential flotation also generates a lead concentrate. A summary of the lead recovery testwork data is shown in Table 11-7.

Table 11-7: Summary of Sequential Flotation Lead Results

Flotation Test	Comp	Head %	Mozley Conc & Cu/Pb Reclnr Conc Total		Cu/Pb Reclnr Conc	
		Pb Calc	Pb % Grade	Pb % Dist	Pb % Grade	Pb % Dist
F28	HG B	0.5	54.3	85.7	55.9	77.5
F29	GNDD280 Comp	0.07	11.7	74.9	12.6	57.4
F30	GNDD264 Comp	0.66	45.5	74.7	41.8	57.1
F31	GNDD-227 Comp	0.12	10.4	72.3	27.8	55.5

To determine the overall plant recovery from the different streams in sequential flotation, the testwork results from test F28 were modelled using Bilco mass balancing and data smoothing software. A schematic of the testwork flowsheet is shown in Figure 11-6 and the balanced results are shown in Table 11-8.

The model shows:

- Combined gravity with cleaner tailings concentrate grading 118 g/t Au (gold recovery of 63%).
- Zinc concentrate grading 50.5% (zinc recovery of 89%).
- Lead concentrate grading 64.6% (lead recovery of 76.8%).
- A combined tailings stream grading 1.64 g/t gold (gold distribution of 16.9%)Gold recovery of 69% from cyanide leaching of these combined tailings.

Figure 11-6: Sequential Zinc Flotation Flowsheet for Composite HGB (Test F28)

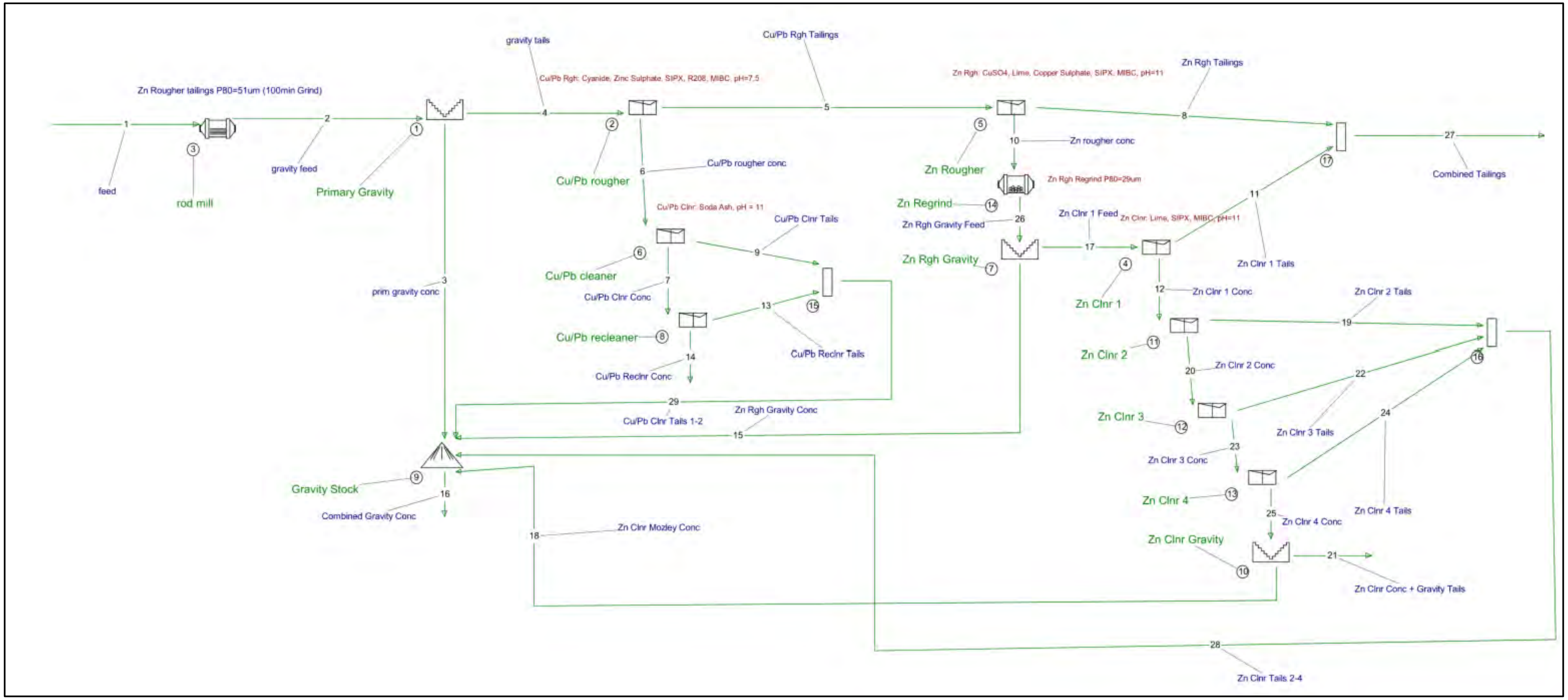


Table 11-8: Sequential Zinc Flotation Results for Composite B (Test F28)

Stream definition	Estimated value/HG B /Recovery/Quantity (%)	Estimated value/HG B /Ratio criterion 1 /Gold (g/t)	Estimated value/HG B /Recovery/Gold (%)	Estimated value/ HG B /Ratio criterion 1/ Silver (g/t)	Estimated value/ HG B /Recovery/Silver (%)	Estimated value/HG B /Ratio criterion 1/ Copper (%)	Estimated value/ HG B /Recovery/Copper (%)	Estimated value/HG B /Ratio criterion 1/ Lead (%)	Estimated value/HG B /Recovery/Lead (%)	Estimated value/HG B /Ratio criterion 1/Zinc (%)	Estimated value/HG B /Recovery/Zinc (%)	Estimated value/HG B /Ratio criterion 1/ Sulphur (%)	Estimated value/HG B /Recovery/Sulphur (%)
Feed	100	8.5106	100	39.63	100	0.14052	100	0.49246	100	3.8986	100	10.125	100
Primary Gravity Conc	0.093756	3728.9	41.079	2526.5	5.9771	0.05	0.03336	42.995	8.1856	0.47	0.011303	23.9	0.22132
Cu/Pb Reclnr Conc	0.58548	177.79	12.231	764.69	11.297	0.40998	1.7082	64.631	76.839	5.59	0.8395	15.9	0.91945
Combined Gravity Conc	4.5601	117.8	63.121	286.6	32.978	0.76686	24.886	1.2991	12.029	6.4754	7.5743	26.157	11.781
Zn Clnr Conc + Gravity Tails	6.8691	9.5927	7.7426	177.8	30.819	1.3279	64.911	0.47996	6.6948	50.542	89.054	33.904	23.002
Combined Tailings	87.989	1.6351	16.905	11.217	24.906	0.013567	8.4951	0.024831	4.4367	0.11221	2.5325	7.3985	64.298

11.5.3 Bulk Flotation Testwork

Results from the high grade Manto Style limestone hosted material (HG A) flotation test (F5) and the low grade dacite hosted (LG DAC) composite flotation test (F10) have been used to estimate the performance of material through the bulk flotation process plant which has been used as the basis of the process design for bulk flotation circuit. These results are considered appropriate for recovery prediction for this material at Scoping Study level of design. In the next project phase, locked cycle testing will be conducted along with a variability program for a more accurate estimate of process plant recovery.

LG DAC was generated from combining material from 5 intervals to represent the low grade dacite material, a summary of the recipe for this composite is shown in Table 11-9.

Table 11-9: Low grade Dacite Hosted Composite Interval Data (LG DAC)

Drillhole	From	To	Length	Au_ppm	Ag_ppm	Cu_ppm	Pb_ppm	Zn_%	Fe_%	S_%
GNDD113	154.0	161.5	7.5	0.86	32.03	566	1264	0.18	1.73	1.18
GNDD113A	352.0	360.0	8.0	1.06	0.90	10	60	0.02	2.27	1.20
GNDD155	195.0	200.0	5.0	0.92	1.26	26	187	0.10	2.17	1.00
GNDD155A	248.0	253.0	5.0	1.39	0.95	14	82	0.07	2.64	1.44
GNDD157	345.0	352.0	7.0	1.27	0.53	12	23	0.11	2.28	1.45
Composite	**154.0**	**360.0**	**32.5**	**1.1**	**7.03**	**124**	**317.0**	**0.1**	**2.21**	**1.25**

HG A was generated from combining materials from 6 intervals to represent the high grade (HG) manto-style, limestone (CAL) hosted, high sulphide material, a summary of the recipe for this composite is shown in Table 11-10.

Table 11-10: High grade (HG) manto-style, limestone (CAL) hosted, high sulphide

Drillhole	From	To	Length	Au_ppm	Ag_ppm	Cu_ppm	Pb_ppm	Zn_%	Fe_%	S_%
GMDD039	67.6	68.6	1.0	24.50	58.30	2700	18350	3.88	17.95	9.32
GMDD040	116.7	125.4	8.7	5.54	11.78	558	27	2.23	13.99	6.98
GMDD041	65.0	68.6	3.6	11.07	110.47	6442	2204	10.64	18.57	17.58
GMDD043	70.5	70.8	0.3	25.90	80.50	3290	31200	9.37	9.24	11.30
GNDD003	55.0	61.1	6.1	34.65	21.92	766	1953	2.86	18.51	11.10
GNDD018	63.2	66.9	3.7	7.05	78.33	2832	36094	3.58	9.52	6.00
Composite	**55.0**	**125.4**	**23.4**	**13.34**	**40.6**	**1968.7**	**5921.6**	**4.07**	**15.27**	**9.61**

A summary of bulk flotation results along with gravity recovery is shown in Table 11-11.

Test	Comp	Primary Grind	Rough Tails	Regrind Grind	Regrind K80 µm	Head g/tg/ tg /t	Primary Gravity Conc Au			Clnr/Reclnr+Gravity		
		min	K80 µm	min		Au Calc	Mass %	Au g/tg/ tg /t	Audist %	Mass %	Au g/tg/ tg /t	Au Rec
F5	HG A	55.0	86.0	-	-	15.8	0.4	1538	40.5	23.9	54.2	82.3
F22	HG B	55.0	68.0	48.0	-	7.7	0.4	566	32.5	12.4	53.6	86.1
F24	Sentazon	55.0	68.0	28.0	29.0	5.5	0.7	273	32.7	11.7	41.5	88.1
F10	LG DAC	55.0	76.0	15.0	29.0	1.1	0.2	418	61.7	2.1	46.8	91.5
F17	LG LUT	55.0	83.0	15.0	20.0	0.7	0.4	67	37.1	2.3	23.6	83.2

Table 11-11: Summary of Bulk Flotation Testwork Results

Test	Comp	Primary Grind	Rough Tails	Regrind Grind	Regrind K80 µm	Head g/t	Primary Gravity Conc Au			Clnr/Reclnr+Gravity		
		min	K80 µm	min		Au Calc	Mass %	Au g/t	Audist %	Mass %	Au g/t	Au Rec
F5	HG A	55.0	86.0	-	-	15.8	0.4	1538	40.5	23.9	54.2	82.3
F22	HG B	55.0	68.0	48.0	-	7.7	0.4	566	32.5	12.4	53.6	86.1
F24	Sentazon	55.0	68.0	28.0	29.0	5.5	0.7	273	32.7	11.7	41.5	88.1
F10	LG DAC	55.0	76.0	15.0	29.0	1.1	0.2	418	61.7	2.1	46.8	91.5
F17	LG LUT	55.0	83.0	15.0	20.0	0.7	0.4	67	37.1	2.3	23.6	83.2

To determine the overall plant recovery from the different streams in bulk flotation, the testwork results from test F10 were modelled using Bilco mass balancing and data smoothing software. A schematic of the testwork flowsheet is shown in Figure 11-7 and the balanced results are shown in Table 11-12.

The model shows:

- Combined gravity with bulk cleaner concentrate grading 42.4 g/t Au (gold recovery of 95%)
- A combined tailings stream grading 0.06 g/t gold (gold distribution of 5%).

Figure 11-7: Bulk Flotation Flowsheet Testwork for LG DAC Composite Sample (Test F10)

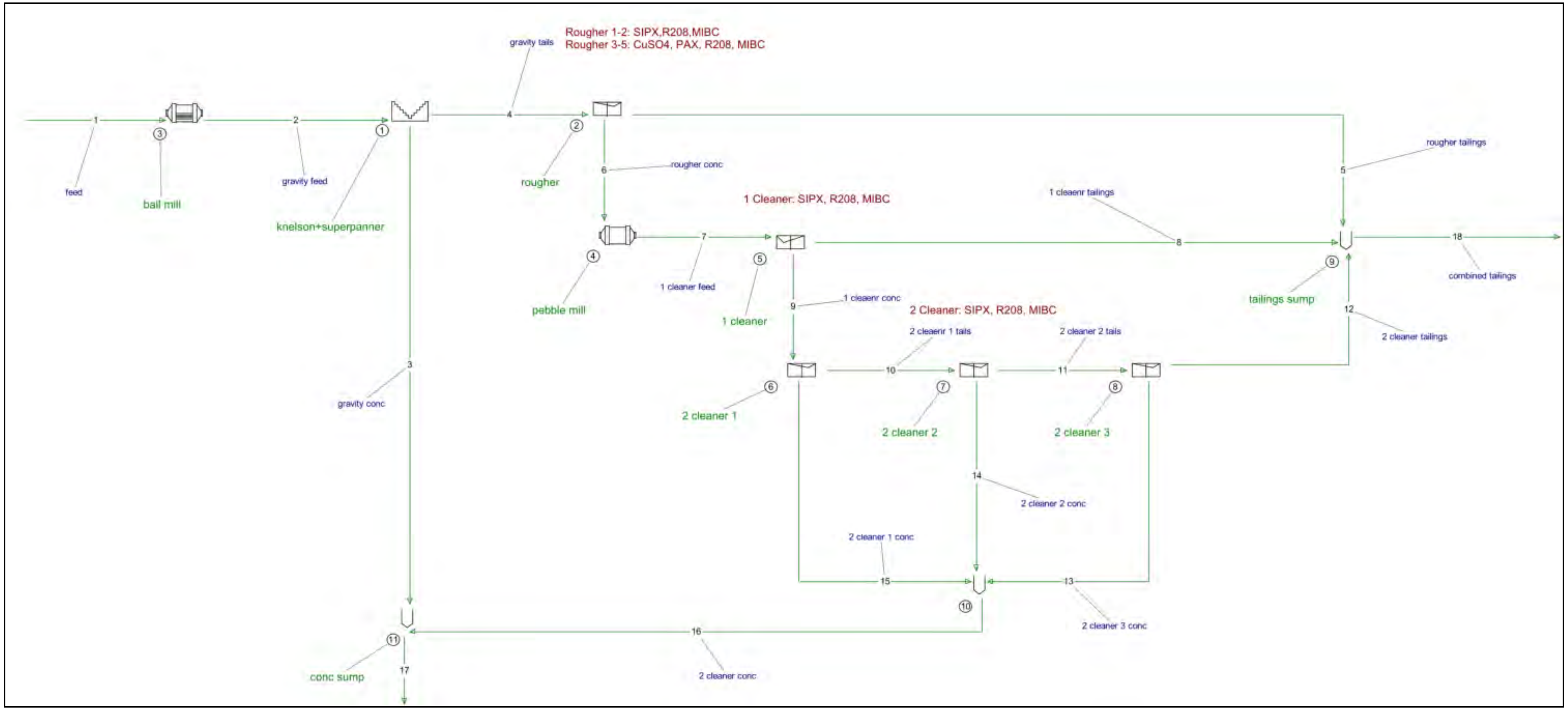


Table 11-12: Bulk Flotation Results for LG DAC Composite Sample (Test F10)

Stream definition	Estimated value / LG / Recovery / Quantity (%)	Estimated value / LG / Ratio criterion 1 / Gold (g/t)	Estimated value / LG / Recovery / Gold (%)	Estimated value / LG / Ratio criterion 1 / Silver (g/t)	Estimated value / LG / Recovery / Silver (%)	Estimated value / LG / Ratio criterion 1 / Copper (%)	Estimated value / LG / Recovery / Copper (%)	Estimated value / LG / Ratio criterion 1 / Lead (%)	Estimated value / LG / Recovery / Lead (%)	Estimated value / LG / Ratio criterion 1 / Zinc (%)	Estimated value / LG / Recovery / Zinc (%)	Estimated value / LG / Ratio criterion 1 / Sulphur (%)	Estimated value / LG / Recovery / Sulphur (%)
Feed	100	1.1858	100	10.858	100	0.019468	100	0.054624	100	0.10712564	100	1.2581	100
Gravity conc	0.16013	468.52	63.27163	1041.7	15.363	0.13004	1.0696	16.075	47.124	0.17996303	0.26901	45.882	5.8398
2 cleaner conc	2.4965	15.077	31.741789	296.68	68.214	0.63215	81.066	0.97079	44.369	2.8310257	65.975	40.932	81.221
Overall conc	2.6566	42.409	95.013419	341.58	83.577	0.60188	82.135	1.8812	91.492	2.6712285	66.244	41.23	87.061
Combined tailings	97.343	0.060744	4.9865807	1.8318	16.423	0.0035728	17.865	0.0047741	8.5078	0.03714779	33.756	0.16724	12.939

Figure 11-8: Bulk Flotation Flowsheet Testwork for HG A Composite Sample (Test F5)

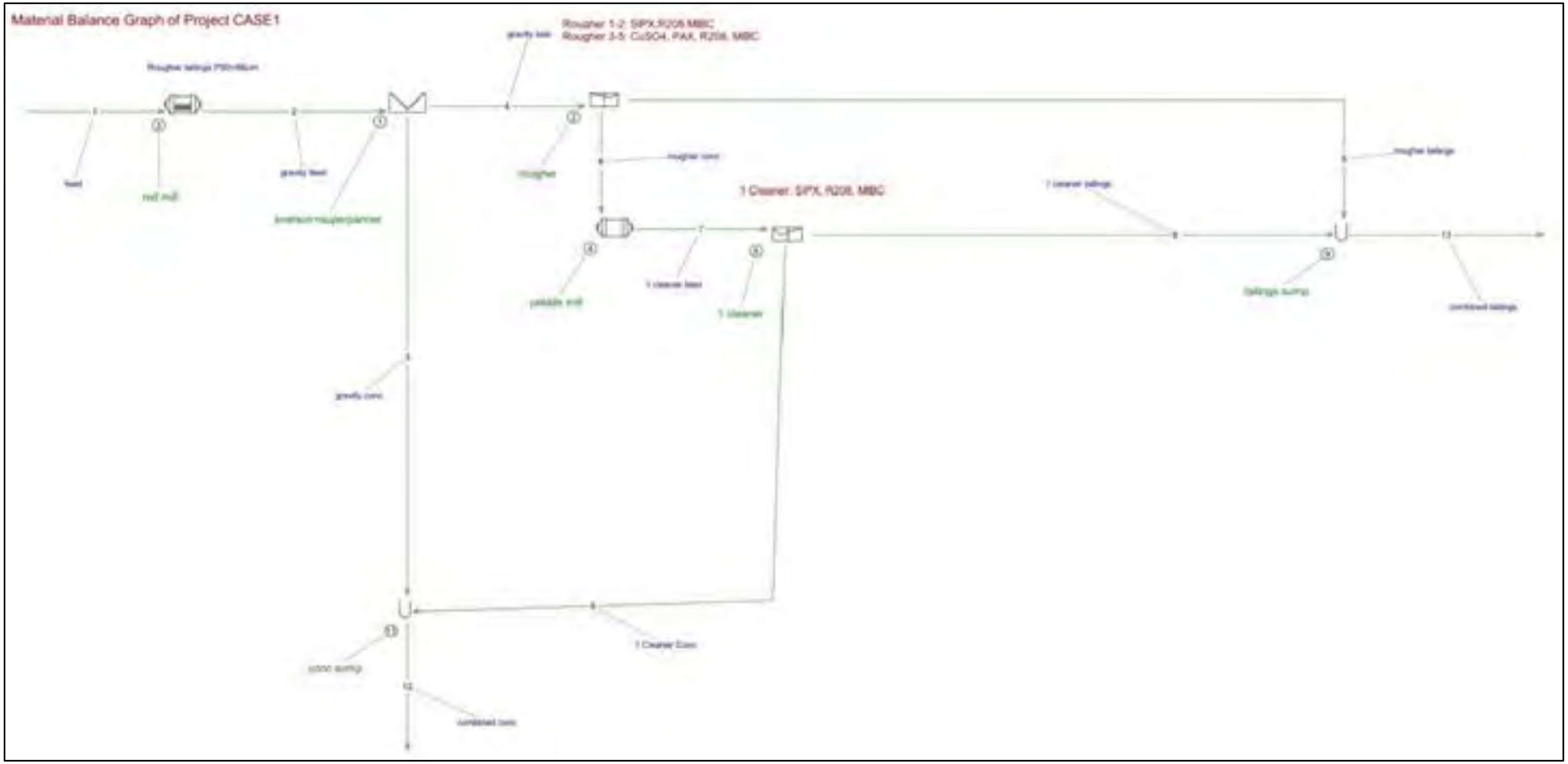


Table 11-13: Bulk Flotation Results for HG A Composite Sample (Test F5)

Stream definition	Estimated value / HG / Recovery / Quantity (%)	Estimated value / HG / Ratio criterion 1 / Gold (ppm)	Estimated value / HG / Recovery / Gold (%)	Estimated value / HG / Ratio criterion 1 / Silver (ppm)	Estimated value / HG / Recovery / Silver (%)	Estimated value / HG / Ratio criterion 1 / Copper (%)	Estimated value / HG / Recovery / Copper (%)	Estimated value / HG / Ratio criterion 1 / Lead (%)	Estimated value / HG / Recovery / Lead (%)	Estimated value / HG / Ratio criterion 1 / Zinc (%)	Estimated value / HG / Recovery / Zinc (%)	Estimated value / HG / Ratio criterion 1 / Sulphur (%)	Estimated value / HG / Recovery / Sulphur (%)
Feed	100	15.443009	100	43.717185	100	0.19994921	100	0.70073863	100	4.3788445	100	10.77624	100
Gravity conc	0.415	1596.3007	42.89739	1018.6283	9.6696694	0.1400001	0.29057401	40.28984	23.860942	0.58000502	0.054969314	32.400887	1.2477792
Rougher tailings	68.65	2.0671341	9.1891909	2.5156696	3.950408	0.030000785	10.300385	0.15997351	15.672293	1.3242981	20.761885	0.53003928	3.3766135
1 cleaner tailings	7.42	7.49459	3.600973	40.617867	6.8939611	0.1900034	7.0509168	0.41998027	4.4470984	2.1512324	3.6452869	2.5700998	1.769647
1 cleaner conc	23.52	29.095131	44.312446	147.74245	79.485961	0.70014634	82.358124	1.6690112	56.019666	14.063288	75.537859	42.887766	93.60596
Combined conc	23.935	56.268321	87.209836	162.84242	89.155631	0.69043417	82.648698	2.3386433	79.880608	13.829507	75.592829	42.705938	94.853739
Combined tailings	76.07	2.5965376	12.790164	6.2322241	10.844369	0.045607718	17.351302	0.18533502	20.119392	1.4049587	24.407171	0.72903033	5.1462606

11.5.4 Leaching Testwork

Results from the leaching of flotation tailings and the leaching of mill discharge material are summarised in Table 11-14. Leaching of flotation tailings from test F28 recovered 69.6% of the contained gold at a cyanide consumption of 0.58 kg/t.

Table 11-14: Cyanide Leaching Results

Test	Circuit	Leach Duration, h	Head Grade		Leach Recovery		Leach Tails		Overall Rec		Cyanide Consumption	
			Au Assay g/t	Ag Assay g/t	Au %	Ag %	Au g/t	Ag g/t	Au %	Ag %	kg/t	Free CN mg/L
CN1	g F14 Ro Tailing	48	3.36	9.78	72.9	72.4	0.91	2.70	72.91	72.41	3.15	488
CN2	g F14 Ro Tailing	48	3.21	9.38	80.7	76.6	0.62	2.20	80.69	76.55	4.62	438
CN3	g F18 Ro Tailing	48	2.81	22.47	78.3	63.5	0.61	8.20	78.30	63.50	7.04	389
CN4	Combined F28 Zn Ro Tailing	48	1.68	9.19	69.6	63.0	0.51	3.40	69.62	63.01	0.58	518
CN5	F34 100 micron Gravity Tailing	72	1.38	8.24	82.0	56.8	0.19	3.30	86.60	59.96	0.55	196
CN6	F34A 100 micron Mill Discharge	72	1.38	8.24	85.2	52.7	0.21	3.90	85.15	52.68	0.72	121
CN7	F35 50 micron Gravity Tailing	72	1.32	8.45	65.0	37.4	0.32	5.10	76.11	39.62	1.00	23
CN8	F35A 50 micron Mill Discharge	72	1.32	8.45	84.1	47.9	0.21	4.40	84.07	47.91	1.04	27

It should be noted that tests CN5 to CN8 were undertaken on a low grade composite designed to mirror the in-pit component of the current Hualilan MRE. Test CN6, which was a cyanide leach test of this composite at a 100 micron grind, shows a recovery of 89.3% for gold after 24 hours compared to the final 72-hour recovery of 85.1% for gold. The difference between the 24- and 72-hour leach recovery possibly indicating preg-robbing (the phenomenon whereby the gold cyanide complex, Au(CN)2), is removed from solution by the constituents of the ore. The preg-robbing components may be carbonaceous matter present in the ore, such as wood chips, organic carbon, or other impurities, such as elemental carbon although these have not be observed.

Potential recoveries of 89% are likely in a standard CIL mill, which has a 24-hour residence and includes carbon addition in the leach for adsorption of gold.

11.5.5 Gravity Gold Recoverable (GRG) Testwork

Both the HG COMM1 and the LG COMM1 composites were subjected to an extended gravity recoverable gold test by applying the standard E-GRG procedure to the sample. The results were then utilised by FLSmidth to model a commercial gravity recovery circuit. The GRG test report is included as an annexure to this study in Appendix 7 – GRG Test Report.

In test GRG-1, the HG COMM1 sample was subjected to four stages of gravity separation with intermittent grinds of the gravity tailings. The corresponding feed P80 sizes were 536 microns, 161 microns, 83 microns, and 44 microns. The concentrates and tailings of each pass were submitted for a 12-fraction size-by-size analysis. The results were used to generate the overall mass balance. A total of 52.5% of the contained gold was recovered in the four stages of gravity separation at a combined grade of 482.4 g/t Au, with the highest recoveries and grades in the 106 micron and 187 micron stages. The total mass pull into the combined gravity concentrate was 2.0%. Eliminating the last stage of gravity separation at the finest grind size reduced the gold recovery marginally to 44.2% and resulted in a gravity concentrate grade increase to 514 g/t.

The second GRG test using the low grade COMM1 sample was performed at grind sizes of 644 microns, 258 microns, 87 microns, and 44 microns. The four stages recovered a total of 82.2% of the contained gold at a grade of 73.7 g/t Au. Eliminating the last stage on the finest product, a high gold recovery of 77.4% was maintained at a higher grade of 86.3 g/t Au.

The two samples tested from the Hualilan project were quite different in both grade and gravity recovery. Both are strong candidates for gravity circuits, but LG COMM1 is more so. Given the high GRG content in LG COMM1 and that the downstream circuit is flotation, a strong gravity effort is highly recommended, treating the entire circulating load.

FLSmidth recommended the installation of two QS48 gravity separators as this allows for gravity to always be utilised even if a single unit is down for maintenance.

12 MINERAL RESOURCE ESTIMATES

The Mineral Resource Estimate (MRE) for Hualilan is dated 29 March 2023 (Table 12-1) is an update of an earlier version dated 1 May 2022.

Table 12-1: Hualilan MRE dated 29 March 2023.

Domain	Category	Mt	Au g/t	Ag g/t	Zn %	Pb %	AuEq[1] g/t	AuEq (Moz)
US$1800 optimised shell ≥ 0.30 ppm AuEq	Indicated	45.5	1.0	5.1	0.38	0.06	1.3	1.9
	Inferred	9.6	1.1	7.3	0.43	0.06	1.4	0.4
Below US$1800 shell ≥1.0ppm AuEq	Indicated	2.7	2.0	9.0	0.89	0.05	2.5	0.2
	Inferred	2.8	2.1	12.4	1.1	0.07	2.8	0.2
Total		**60.6**	**1.1**	**5.8**	**0.43**	**0.06**	**1.4**	**2.8**

Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding.

[1] Gold Equivalent (AuEq) values:

- Assumed commodity prices for the calculation of AuEq is Au US$1,900 Oz, Ag US$24 Oz, Zn US$4,000/t, Pb US$2,000/t
- Metallurgical recoveries are estimated to be Au (95%), Ag (91%), Zn (67%) Pb (58%) across all mineralised material types based on metallurgical test work.
- The formula used: AuEq (g/t) = Au (g/t) + [Ag (g/t) x 0.012106] + [Zn (%) x 0.46204] + [Pb (%) x 0.19961]
- CEL confirms that it is the Company's opinion that all the elements included in the metal equivalents calculation have reasonable potential to be recovered and sold.

12.1 Modelling Procedure

The procedure for estimating the Mineral Resource is:

1. Identification of mineralised domains and controls on mineralisation from surface exposure, drill hole data and channel logging.
2. Establishing 3 domain groups based on mineralisation controls.
3. Wireframing (explicit) was completed in Micromine Origin according to geological and structural controls to a nominal 0.2 g/t AuEq grade boundary.
4. Analysing the SG data measured in the domains for correlations with lithology or assay data and calculating a density formula based on estimated Fe% and S% grades.
5. Extracting 2 m composites from assay data within the mineralised domains (Au, Ag, Fe, Pb, S, Zn) and analysing the composite statistics.
6. Assessing limits for outliers and establishing top cut grades.
7. Establishing variogram parameters for each domain group using group domain structural controls on mineralisation to guide orientation.
8. Building and populating a block model using Ordinary Kriging (grade, density, estimation and Kriging parameters).
9. Analysing the results in section and 3D against the original data and from swath plots.
10. Estimation of parameters to establish possibility of future economic extraction and by what possible methods.

11. Running a pit optimisation at USD$1800 to constrain and report the resource by potential open pit and underground mining scenarios.
12. Determining the cut-off grades for reporting the MRE.

12.1.1 Data Used

The data used for the geological model, mineralisation domains and MRE are:

1. Selected historic drill holes (total of 8,030.40 metres) that are validated by CEL/GMSA drilling as detailed in Section 2 (5 DDH holes from Compañía Minera Aguilar S.A, 10 RC holes and 40 DDH holes from Compañía Minera El Colorado S.A. and 20 DD holes from La Mancha Resource Inc.).
2. Underground and surface channel sampling completed by CEL/GMSA (1,767 samples averaging 1.5 metres per sample).
3. Drilling completed by CEL/GMSA (total of 227,103.60 metres) from October 2019 to January 2023, including 37 RC holes and 762 DDH holes.
4. Drill hole logging (weathering, lithology, alteration, mineralisation, structure).
5. SG measurements of representative drill core samples.
6. Surface mapping and geological modelling of drill hole data.
7. Digital terrain model (DTM) developed from drone survey images, registered to surveyed ground control points and drill holes. The DTM has a pixel cell of 0.1 x 0.1 metres when collected which is reduced to 1 m x 1 m resolution. The elevation precision is also expected to be about 1 m.
8. Nine wireframes of existing underground workings have been built using historic maps, underground surveys and surveyed ground control points. After estimation, these volumes were used to create voids that effectively deplete mined areas in the block model.

12.1.2 Geological Interpretation

There are three dominant host rocks that host the mineralisation. Each has a distinctive structural association which controls mineralisation.

Limestone-hosted domains (CAL)

Mineralisation in the limestone (rock code CAL) is controlled by faults that are either near-parallel to bedding (moderately west dipping) or ramp up through the bedding (thrust faults). The limestone host rock is relatively competent and has low permeability. Movement of hydrothermal fluids are typically controlled by reactivating fault zones, and subsequent skarn alteration of the limestone and mineralisation controls the location of these domains. In some cases, the faults extend along dacite contacts, resulting in some dacite in the CAL domains.

Also included in the CAL domains are the mineralised zones at Magnata Fault and Sanchez Fault. These faults are mineralised in the limestone and the faults have also formed contacts with dacite and younger sedimentary rocks, although limestone is the dominant host rock.

The CAL domains and controls on mineralisation that have been interpreted are shown in Table 12-2.

Table 12-2: CAL domains and controls on mineralisation

Code	Domain Name	Control on Mineralisation
101	sanchez_101	Part of the Sanchez Fault Zone, strike 080°, dip 60°-70° south
102	sanchez_102	Part of the Sanchez Fault Zone, strike 080°, dip 75° south
103	sanchez_103	Part of the Sanchez Fault Zone, strike 080°, dip 45° south
104	sanchez_104	Part of the Sanchez Fault Zone, strike 080°, dip 75° south
115	verde_115	Manto extending at depth into DAC host rocks, strike 020°, dip 55° NW
121	norte_121	Norte Manto strike 030°, dip 40-50° NW, shoots plunging 25° south
201	magnata_M1_M2_201	Magnata Fault Zone, strike 080°, dip north 85°, shallowing to 45° dip at depth
202	magnata_202	Magnata manto, strike 010°, dip 60° west
211	muchilera_211	Muchilera manto, strike 005°, dip 50° west
221	sentazon_221	Sentazon FW manto, strike 175°, dip 55° west
222	sentazon_222	Sentazon HW manto, strike 060° - dip 45° NW, rolling around to strike 150° and dip 45° SW (transform fracture zone)
223	sentazon_223	Sentazon FW manto, strike 065, 60° west, shallowing to 30° at depth and extending into DAC, plunge down dip (60-295) in the upper levels. Interpreted sub-zone dipping east at depth.

Sediment-hosted domains (LUT)

Overlying the limestone is a younger sequence of siltstone and fine-grained sandstone (rock code LUT and ARN respectively). These host rocks have a strong basal fault detachment and faults that are near-parallel to bedding that controlled hydrothermal fluid flow during mineralisation. Skarn alteration of the sediment, brittle fracture around the faults and pervasive fluid flow has resulted in broader low grade mineralised zones. Also, locally developed are fault breccia (cataclasite) zones where mineralisation has formed in the breccia matrix.

The LUT domains and controls on mineralisation that have been interpreted are shown in Table 12-3.

Table 12-3: LUT domains and controls on mineralisation

Code	Domain Name	Control on Mineralisation
111	verde_111	Bedding parallel (minor DAC and CAL at the contact), strike 010° – 015°, dip 40° W
112	verde_112	Bedding parallel – HW split to domain 111 (minor DAC), strike 010 – 015°, dip 40° W
114	verde_114	Bedding parallel, strike 025°, dip 30° NW
212	muchilera	Bedding parallel, strike 010°, dip 40° west
231	magnata HW	Bedding parallel HW splay to Magnata FZ

Intrusion (dacite) hosted mineralisation (DAC)

Mineralisation in the dacite intrusions is controlled by reactivation of faults into the dacite, formation of fracture networks around the faults and pervasive fluid flow, resulting in lower grade vein and disseminated mineralisation. Skarn alteration in the dacite is developed around the larger faults and in the matrix to breccia zones.

The DAC domains and controls on mineralisation that have been interpreted are shown in Table 12-4.

Table 12-4: DAC domains and controls on mineralisation

Code	Domain Name	Control on Mineralisation
113	verde_113	dacite in the LUT, strike 015°, dip 35° NW
131	gap zone_131	dip 75° SE, strike 035°
132	gap zone_132_	dip vertical, strike 030°
133	gap zone_133_	dip 85° NW, strike 030°
134	gap zone_134_	dip 80° NW, strike 030°
203	magnata_203	strike 035°, dip 85° NW
213	muchilera HW_213	dacite in the LUT, strike 160°, dip 50° west
224	sentazon_224	FW to sentazon manto in the dacite - strike 030, 60 west
301	pizzaro_301	dip 50° W strike 000°
302	pizzaro_302	dip 85° W, strike 030°
303	pizzaro_303	dip 50° W, strike 350°
304	pizzaro_304	strike 165°, dip 70° SW - transfer fault zone in dacite and limestone
305	pizzaro_305	strike 005°, dip 85° west
306	pizzaro_306	strike 090° dip 60° west

Model surfaces for the base of transported cover and the top pf fresh rock were generated in Micromine from the drill hole data. The base of transported cover and the DTM were used to constrain the resource model (no mineralisation in the cover). The top of fresh rock surface was used to constrain block density estimates.

12.1.3 Definition of Estimation Domains

Using Micromine Origin software, within each of the domains and following the controls on mineralisation and observing a nominal 0.2 g/t AuEq grade boundary, strings were snapped to the drill holes. The strings were used to build wireframes for each domain. Wireframes were validated in Micromine (Figure 12-1). There were 31 separate wireframes built, some of which have multiple sub-domains (trisolations).

Resource estimation was done using Surpac™ V6.6. The domain wireframes were validated a second time using Surpac.

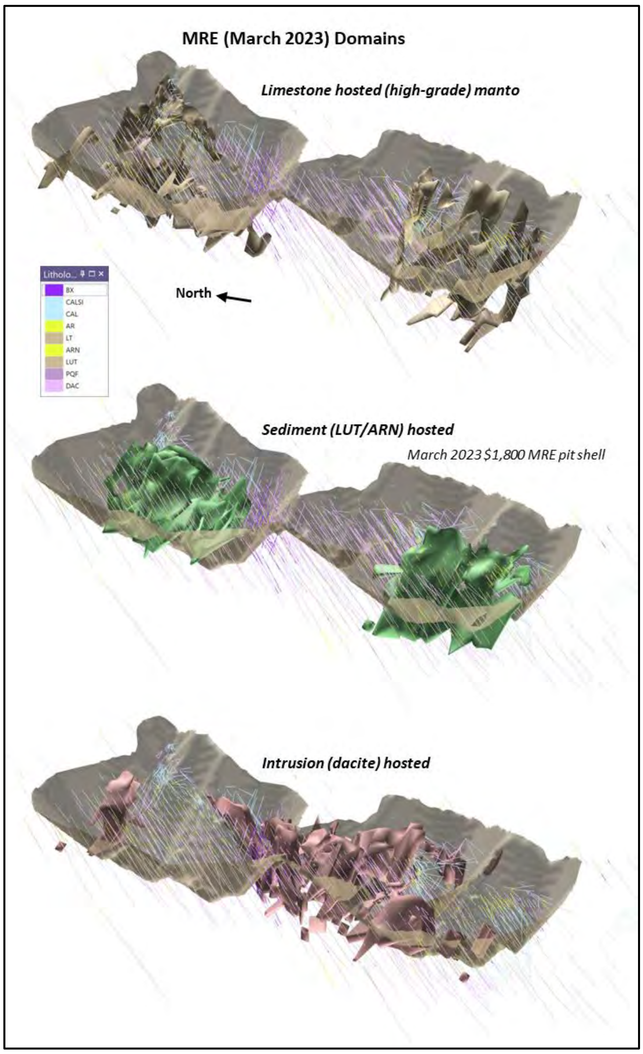


Figure 12-1: Wireframes of mineralised domains grouped according to dominant lithology host with the US$ 1,800 MRE pit shell and drill holes.

12.2 Database

CEL/GMSA Drill hole data were exported from the Azure database. CEL/GMSA Channel and historic drill hole data were exported from MS Excel files.

1. Drill hole data comprised Collar, Survey, Weathering, Lithology, Alteration, Structure, Mineralisation, SG and Assays.
2. Channel sample data comprised, Survey, Lithology, Structure, Mineralisation and Assays.
3. Historic drill hole data comprised Collar, Survey, Lithology and Assays.

The 3 data sets were loaded into an Access database and an audit was run on the merged data. A small number of errors were found which appeared to be mainly data entry related. The errors were checked and fixed.

A domain number of 999 was assigned to all drill hole intervals outside the domain wireframes.

12.2.1 Other Elements

The MRE includes estimates of Au, Ag, Pb and Zn being the elements of economic interest. Fe and S were also estimated to allow calculation of density.

12.3 Compositing, Statistics and Outliers

A two metre composite length was selected after reviewing the original sample lengths from the drilling which have an average length of 1.54 m for samples taken within the mineralised domains (Table 12-5).

Table 12-5: Composite assay statistics for samples in mineralised domains

2m composites	Au (ppm)	Ag (ppm)	Zn (%)	Pb (%)
Number	14450	14346	14335	14346
Minimum value	0.001	0.01	0.0002	0.0001
Maximum value	301.51	1364.62	29.68	9.49
Mean	1.09	6.70	0.49	0.07
Median	0.19	1.00	0.05	0.01
Geometric Mean	0.19	1.16	0.05	0.01
Variance	31.51	1159.61	3.64	0.10
Standard Deviation	5.61	34.05	1.91	0.31
Coefficient of variation	5.14	5.08	3.87	4.51

A statistical analysis was undertaken on the sample composites top cuts for Au, Ag, Zn and Pb composites on a domain-by-domain basis. The domains were then grouped by host rock and mineralisation style and group domain top cuts were applied to reduce the influence of extreme values on the resource estimates without downgrading the high grade composites too severely (Table 12-6). The top-cut values were chosen by assessing the high-end distribution of the grade population within each group and selecting the value above which the distribution became erratic. The following table shows the top cuts applied to each group and domain for Au, Ag, Zn and Pb. No top cut was applied to estimation of Fe and S.

Table 12-6: Top cuts applied to the composite data prior to estimation

Group	Au (ppm)	Ag (ppm)	Zn (%)	Pb (%)
Fault Zone hosted (Magnata and Sanchez) and CAL (limestone) hosted	80	300	20	5
LUT (siltstone) hosted	20	100	5	1
DAC (intrusive) hosted	15	70	5	1.8

12.4 Contact Analysis

Figure 12-2 summarises the Au and Ag grade at the domain contacts.

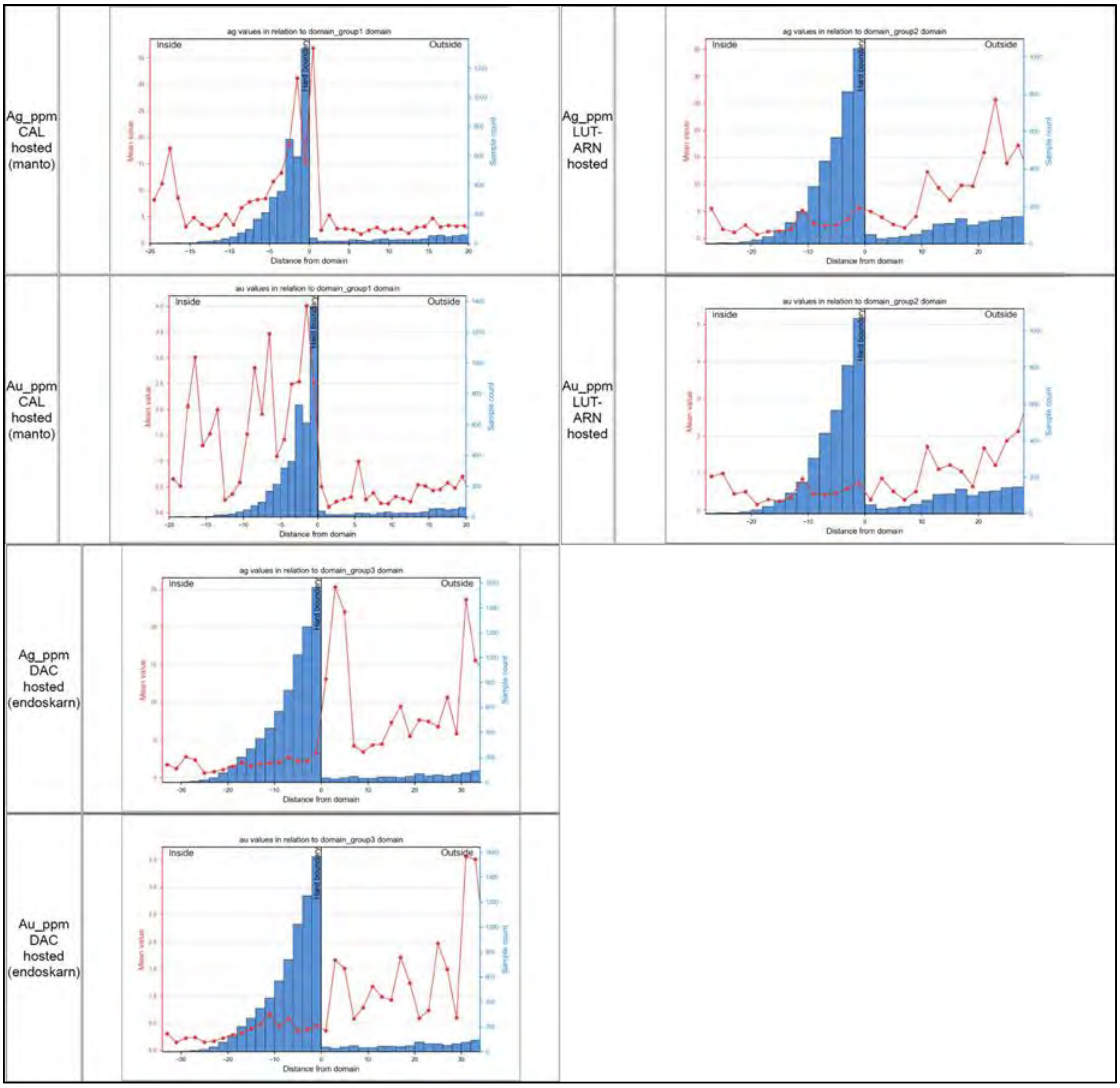


Figure 12-2: Contact graphs for Au and Ag for the 3 group domain types.

12.5 Variography

Group variography was carried out using Leapfrog Edge software on the two metre composited data from each of the 31 domains for each variable.

For each domain, the orientation of the plane of mineralisation was aligned with the interpreted wireframe. The plunge direction for the lodes was generally down dip within the plane of mineralisation. The search ellipse orientation were defined per-domain and were used for the variogram directions.

Variograms were modelled in three directions for each domain group and each element as well as a downhole direction. The nugget effect was modelled by extrapolation of the first two experimental data points from the down-hole variogram set at the same lag as the composite length (2 m).

12.6 Block Model and Resource Estimation Plan

A block model was set up with a parent cell size of 10 m (E) x 20 m (N) x 10 m (RL) with standard sub-celling to 2.5 m (E) x 5.0 m (N) x 2.5 m (RL) to maintain the resolution of the mineralised domains. The 20 m Y and vertical block dimensions were chosen to reflect drill hole spacing and to provide definition for potential mine planning. The shorter 10 m X dimension was used to reflect the geometry and orientation of the majority of the domain wireframes.

An oriented "ellipsoid" search for each domain was used to select data for interpolation.

All relevant variables (Au, Ag, Pb, Zn, Fe and S) in each domain were estimated using Ordinary Kriging using only data from within that domain. The orientation of the search ellipse and variogram model was controlled using surfaces designed to reflect the local orientation of the mineralised structures.

A three passes estimation search was conducted, with expanding search ellipsoid dimensions and decreasing minimum number of samples with each successive pass. First passes were conducted with ellipsoid radii corresponding to 40% of the complete range of variogram structures for the variable being estimated. Pass 2 was conducted with 60% of the complete range of variogram structures for the variable being estimated. Pass three was conducted with dimensions corresponding to 200% of the semi-variogram model ranges. Blocks within the model where Au was not estimated during the first 3 passes were assigned as unclassified. Blocks for Ag, Pb, Zn, Fe and S that were not estimated were assigned the average values on a per-domain basis.

For the estimation of the orphan grades lying outside the domain wireframes, a short search ellipse range of 20 m was applied to these composites with an orientation of 50° dip to 200° dip azimuth and a major-semi major ratio of 1 and minor axis ratio of 3.

Nine volumes of historic mine development were used to create voids in the block model. The void areas were assigned a zero density and zero grade for all elements estimated.

12.6.1 In Situ Dry Bulk Density Estimation

Specific gravity (SG) measurements from drill core were made during the logging of the core. These measurements have been used to estimate block densities for the Resource estimate.

Within the mineralised domains there are 956 SG measurements made on drill core samples of 0.1 – 0.2 metres length. Measurements were determined on a dry basis by measuring the difference in sample weight in water and weight in air. For porous samples, the weight in water was measured after wrapping the sample so that no water enters the void space during weighing.

In oxidised and partially oxidised rocks, SG clusters around an average of 2.49 g/cc (2,490 kg/m^3) which is independent of depth and independent of Fe and S interval assay values (Figure 12-3). A uniform density of 2,490 kg/m^3 has been used for oxidised, fracture oxidised and partially oxidised blocks.

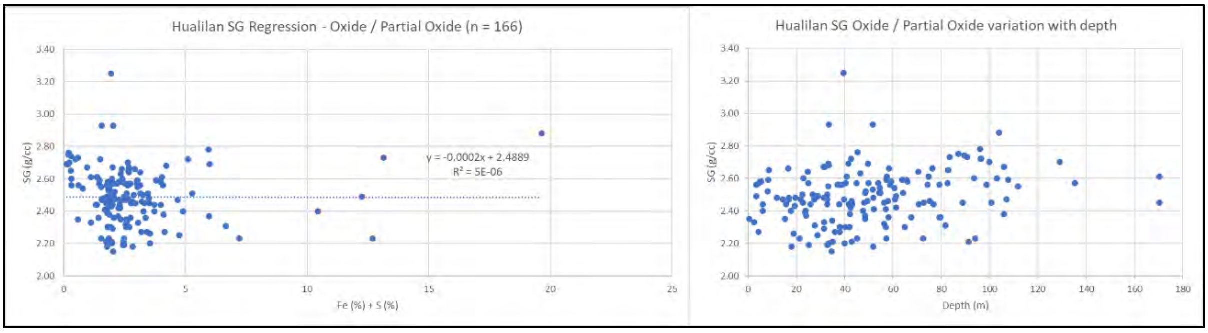


Figure 12-3: Plots of SG in partially oxidised and fracture oxidised rock with Fe + S (%) and depth.

For fresh rock, a regression model for block density determination has been made by plotting assay Fe (%) + S (%) from the interval where the SG measurement was made against the SG measurement. Fe and S are the two elements that form pyrite which is the mineral that is commonly associated with gold and base metal mineralisation at Hualilan. SG plotted against (Fe+S %) follows a linear trend within the mineralised domains (Figure 12-4).

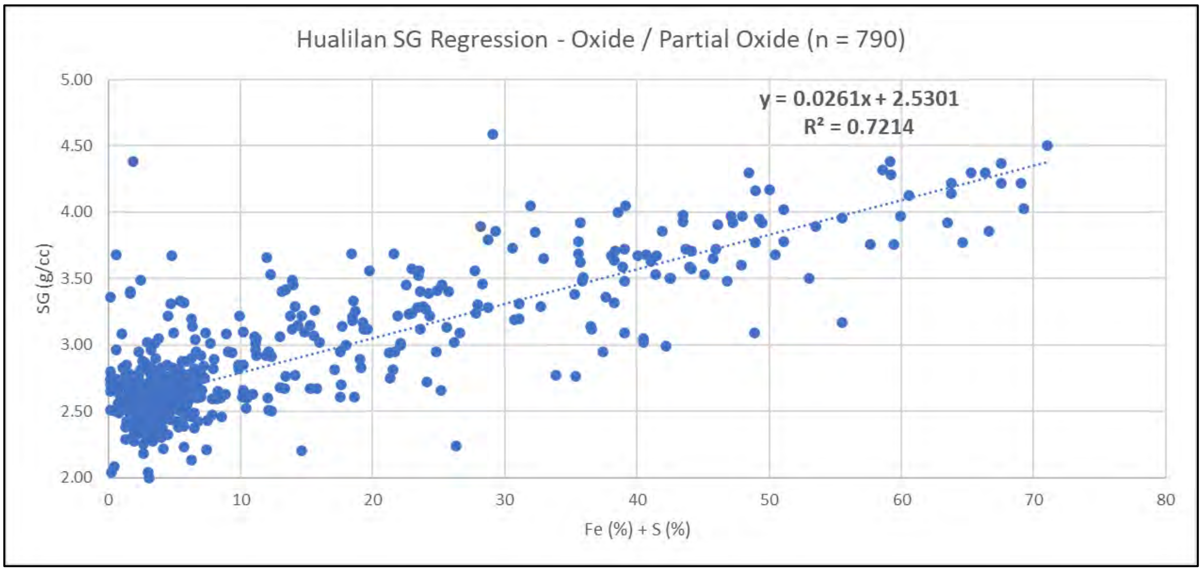


Figure 12-4: Plot of SG against interval Fe + S (%) and linear regression used as a model to assign block density values.

For fresh rock at zero Fe + S (%) the density is assumed to be 2,530 kg/m^3 (2.52 g/cc). The regression slope has a linear increase in density of 26.1 kg/m^3 (0.0261 g/cc) for each 1 percent increase in Fe + S (%). The formula used for block density (kg/m^3) determination in oxide rock is 2,530 + 26.1 x (Fe % + S%).

For the POX zone, a fixed average value of 2.49 (2,490 km/m^3) was used which was very close to the POX zone density used in the previous MRE. A fixed density of 2.0g/cc was used for the cover material.

12.7 Validations

Validation checks included statistical comparison between drill sample grades and Ordinary Kriging block estimate results for each domain on standard 40 metre spaced sections.

A swath plot was generated (Figure 12-5) on 100 m spaced northings to compare sample composite input grades with estimated block model output grades. This showed good correlation except on one Northing (Y) at the Norte manto (domain 121) where the block model grades have smoothed out some very high grade composites from the underground channel samples.

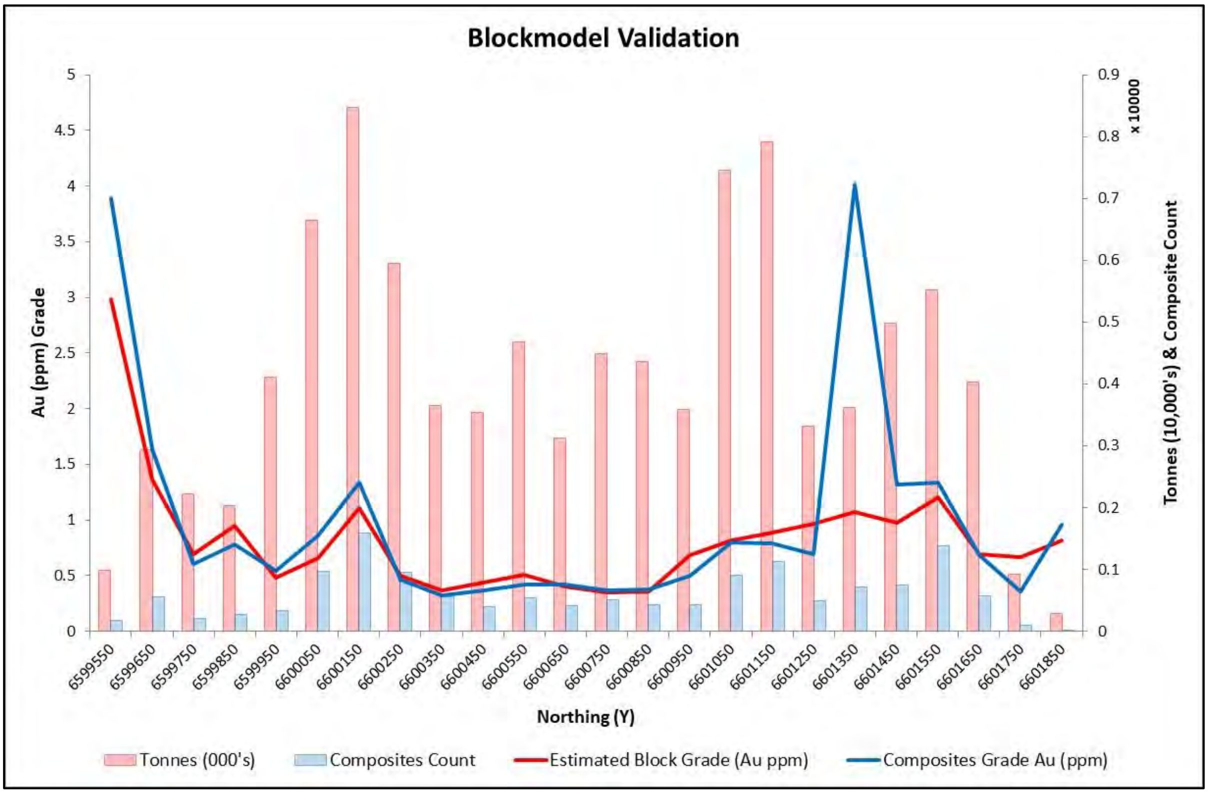


Figure 12-5: Swath plot for Hualilan MRE.

12.7.1 Global Bias

Global bias has not been reviewed for the Hualilan MRE. Only the Ordinary Kriging method has been used, completed by a well-qualified resource estimation consultant in conjunction with in-house geologists who understand the geological uncertainties within the deposit. The application of other estimation techniques, including non-linear estimation methods and peer review is expected to be the subject of additional studies as part of the pre-feasibility. There is no production data available that can be used to compare with the current resource estimates.

12.7.2 Drift

Drift has not been studied in detail at Hualilan. Drift can be estimated using comparisons of two Ordinary Krig estimates that have been completed at Hualilan, summarised at various cut-off grades for the global estimate (Table 12-7). The most recent estimate includes extension and infill drilling, and so provides a partial guide on the impact of closer spaced drilling. At lower cut-off grades there is a significant increase in tonnage and lower average grades as the drilling reached the periphery of the deposit. At higher cut-off grades, infill drilling has increased the tonnage marginally and average grade has remained stable suggesting drift is small.

Table 12-7: Hualilan global MRE's at various AuEq cut-off grades

	1 May 2022 MRE					29 March 2023 MRE				
AuEq cutoff	Mt	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Mt	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)
0.25	47.7	1.10	6.0	0.46	0.06	83.9	0.91	4.9	0.36	0.05
0.30	42.7	1.20	6.5	0.50	0.07	74.3	0.99	5.3	0.39	0.06
0.50	29.6	1.60	8.3	0.66	0.08	46.3	1.41	7.1	0.55	0.07
1.00	16.5	2.44	11.8	1.02	0.11	21.1	2.46	10.9	0.97	0.10
1.50	10.8	3.21	14.9	1.37	0.14	13.6	3.30	13.5	1.30	0.12
2.00	8.0	3.87	17.2	1.68	0.16	9.9	4.02	15.8	1.57	0.13
2.50	6.3	4.43	19.4	1.95	0.17	7.7	4.64	17.8	1.82	0.14
3.00	4.9	5.05	21.9	2.27	0.19	6.4	5.15	19.0	1.99	0.15

Drift can also be estimated by analysing key statistical indicators of 2 m composite samples from the two Resource Estimates completed from within the mineralised domains as shown in Table 12-8. Lower mean, bi-weight, Sichel-t and tri-mean values relate to drilling of extensions of the deposit at the lower grade extremities rather than a change in drift. Marginally lower variance and standard deviation are a function of closer spaced composites as a result of infill drilling which suggest drift is small.

Table 12-8: Key statistical measures of 2m composites from the 1 May 2022 and 29 March 2023 Ordinary Kriging Resource Estimates for Hualilan

	2022 2m Composites				2023 2m Composites			
	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)
Number of samples	9829	9701	9260	9260	14450	14346	14335	14346
Mean	**1.33**	**7.91**	**0.59**	**0.08**	**1.09**	**6.70**	**0.49**	**0.07**
Median	0.18	1.06	0.05	0.01	0.19	1.00	0.05	0.01
Geometric Mean	0.19	1.27	0.06	0.01	0.19	1.16	0.05	0.01
Variance	45.85	1454.42	4.70	0.11	31.51	1159.61	3.64	0.10

	2022 2m Composites				2023 2m Composites			
	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)
Standard Deviation	6.77	38.14	2.17	0.33	5.61	34.05	1.91	0.31
Coefficient of variation	5.11	4.82	3.68	4.41	5.14	5.08	3.87	4.51
Skewness	18.539	17.544	6.483	13.822	21.054	19.282	7.186	14.230
Natural Log Mean	-1.667	0.239	-2.808	-4.589	-1.681	0.146	-2.926	-4.650
Log Variance	3.256	2.917	3.717	3.357	2.912	2.766	3.601	3.355
Trimean	0.238	1.494	0.072	0.013	0.230	1.357	0.067	0.013
Biweight	0.206	1.260	0.060	0.011	0.202	1.166	0.057	0.011
MAD	0.168	0.987	0.051	0.010	0.158	0.913	0.049	0.009
Alpha	-0.001	-0.012	0.000	0.000	0.003	-0.010	0.000	0.000
Sichel-t	0.961	5.459	0.387	0.054	0.799	4.615	0.324	0.051

12.7.3 Visual Validation

Visual validation of grade trends for each element along the drill sections was completed. These checks show good correlation between estimated block grades and drill sample grades.

Thickness (Figure 12-6) and grade (Figure 12-7) trends correlate with known mineralised zones.

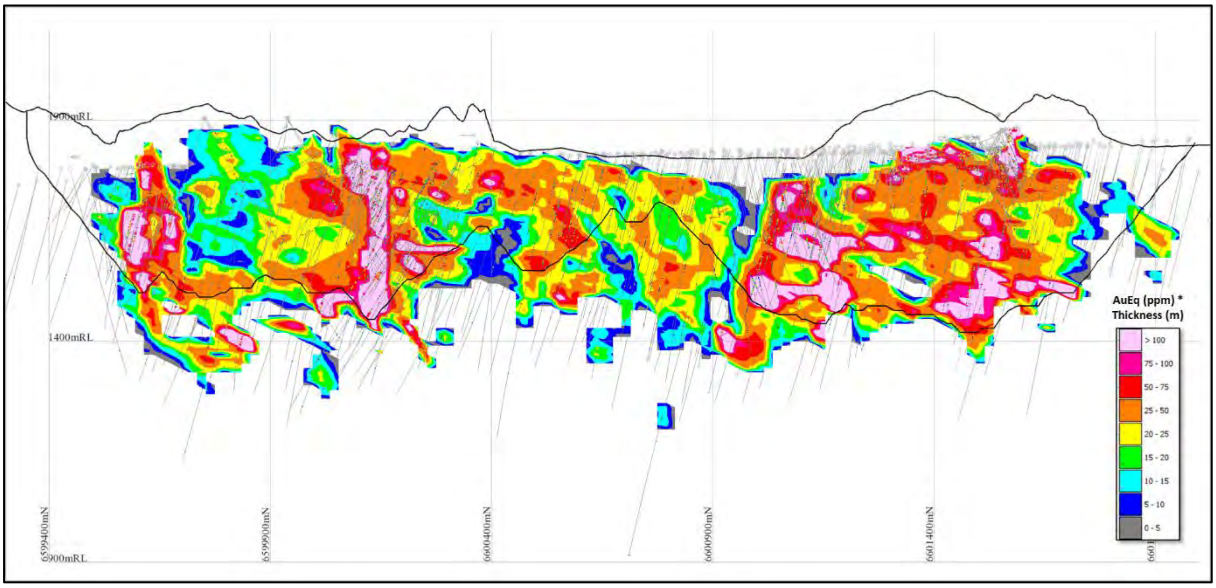


Figure 12-6: Long section block thickness above cut-off for the Hualilan MRE.

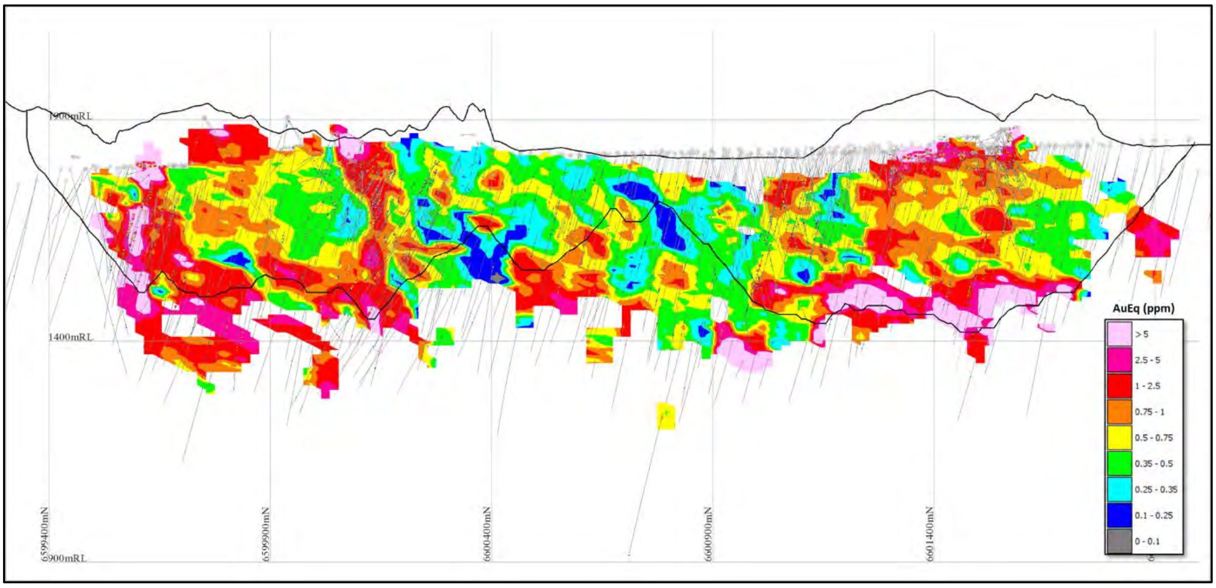


Figure 12-7: Long section block thickness AuEq grade above cut-off for the Hualilan MRE.

12.8 Resource Classification

The estimation search strategy was undertaken in three separate passes with different search distances, and the minimum number of samples used to estimate blocks which were then used as a guide for the classification of the resource into Indicated, Inferred and Unclassified.

The classification was then visually assessed and further modified to restrict the Indicated Resource to domains with closer spaced drilling.

The Mineral Resource has been classified (Figure 12-8) based on the guidelines specified in the JORC Code 2012. The classification level is based upon semi-qualitative assessment of the geological understanding of the deposit, geological and mineralisation continuity, drill hole spacing, QC results, search and interpolation parameters and an analysis of available density information.

Orphan grades in domain 999 within the optimised pit were classed as Inferred and were excluded outside the pit.

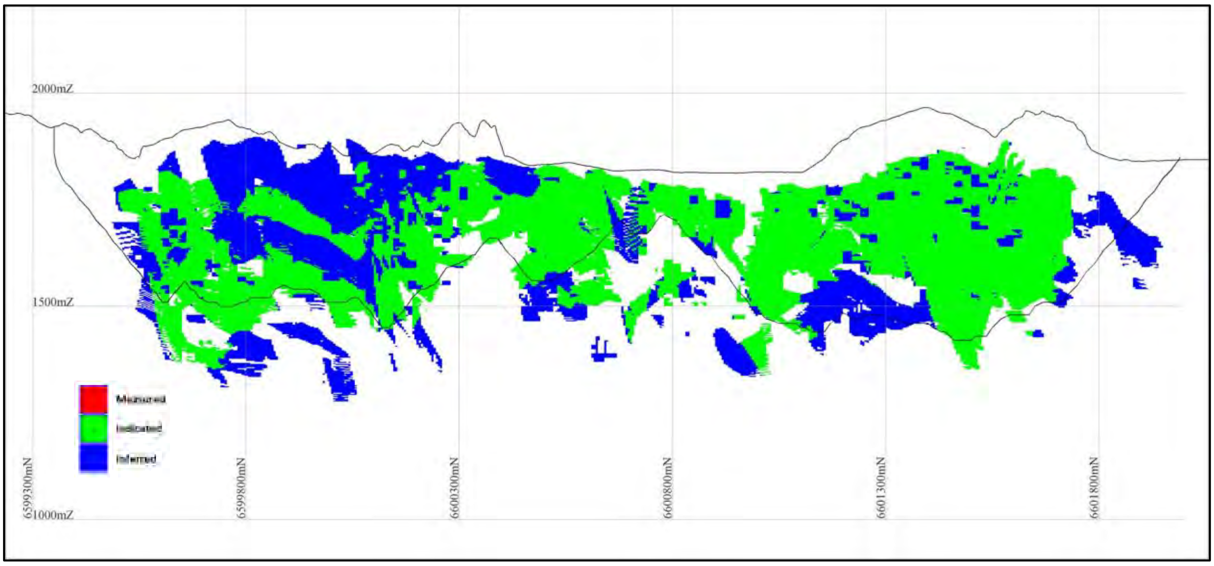


Figure 12-8: Long section block classification for the Hualilan MRE.

12.9 Resource Tabulation

The Resource estimate has assumed that near surface mineralisation would be amenable to open pit mining given that the mineralisation is exposed at surface and under relatively thin unconsolidated cover. A surface pit optimiser has been used to determine the proportion of the Resource estimate model that would be amenable to eventual economic extraction by open pit mining methods. The surface pit optimiser was built using the following parameters with prices in USD:

- Au price of $1,800 per oz, Ag price of $23.4 per oz, Zn price of $3,825 per tonne and Pb price of $1,980 per tonne.
- Average metallurgical recoveries of 94.9% for Au, 90.9% for Ag and 67% for Zn and 57.8% for Pb.
- Ore and waste mining cost of $2.00 per tonne.
- Unconsolidated cover removal cost of $0.10 per tonne.
- Processing cost of $10.00 per tonne.
- Transport and marketing of $50/oz of AuEq (road to San Juan then rail to Rosario Port).
- Royalty of $60 per oz Au, 3% for Ag, Zn and Pb.
- Assumed concentrate payability of 94.1% for Au, 82.9% for Ag, 90% for Zn and 95% for Pb.
- 45° pit slopes on the western side of the pit and 55° on the eastern side of the pit.

The following metals and metal prices have been used to report gold grade equivalent (AuEq): Au US$1900/oz, Ag US$24/oz, Zn US$4,000/t and Pb US$2,000/t.

Average metallurgical recoveries for Au, Ag, Zn and Pb have been estimated from the results of Stage 1 metallurgical test work completed by SGS Metallurgical Operations in Lakefield, Ontario using a combination of gravity and flotation combined metallurgical samples as detailed in the Criteria below.

For the AuEq calculation average metallurgical recovery is estimated as 94.9% for gold, 90.9% for silver, 67.0% for Zn and 57.8% for Pb.

Accordingly, the formula used for Au Equivalent is: AuEq (g/t) = Au (g/t) + [Ag (g/t) x (24/1900) x (0.909/0.949)] + [Zn (%) x (40.00 x 31.1/1900) x (0.670/0.949)] + (Pb (%) x 20.00 x 31.1/1900) x (0.578/.9490).

A summary of the MRE by domain and dominant lithology is shown for the pit at 0.30 g/t AuEq cut off in Table 12-9 and at 1.0 g/t AuEq cut-off in Table 12-10.

Table 12-9: Summary of the MRE by domain and domain group in the US$1,800 pit at 0.30 g/t AuEq cut off.

Domain in pit ≥ 0.30 g/t AuEq	ID	Tonnes ('000)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Au Eq (g/t)	Au ('000 ozs)	Fe (%)	S (%)	Density (g/cc)
Limestone (CAL)											
101_sanchez	101	271	1.1	4.7	0.35	0.12	1.3	11	2.23	0.23	2.53
102_sanchez	102	37	0.38	3.4	0.19	0.04	0.52	1	1.35	0.05	2.54
103_sanchez	103	194	4.9	9.4	0.60	0.16	5.3	33	2.50	0.39	2.50
104_sanchez	104	158	1.6	6.4	2.6	0.06	2.8	14	2.56	0.39	2.51
115_verde	115	4,490	2.8	5.1	0.60	0.03	3.2	459	2.85	2.02	2.66
121_norte_manto	121	1,140	2.2	16.3	1.4	0.17	3.1	114	2.64	0.96	2.58
201_magnata	201	3,547	1.9	15.6	1.1	0.18	2.6	297	3.90	2.45	2.67
202_magnata_manto	202	510	2.3	32.1	1.1	0.28	3.2	53	2.93	1.41	2.53
211_muchilera	211	490	0.86	21.4	0.88	0.19	1.6	25	2.87	0.71	2.56
221_sentazon	221	422	4.5	17.5	3.0	0.12	6.1	83	9.42	3.91	2.77
222_sentazon	222	448	2.3	37.5	3.6	0.15	4.4	64	9.43	8.16	2.98
223_sentazon	223	465	5.4	12.1	1.8	0.03	6.3	95	7.14	4.70	2.82
		12,172	**2.5**	**13.0**	**1.1**	**0.11**	**3.2**	**1,250**	**3.75**	**2.27**	**2.66**
Sediment (LUT/ARN)											
111_verde	111	7,824	0.71	2.1	0.24	0.06	0.86	215	2.86	0.85	2.61
112_verde_HW	112	2,010	0.72	2.2	0.14	0.05	0.82	53	2.49	0.73	2.60
114_verde	114	4,524	0.78	2.6	0.12	0.03	0.87	127	2.87	0.97	2.63
212_muchilera_LUT	212	6,089	0.46	6.6	0.22	0.05	0.65	128	3.11	0.85	2.62
231_magnata_HW	231	594	1.6	4.5	0.20	0.08	1.7	33	3.24	1.34	2.65
		21,040	**0.68**	**3.6**	**0.20**	**0.05**	**0.82**	**556**	**2.91**	**0.88**	**2.62**
Dacite (DAC)											
113_verde	113	1,931	0.46	1.4	0.28	0.15	0.63	39	2.49	0.93	2.57
131_gap	131	286	0.37	4.9	0.22	0.10	0.56	5	2.18	0.69	2.61
132_gap	132	2,150	0.89	2.0	0.14	0.03	0.98	68	2.21	0.90	2.61
133_gap	133	353	0.57	1.9	0.24	0.02	0.71	8	2.05	1.08	2.61
134_gap	134	1,137	0.64	1.3	0.09	0.01	0.70	26	2.13	0.82	2.59
203_magnata_DAC	203	443	0.48	2.5	0.18	0.03	0.60	9	2.00	0.69	2.56
213_muchilera_HW	213	2,914	0.46	4.6	0.07	0.02	0.55	51	2.70	1.38	2.63
301_pizzaro	301	7,560	0.50	3.0	0.13	0.03	0.60	146	2.24	0.71	2.56
302_pizzaro	302	346	1.1	4.8	0.39	0.06	1.4	15	2.94	1.83	2.64
303_pizzaro	303	1,207	0.74	4.3	0.34	0.07	0.97	38	2.27	1.26	2.62
304_pizzaro	304	38	0.84	7.9	0.69	0.09	1.3	2	3.91	3.41	2.72
306_pizzaro	306	1,047	0.38	6.5	0.11	0.02	0.51	17	2.96	1.07	2.63

Domain in pit ≥ 0.30 g/t AuEq	ID	Tonnes ('000)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Au Eq (g/t)	Au ('000 ozs)	Fe (%)	S (%)	Density (g/cc)
		19,412	**0.52**	**2.9**	**0.14**	**0.04**	**0.63**	**424**	**2.19**	**0.87**	**2.39**
Orphans	999	2,514	0.60	3.1	0.20	0.04	0.74	60	2.02	0.64	2.58
Total		**55,138**	**1.0**	**5.5**	**0.39**	**0.06**	**1.3**	**2,289**	**2.86**	**1.20**	**2.61**

Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding.

Table 12-10: Summary of the MRE by domain and domain group below the US$1,800 pit at 1.0 g/t AuEq cut off.

Domain below pit ≥ 1.0 g/t AuEq	ID	Tonnes ('000)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	AuEq (g/t)	Au ('000 ozs)	Fe (%)	S (%)	Density (g/cc)
Limestone (CAL)											
101_sanchez	101	34	1.7	1.0	0.06	0.00	1.7	1.8	2.04	0.05	2.54
103_sanchez	103	19	1.7	5.0	0.39	0.05	1.9	1.2	1.72	0.05	2.52
104_sanchez	104	10	0.76	3.9	1.7	0.03	1.6	0.5	1.52	0.23	2.56
115_verde	115	1,002	3.0	9.7	1.3	0.02	3.7	119.8	3.36	2.43	2.68
121_norte_manto	121	140	2.2	17.4	0.70	0.35	2.8	12.7	1.41	1.22	2.60
201_magnata	201	482	1.9	14.2	2.0	0.11	3.0	46.3	5.87	4.33	2.80
211_muchilera	211	313	1.3	17.9	0.78	0.17	1.9	19.2	4.20	3.13	2.72
222_sentazon	222	229	2.0	33.7	3.0	0.18	3.8	28.1	8.97	9.01	3.00
223_sentazon	223	1,039	2.1	11.6	1.1	0.03	2.7	90.8	6.50	4.68	2.82
224_sentazon	224	87	1.1	1.4	0.04	0.03	1.2	3.3	3.48	1.22	2.65
		3,355	**2.2**	**13.3**	**1.3**	**0.07**	**3.0**	**323.6**	**5.05**	**3.79**	**2.76**
Sediment (LUT/ARN)											
111_verde	111	39	2.1	9.3	0.24	0.03	2.4	3.0	2.82	1.67	2.65
212_muchilera_LUT	212	598	2.5	12.4	0.24	0.03	2.8	53.4	3.41	2.60	2.69
231_magnata_HW	231	79	1.7	4.0	0.15	0.05	1.8	4.6	2.25	0.82	2.61
		716	**2.4**	**11.3**	**0.23**	**0.03**	**2.7**	**61.0**	**3.25**	**2.36**	**2.68**
Dacite (DAC)											
113_verde	113	8	0.30	9.7	1.4	0.12	1.1	0.3	2.09	0.70	2.60
131_gap	131	4	0.81	4.0	0.58	0.07	1.1	0.1	2.47	1.42	2.63
132_gap	132	217	1.5	1.5	0.21	0.02	1.6	11.1	2.21	1.21	2.62
133_gap	133	153	1.3	2.3	0.35	0.09	1.5	7.4	2.27	1.23	2.62
134_gap	134	216	1.3	4.7	0.69	0.03	1.6	11.4	4.40	2.59	2.71
203_magnata_DAC	203	65	1.4	1.0	1.3	0.00	2.1	4.3	5.33	3.25	2.75
213_muchilera_HW	213	31	2.3	1.9	0.04	0.01	2.3	2.3	4.68	2.04	2.71

Domain below pit ≥ 1.0 g/t AuEq	ID	Tonnes ('000)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	AuEq (g/t)	Au ('000 ozs)	Fe (%)	S (%)	Density (g/cc)
301_pizzaro	301	60	1.3	6.4	0.39	0.04	1.5	2.9	2.86	0.94	2.63
303_pizzaro	303	248	1.1	6.9	0.61	0.07	1.5	12.2	3.29	1.89	2.67
304_pizzaro	304	359	1.9	6.0	0.90	0.06	2.4	27.7	4.54	3.15	2.73
305_pizzaro	305	76	1.9	3.8	0.89	0.05	2.3	5.7	2.95	2.03	2.65
306_pizzaro	306	3	1.0	1.3	0.05	0.02	1.0	0.1	2.83	1.20	2.64
		1,441	**1.5**	**4.48**	**0.64**	**0.05**	**1.8**	**85.7**	**3.57**	**2.16**	**2.68**
Total		**5,511**	**2.1**	**10.7**	**1.0**	**0.06**	**2.7**	**470.3**	**4.43**	**3.18**	**2.73**

Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding.

Grade – tonnage curves for the MRE in the pit shell and below the pit shell are shown in Figure 12-9.

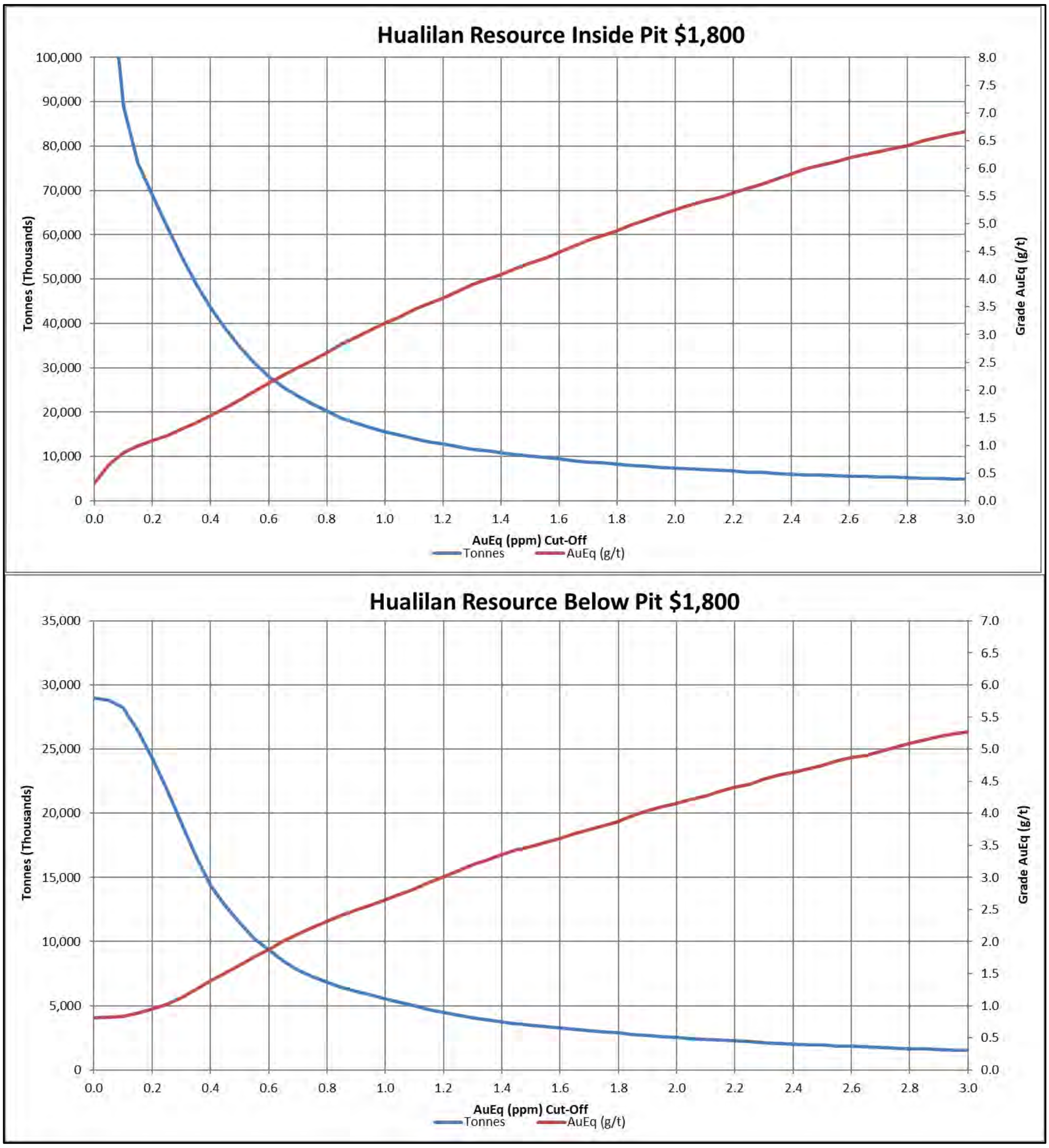


Figure 12-9: AuEq grade-tonnage curves for the Hualilan MRE.

12.10 Mine Optimisation

No mining optimisation was completed for the MRE.

Based on the break-even grade for an optimised gold equivalent pit shell, an AuEq cut-off grade of 0.30 ppm is used to report the resource within the pit shell at a gold price of US$1,800 per ounce and allowing for Ag, Zn and Pb credits. Under this scenario, blocks with a grade above the 0.30 g/t AuEq cut off are considered to have reasonable prospects of mining by open pit methods.

An AuEq cut-off grade of 1.0 ppm was used to report the resource beneath the optimised pit shell run as these blocks are considered to have reasonable prospects of future mining by underground methods.

12.11 Comparison with Prior Estimates

CEL completed a prior MRE, dated 1 May 2022 using part of the data that was used for the MRE dated 29 March 2023 (Table 12-11) and used for this scoping study. The prior estimate used the same techniques as escribed above but was reported at 0.25 g/t AuEq for the pit optimisation component.

Table 12-11: MRE for Hualilan dated 1 May 2022.

Domain	Category	Mt	Au g/t	Ag g/t	Zn %	Pb %	AuEq[1] g/t	AuEq (Mozs)
US$1800 optimised shell ≥ 0.25 ppm AuEq	Indicated	18.7	1.1	5.4	0.41	0.07	1.3	0.80
	Inferred	25.0	1.0	5.6	0.39	0.06	1.2	1.00
Below US$1800 shell ≥1.0ppm AuEq	Inferred	4.0	1.9	11.5	1.04	0.07	2.6	0.33
Total		**47.7**	**1.1**	**6.0**	**0.45**	**0.06**	**1.4**	**2.13**

Data is reported to significant figures to reflect appropriate precision and may not sum precisely due to rounding.

[1] Gold Equivalent (AuEq) values:

- Assumed commodity prices for the calculation of AuEq is Au US$1900 Oz, Ag US$24 Oz, Zn US$4,000/t, Pb US$2000/t
- Metallurgical recoveries are estimated to be Au (95%), Ag (91%), Zn (67%) Pb (58%) across all mineralised material types based on metallurgical test work.
- The formula used: AuEq (g/t) = Au (g/t) + [Ag (g/t) x 0.012106] + [Zn (%) x 0.46204] + [Pb (%) x 0.19961]
- CEL confirms that it is the Company's opinion that all the elements included in the metal equivalents calculation have reasonable potential to be recovered and sold.

No production data is available for comparison.

13 GEOTECHNICAL

13.1 Geotechnical Parameters

The geotechnical evaluation was focused on the surface mining method, taking as reference the report AKL Ingeniería y Geomecánica Ltda. (AKL) "IG GMSA 2023-01 Geotechnical Evaluation Hualilan OP Stage Conceptual Engineering" (AKL Ingenieria y Geomecanica Ltda., 2023).

During the economic evaluation of the project a case was developed for underground mining. For the underground mining support and geomechanical support an intermediate rock was considered since there are no geomechanical studies and reports.

The concepts related to the qualification of the condition or degree of slope stability used are shown in Table 13-1.

Table 13-1: Definition of concepts of the degree of stability of a slope.

Concept	Definition
Design Acceptance Criteria, DAC	*Corresponds to the set of requirements that a slope must meet for its design to be considered acceptable. Generally, the acceptability criteria depend on the magnitude and consequences of eventual slope instability and are defined in terms of minimum or maximum permissible values for one or more of the following parameters: safety factor, safety margin, probability of failure, and /or index of reliability.*
Factor of Factor, FoS	*Corresponds to the ratio between the resistance of the material and the stress acting on it. FoS is dimensionless and is generally defined in terms of its mean value at a potential rupture surface. According to this, if FoS is greater than 1.0 there is a stable or "no failure" condition; if FoS is equal to 1.0, there is a condition of "limit equilibrium" or "incipient failure"; and if FoS is less than 1.0 you have a fault or instability condition. It is generally accepted that the higher the value of FoS, the lower the probability of failure.*
Failure of a Mining Slope	*A condition where part of the material that makes up the slope suffers excessive or inadmissible displacements, modifying the geometry of the slope in such a way that it affects the normal operation of the sector. Failure can be fast or slow. The volume of material affected, usually proportional to the consequences of the failure, is delimited by the geometry of the slope and by a rupture surface or limit of the flow zone.*
Probability of Failure, PoF	*It corresponds to the probability that a mining slope failure will occur. It is generally defined as the probability that the FoS is equal to or less than 1.0.*

The geometry of a mining slope is determined by the parameters listed in Table 13-2 with support from Figure 13-1.

Table 13-2: Parameters that define the geometry of a mining slope.

Parameter	Definition
Bench height, h_B	It is usually defined by operational considerations (loading equipment efficiency), and not for geotechnical parameters.

Parameter	Definition
Bench face slope, a_B	This is generally defined by the structures present in the rock mass at bench level, but it also strongly depends on the quality of the blasting and the damage induced in the rock mass.
Berm Width, B	Typically defined by the volume of spills associated with structurally controlled instabilities at the bench level, which must be contained by the berms.
Inter-ramp angle, a_{IR}	This corresponds to the inclination with respect to the horizontal of an imaginary line that joins the toe of the benches. This value is commonly used for mining planning and, although it does not correspond to the geotechnical inclination of the inter-ramp slope, it has the advantage of not varying with the number of benches. The inter-ramp angle is determined by the geometry of the bench-berm system.
Inter-ramp height, h_{IR}	Corresponds to the maximum allowable height between ramps. It is usually defined by geotechnical parameters.
Ramp width, R	Defined for operational reasons, associated with transportation equipment.
Global angle (OSA), a_O	Corresponds to the angle that defines the wall of the pit, measured as the inclination with respect to the horizontal of an imaginary line that joins the toe of the lower bench with the crest of the upper bench of the wall in the considered sector.
Overall height, h_O	Corresponds to the height of the pit wall, measured from the toe of the lower bench to the crest of the upper bench of the wall in the considered sector.

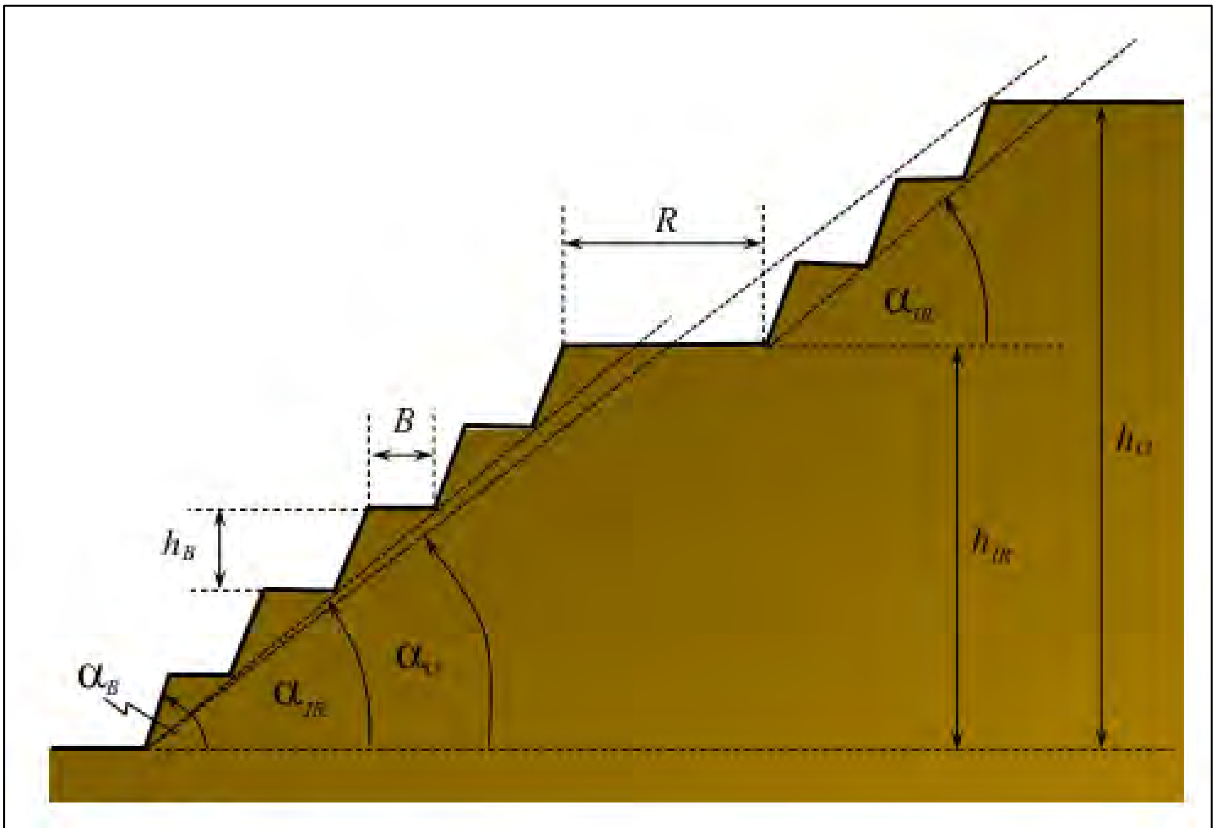


Figure 13-1: Parameters that define the geometry of a mining slope.

13.2 Geological Structures

The north-trending and west-dipping low-angle faults are located along the eastern and western margins of the Hualilan Hills parallel to the regional strike and similar in-plane orientation. Propagated by compressive forces from the west, these thrusts were probably accompanied by in-plane faulting on a smaller scale that may have provided pathways for the introduction of porphyry dacite dykes and mineralisation.

A later set of faults trend roughly east-west to east-northeast across the regional strike. These are recognized as high-angle normal faults with normal displacements in the order of 10 or 20 m. Significant apparent transverse movement has occurred along these structures, as indicated by lateral displacement of bedding on the order of 250 to 300 metres. Although initially formed prior to the emplacement of porphyries, veins, and layers, these crosscutting structures have had an extensive history of activation and reactivation that extends beyond the mineralisation phases. Ground water springs mark the location of the Hualilan thrust fault, ten kilometres south of the deposit.

Local faults are shown in Figure 13-2. These faults don't compromise the stability of the walls of the future proposed Hualilan pits, because either they are perpendicular to the pit walls, or they pass through the East leg of the pit, or on the crest of the inside hill.

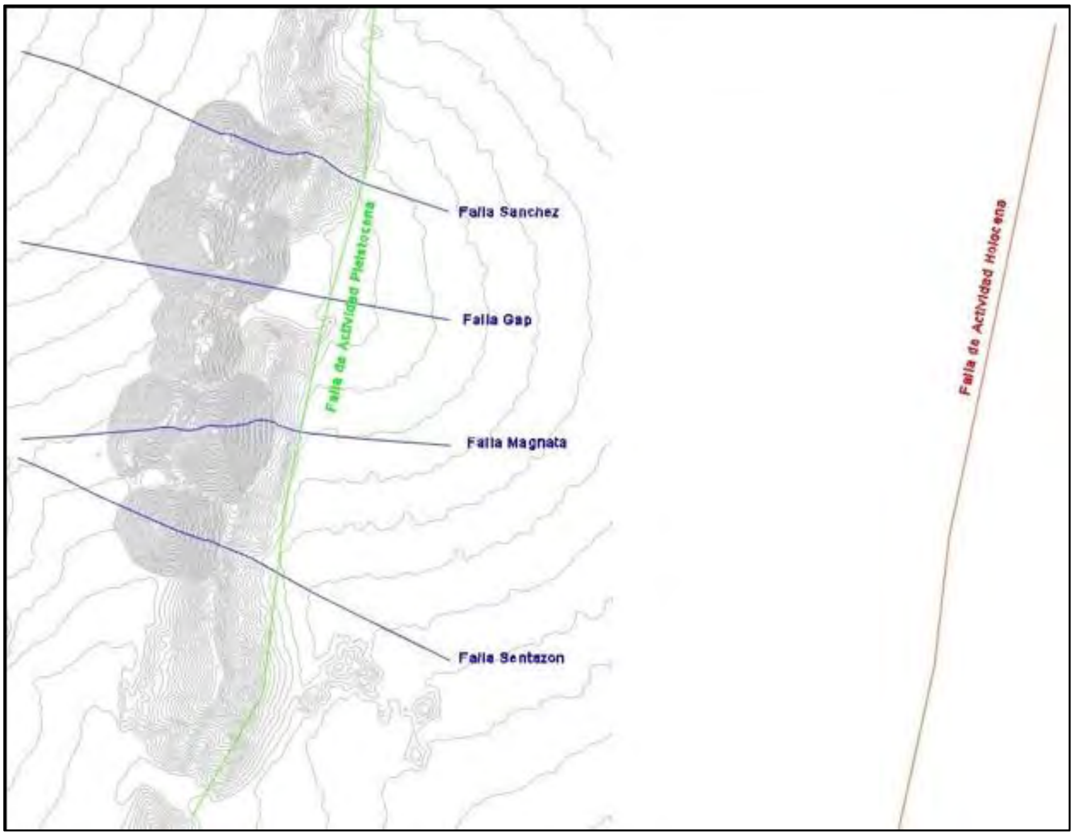


Figure 13-2: Plan that shows the Local and Regional Faults. Regional-type faults pass behind the pit walls and local faults are pseudo-perpendicular to the pit walls or cut into the toe or inside the crest of the hill

13.3 Regional Seismicity

For the purposes of evaluating the seismic hazard and to evaluate seismic movements for the purposes of verifying the proposed project, baseline information has been collected from:

- Neotectonic Analysis of the Hualilan Mining Project
- NEIC / USGS Earthquake Catalog
- CERESIS Earthquake Catalogue
- PEER Strong Motion Database
- ISC (International Seismological Centre) Database.

Based on the information collected, the seismogenic sources considered in the present study are classified as:

1. Subduction interface (Nazca Plate under South American Plate)
2. Subducted Plate
3. Crust – Eastern Precordillera and Sierras Pampeanas
4. Crust – Central Precordillera and Western Precordillera
5. Crust – Front Range and Main Range.

For the characterisation of the seismogenic sources, the geometry and maximum magnitude were considered.

For evaluation of the seismic risk, a search was conducted of past events within 100 kilometres of the project. Seismogenic sources with the least favourable scenarios being maximum magnitude and minimum distance were prioritised (related to the subducted plate, La Cantera Fault and El Tigre Fault).

To estimate the seismic movement created by past events, the appropriate attenuation laws for each source were used. The study includes mean spectra of horizontal pseudo-acceleration with 5% damping, for each of the sources considered, calculated as the average of the pseudo-acceleration values.

The analysis of the seismological information included a review of the earthquake catalogue for the area in study (INPRES, CERESIS, NEIC, ISC, Seismological Service of the University of Chile, Servicio Geológico Minero Argentino). The characterisation of seismic activity over time was evaluated through the parameters of the Guttenberg-Richter Law. In addition, the recurrence of earthquakes was established for the five seismogenic sources considered in this study.

Probabilistic analysis of the seismic risk for 18 seismogenic sources uses the methodology proposed by Cornell (1968), Esteva (1969) and Merz and Cornell (1973), applying specific formulas for crustal and subduction sources.

Seismic verification criteria and verification scenarios have been defined for infrastructure (critical and non-critical), as well as a selection of seismic movements for verification of earthquake safety for critical structures.

For the evaluation of the seismic hazard of an area, it is necessary to recognize seismogenic sources that intervene, actively or potentially, on the seismicity. An active fault is a geological structure that it is likely to produce an earthquake in the near future and has recent movement (historical, Holocene, Quaternary and/or within the recent seismotectonic regime), (Project Andean Multinational, 2009).

Active faults are distinguished by separating the proven neotectonic faults from the inferred faults. Those faults that manifested an obvious failure were considered proven or observed. Faults that modify the geometry or changes in the arrangement of the quaternary deposits, or presence of some fault element, either with stratigraphic or morphological evidence are inferred.

Each potential seismogenic source was digitised, determining those parameters necessary for the subsequent analysis of seismic hazard.

The parameters recorded were:

- movement direction
- distance to the project
- orientation of the fault plane
- length
- estimated rupture width
- rupture surface
- displacement estimate
- age of most recent displacement
- displacement rate
- recurrence interval, and
- estimated variation of Maximum Earthquake (Mw).

The geological and geometric characteristics for each potential seismogenic source were obtained from field observations. The width and estimated rupture area for each fault were determined based on the empirical formulas proposed by (Wells & Coppersmith, 1994).

Displacement rates and/or fault recurrence information were estimated from the available bibliography (Siame, et al., 1997) (Siame, et al., 1997) (Siame, et al., 2002) (Perucca & Vargas, 2014) (Perucca, Rothis, & Vargas, 2014) (Perucca, Rothis, & Vargas, 2014) (Perucca, Rothis, Bezerra, Vargas, & Lima, 2015) (Servicio Geológico Minero Argentino (SEGEMAR)).

To obtain the Mw for each potential seismogenic source, the methodologies proposed by (Wells & Coppersmith, 1994) and (Anderson, Wesnousky, & Stirling, 1996) were used based on the average of the magnitude relationships of the earthquake-rupture of the fault. The parameters used in these methods were surface rupture length ((Wells & Coppersmith, 1994), considering a 95% correlation coefficient), and surface break length and displacement rate (Anderson, Wesnousky, & Stirling, 1996) when the latter data was available.

1) M= a + b x log (LRS)

where LRS = Surface Rupture Length (Wells & Coppersmith, 1994) a and b are constants whose values are: a=5.08 and b=1.16.

2) M = 5.12 + 1.16 x logL – 0.2 log S

where L = Rupture length and S = slip rate (Anderson, Wesnousky, & Stirling, 1996)

To determine the estimated width of the Rupture (downdip rupture width) in km, the following equation was used:

3) log RW= a + b x M (Wells & Coppersmith, 1994)

where RW = Estimated width of the rupture (km) (Wells & Coppersmith, 1994). a and b are constant whose values for reverse faults (practically all those surveyed) are a= -1.61 and b=0.41.

To calculate the rupture area in km^2, the following formula was applied.

4) log(RA) = a + b x M (Wells & Coppersmith, 1994)

where RA = Rupture Area in km^2, a and b are constants whose values for reverse faults are a=3.99 and b=0.98.

Furthermore, since for reverse faults the relationships used by (Wells & Coppersmith, 1994) very poorly correlate rupture length with coseismic displacement (because these displacements for landslides are very scarce), to evaluate earthquakes that occurred in structures that did not produce a significant failure surface, the following relationship between the rupture area (RA) and the magnitude (Mw).

5) Mw = 4.33 quartet + 0.90 log (RA)

13.4 Material Properties

In the case of the Hualilan project, it is considered in this stage of conceptual engineering that the geotechnical units will be defined only based on lithology. In the next engineering stages, the alteration and mineralisation component should be reviewed as it may or may not affect the geotechnical units.

Lithology units considered are:

- Breccia
- Dacite
- Sandstone
- Shale
- Limestone
- Colluvium.

13.5 Rock Mass Characterisation

Information is only available from exploratory drilling where RQD was measured for each lithological unit. Together with this information, more information from the literature, especially the Guidelines for Open Pit Design, and more information on these lithologies in other mining operations where similar studies have been completed, the geotechnical properties of the rock mass were broadly defined.

To evaluate the resistance of the rock mass corresponding to the different geotechnical units present in the slopes to be analysed, the process was as follows:

- The information was calculated and reviewed and those values that were dispersed beyond one standard deviation were "cleaned". The data count of each of the lithologies were reduced and remaining data used as shown in Table 13-3.
- Using RQD values, a Rock Mass Quality value was assigned according to the Guidelines for Open Pit Design (Table 13-3**)**. This treatment is sufficient for this stage of conceptual engineering. For future work, certified analyses from rock mechanics laboratories are required for representative samples.
- Cohesion and friction were assigned according to the Rock Mass Quality.

Values obtained from similar mining projects with similar characteristics and lithological units were also verified, to see if the values are within the expected scenario.

It is assumed that there would be a strip of disturbed material of the order of 5 to 10 metres behind the pit slope as a result of production blasting. For this reason, there are high confinement (AC) and low confinement (BC) properties for geotechnical units. That is, towards the edge of the slopes or close to them, the low confinement strip can be seen, which affects the properties of the rock mass due to relaxation and damage from blasting.

In accordance with the above, the geomechanical properties of the rock mass were estimated for each geotechnical unit. The results obtained are summarised in Table 13-4.

Table 13-3: Average RQD And Rock Mass Quality By Geotechnical Unit.

Unit	Code	Data n°	Average RQD	Rock Mass Quality
Breccia	BX	2,831	80%	GOOD geotechnical quality
Dacite	DAC	34,576	58%	FAIR geotechnical quality
Sandstone	ARN	4,104	56%	FAIR geotechnical quality
Shale	LUT	13,232	57%	FAIR geotechnical quality
Limestone	CAL	23,267	64%	FAIR geotechnical quality
Colluvium	COV	N/A	N/A	N/A

Table 13-4: Properties of Geotechnical Units for the Hualilan Project.

		Estimated Hualilan Project Values				Other Similar Projects			
		ϕ		C kPa		ϕ		C kPa	
Unit	Code	AC	BC	AC	BC	AC	BC	AC	BC
Breccia	BX	45	40	400	300	30	34	1,380	340
Dacite	DAC	45	40	400	300	49	60	5,690	2,270
Sandstone	ARN	35	30	300	225	29	40	2,160	390
Shale	LUT	35	30	300	225	31	39	720	300
Limestone	CAL	35	30	300	225	32	40	850	320
Colluvium	COV	37	35	80	60	37	37	140	110
FAULT		35		40		35		40	

13.6 Pit Failure Mechanisms and Stability Modelling

The acceptability criteria recommended by Read and Stacey (2009) are mentioned in Table 13-5.

Table 13-5: Minimum acceptability criteria recommended by Read and Stacey (2009).

		ACCEPTANCE CRITERION [a]		
SLOPE SCALE	CONSEQUENCES OF FAILURE	FoS (min) (Static)	FoS (min.) (Dynamic)	PoF (max) P(FoS<1.0)
Bench	Low-High [b]	1.10	NA	25-50%
Inter-ramp	Low	1.15-1.20	1.00	25%
	Moderate	1.20	1.00	20%
	High	1.20-1.30	1.10	10%
Overall slope	Low	1.20-1.30	1.00	15-20%

		ACCEPTANCE CRITERION [a]		
SLOPE SCALE	CONSEQUENCES OF FAILURE	FoS (min) (Static)	FoS (min.) (Dynamic)	PoF (max) P(FoS<1.0)
	Moderate	1.30	1.05	10%
	High	1.30-1.50	1.10	5%

Note: [a] Must meet all acceptance criteria.
[b] Semi-quantitatively evaluated.
The values in the yellow blocks are those used in the Ingeroc 2022 report

The Design Acceptance Criteria (DAC) proposed for this study is shown in Table 13-6.

Regarding the interpretation of these criteria, it should be noted that:

- These acceptability criteria are relatively conservative, given that currently mining slopes in the open pit mining industry are considering FoS ≥1.20, for global slopes and inter-ramps.
- For this study, the value of FoS ≥1.25 was considered for global slopes, since the information on rock mass properties is mainly estimated according to the level of engineering under study (conceptual in this case). In the future, for the following levels, the quality of the information and the uncertainties of the same will be improved. In this case, the FoS could be reduced to standard values and with this the design slope angles can be optimised.
- If the evaluated designs meet the respective criteria, the designs are acceptable; otherwise they must be modified. Additionally, if the designs present much higher results than the respective acceptability criteria, it is feasible to optimise them.
- The occurrence of instabilities at the bench level is, in practice, inevitable. In accordance with this, the acceptability criterion should not be defined in terms of the Factor of Safety, FoS, but rather based on a maximum permissible Probability of Failure (PoF) and considering the distribution of volumes in the definition of the berm width of these minor instabilities, such that the berm can contain such a volume that its probability of exceedance is small.
- In addition, the smaller the volume affected by the instability, the less its relevance, and vice versa. Therefore, the maximum allowable PoF will be higher in the case of small volumes and lower otherwise.
- On the other hand, if a bench located immediately above a ramp suffers instability, its potential effect will be greater than that of a similar instability in a bench not adjacent to a ramp.

Table 13-6: Acceptability criteria for the slopes of the Hualilan block.

SLOPES	STATIC FoS	STATIC PoF	FoS PSEUDO MAX	PoF PSEUDO MAX
Global	≥ 1.25	≤ 18%	≥ 1.05	≤ 40%
Interrupt	≥ 1.20	≤ 20%	≥ 1.02	≤ 45%

Finally, it must be noted that a final wall bench should have a longer operational life than a non-permanent wall bench, such as in the case of an expansion.

- The occurrence of instabilities in the inter-ramp slope has an effect that can become very important and, if possible, the occurrence of this type of instabilities should be avoided. However, the DAC must, also in this case, be defined according to the potential for damage associated with instability; which basically depends on two factors: the magnitude of the ramp loss and the volume affected by the instability, as illustrated in Figure 13-3.
- In addition, the smaller the volume affected by the instability and the smaller the possible ramp loss, the less the relevance of the instability, and vice versa. Therefore, the maximum allowable failure probability will be higher and the minimum allowable safety factor will be lower for small ramp volumes and losses.
- However, the greater the volume affected by the instability, the greater its relevance, and vice versa. Therefore, the acceptability criteria should be stricter in those cases where the volume potentially affected is more important.
- No important infrastructure is proposed to be located in proximity to the slopes, since if this were the case, the acceptability criteria for the slopes adjacent to said infrastructure would have to be higher. Therefore, if in the future it is decided to install infrastructure inside the Hualilan pit, the stability of the slopes adjacent to said infrastructure must be reassessed in accordance with DAC that considers the possible consequences of instability affecting said infrastructure.
- Slope design recommendations will be strictly adhered to, regarding:
    - If necessary, implementation of drainage measures.
    - Execution of controlled blasting to minimize the damage induced in the rock mass.
    - Control of compliance with the program line in each bench.

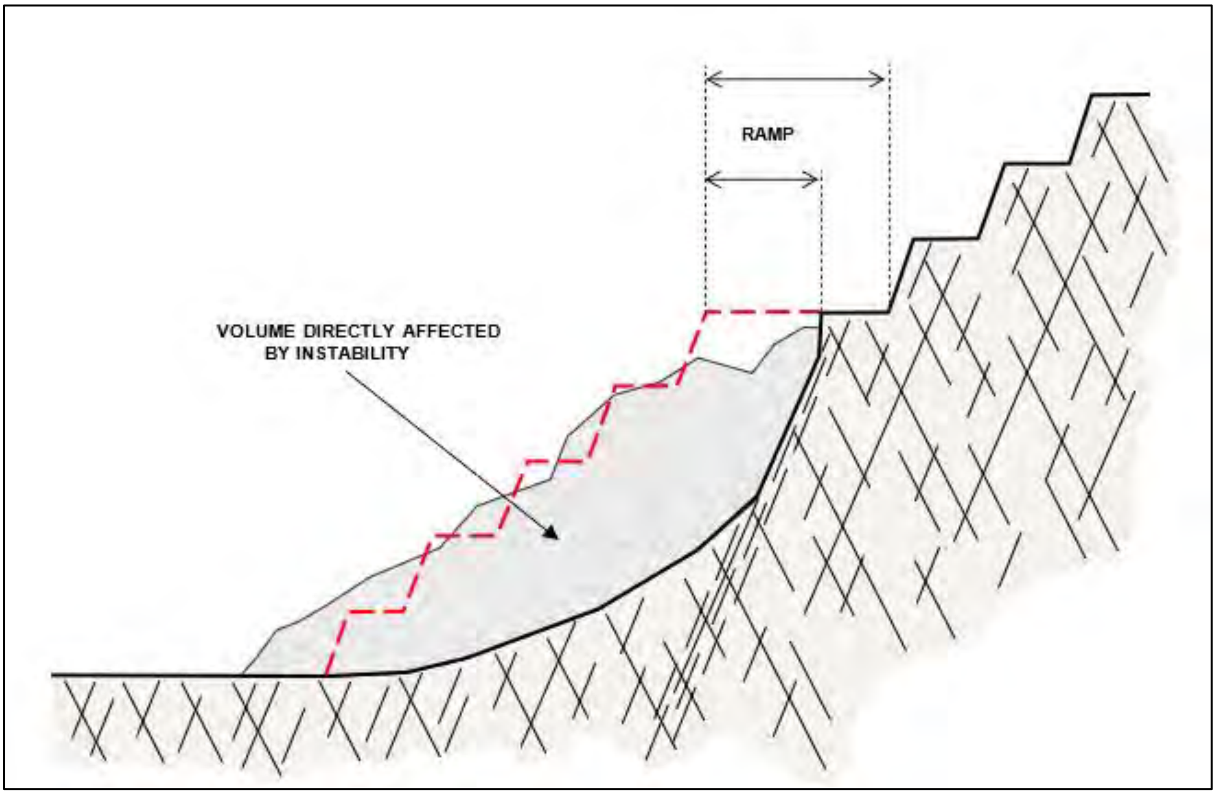


Figure 13-3: Diagram of instability in an inter-ramp slope, showing the volume affected and the loss of ramp.

The geotechnical evaluation of the slope designs that are of interest here is based on the application of acceptability criteria defined in previous studies in projects with similar characteristics and that are within the international standards defined by Table 1 in the Guidelines for Open Pit Slope Design (Read & Stacey, 2009), and the interpretation of these corresponds to stating that: if the evaluated designs meet the respective criteria, the designs are acceptable; Otherwise, its design must be modified.

13.7 Pit Slope Parameters

The results of the stability analysis for the slopes of the original design for the project are summarised in Table 13-7.

- As indicated in the previous paragraphs and the results obtained in the analyses, not all sections met the acceptability criteria. For this reason, it was necessary to set the ramp width and inter-ramp height and modify the global slope angle to comply with the acceptability criteria.
- For sections XS01N, XS06E, XS07S and XS08E, it was necessary to incorporate inter-ramps (IRA) to uncouple the slope and with these the inter-ramp heights and ramp width are defined. Consequently, slopes of more than 250 m in height could be uncoupled.
- With the values obtained from the other sections, even when the heights are higher than the inter-ramps defined in the previous point, it is necessary for them to be

operationally feasible to construct them at these heights. With this, it is possible to identify the other inter-ramp angles (IRA) since these comfortably pass the values of the DAC for the heights of these sections.

- Based on these analyses (global and inter-ramp), it is determined that the stability control is given by the rock mass. This consideration and/or conclusion must be ratified in the next engineering stage to ascertain if the slopes are rock mass controlled, partially structural control, in combination with rock mass or totally structural.
- The heights of the basic parameters of the slopes are given by:
    - Inter-ramp height, 150 m.
    - Ramp width, 22 m.
    - Inter-ramp angle (IRA), depends on the sector according to Figure 13-4.
    - Global Angle (Overall), depends on the sector according to Figure 13-4.

Table 13-7: Results of The Stability Analysis of The Hualilan Project

SECTION	COMBINATION	α o (°)	α g (°)	H (m)	STATIC COND.		PSEUDOESTATIC COND. ,SMAX	
					FoS	PoF (%)	FOS	PoS (%)
XSO1N	Global Original	51	51	277	1.18	20	1.02	46
	Overall Recommended	48	51	277	1.26	13	1.09	33
	LOWER Recommended		51	149	1.57	2	1.38	7
	UPPER Recommended		51	128	1.44	5	1.25	14
XS02E	Global	43	43	293	1.27	13	1.07	36
XS02W	Global	34	34	288	1.66	2	1.38	7
XS03SW	Global	49	49	379	1.31	10	1.12	28
XS04E	Global	54	54	116	1.45	5	1.28	12
XS04W	Global	44	44	154	1.88	1	1.62	2
XS05E	Global	51	51	263	1.36	8	1.17	22
XS05W	Global	44	44	233	1.62	2	1.39	6
XS06E	Global Original	57	57	431	0.98	54	0.85	83
	Global Recommended	48	51	431	1.25	14	1.05	40
	LOWER Recommended		51	133	1.73	1	1.53	3
	INTERMEDIATE Reco-mended		51	151	1.71	1	1.51	3
	UPPER Recommended		51	148	1.54	3	1.34	9
XS06W	Global		48	289	1.46	4	1.26	13
	Global Original		48	301	1.17	22	1.01	48
	Global Recommended		48	301	1.26	13	1.09	33

SECTION	COMBINATION	α o (°)	α g (°)	H (m)	STATIC COND.		PSEUDOESTATIC COND. ,SMAX	
					FoS	PoF (%)	FOS	PoS (%)
XS07S	LOWER Recommended	48	51	150	1.64	2	1.45	5
	UPPER Recommended		51	151	1.52	3	1.34	9
XS08E	Global Original	56	56	366	0.92	68	0.80	91
	Global Recommended	48	48	366	1.28	12	1.09	33
	INTERMEDIATE Recommended		51	152	1.55	3	1.35	8
	UPPER Recommended		51	149	1.47	4	1.27	13
XS08W	Global	46	46	203	1.60	2	1.39	6

NOTE: Values in red do not meet the acceptability criteria.

α_O: OSA

α_g: Inter-ramp Angle

H: Rupture surface height.

FoS: Factor of Safety

PoF Probability of Failure

The results obtained from the analysis of the sections indicate that they are stable in most cases and meet the DAC set forth for the slopes in the Hualilan Pit.

The original sections that did not meet the DAC were modified to achieve this objective and with this, it is possible to obtain the maximum inter-ramp heights, inter-ramp angles, global angles, and ramp widths. This condition is independent if the benches are from 5 to 14 m high.

Figure 13-4 shows the zoning by IRA and Global (Overall) angles.

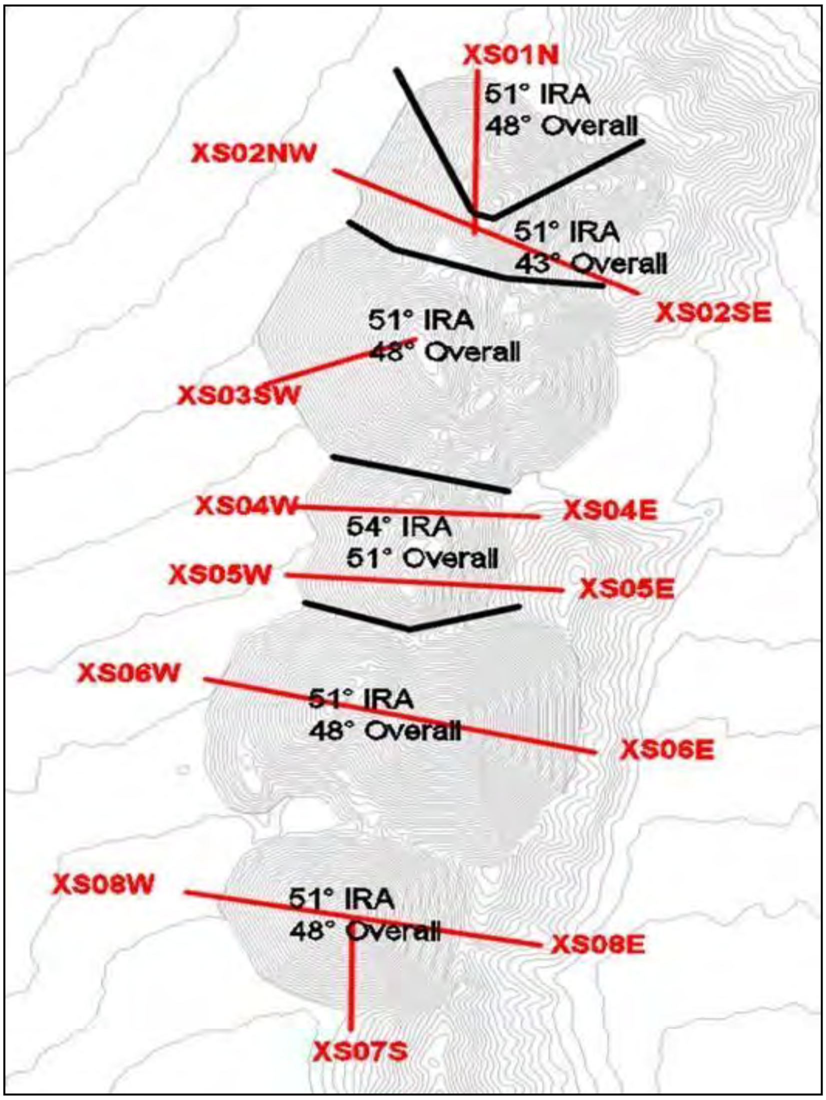


Figure 13-4: Plan showing zoning by IRA and Global angles (Overall), for the design of the Final Pit of the Hualilan block.

Further, it should be noted that in this study the stability of the slopes in seismic conditions was evaluated, although this is not a critical factor for the rocky slopes of an open pit mine.

There are no known cases in which an earthquake has been the cause of a major failure of a slope in an open pit mine[1].

The main reason for this is the fact that typical rock masses, from an open pit mine, do not suffer a reduction in their resistance due to seismic stress, rather this happens in loose granular soils.

[1]In some cases where the condition of the slopes was potentially unstable, in a static condition, the occurrence of an earthquake aggravated the problem, but the earthquake itself was not the cause of the instability but only an event that "triggered" or anticipated the occurrence of the same

14 HYDROGEOLOGY AND HYDROLOGY

14.1 Hydrogeology

14.1.1 Regional Hydrogeology

In the territory of the Province of San Juan, three hydrogeological regions are recognized according to the classification proposed by Auge, et al, 2006. See Figure 14-1.

*14.1.1.1 **Sierras Pampeanas and its valleys***

This region occupies around 230,000 km^2 in the central-northern part of Argentina. The Cenozoic tectonics generated intermontane depressions between raised blocks of the Proterozoic - Lower Paleozoic basement, oriented north - south, filled by Tertiary and Quaternary sediments that constitute groundwater basins of varying magnitude. This region develops in the eastern end of the Province of San Juan and includes the Valle Fértil – Mascasín groundwater basin.

*14.1.1.2 **Piedmont and Cuyana Plain***

This region occupies the NNW sector of Argentina, covering some 86,000 km^2. The mountainous reliefs of a general North-South course are made up of rocks of different ages (Paleozoic to the present) and lithologies, defined basins filled by sediments of different origin, varied granulometry and thickness. These groundwater basins are fed by fluvial channels of the pluvo-nival regime. In the territory of the Province of San Juan it includes the basins of the valley of Tulum, Valdivia - Ramblón and Pedernal - Acequión and Bachongo.

*14.1.1.3 **Precordillera, Frontal Cordillera, Principal Cordillera and its valleys***

This region located in the NNW of Argentina covers an area of 129,000 km^2 and contains the Hualilan Project Area. The main valleys to the west are those of Iglesia - Calingasta - Barreal - Uspallata and to the east in pre-Andean depressions Jáchal, Gualilán, Matagusanos, Ullum - Zonda, etc. These are elongated basins running north-south, filled by coalescing alluvial cones and recharged by rivers and streams with a pluvo-snow regime. This region includes the groundwater basins of Ullum - Zonda, Calingasta - Barreal, Iglesia, Jáchal, Gualilán, Matagusanos, and others.

The most conspicuous valleys are those of Rodeo - Iglesia, Calingasta - Barreal, Jáchal, Ullúm - Zonda and the depressions of Gualilán and Matagusanos. These valleys of tectonic origin usually present important alluvial and colluvial fillings from the Cenozoic, with the capacity for the storage and production of high volumes of groundwater.

Given that it is a region with little rainfall, the recharge comes from infiltration during the flood season coinciding with the snowmelt. The most exploited sites for irrigation are the valleys of Tulum, Ullum - Zonda and Jáchal.

In the Precordillera, within the framework of this hydrogeological region, the mountain ranges are made up of Paleozoic rocks with a predominance of clastic and calcareous sequences, along with igneous rocks.

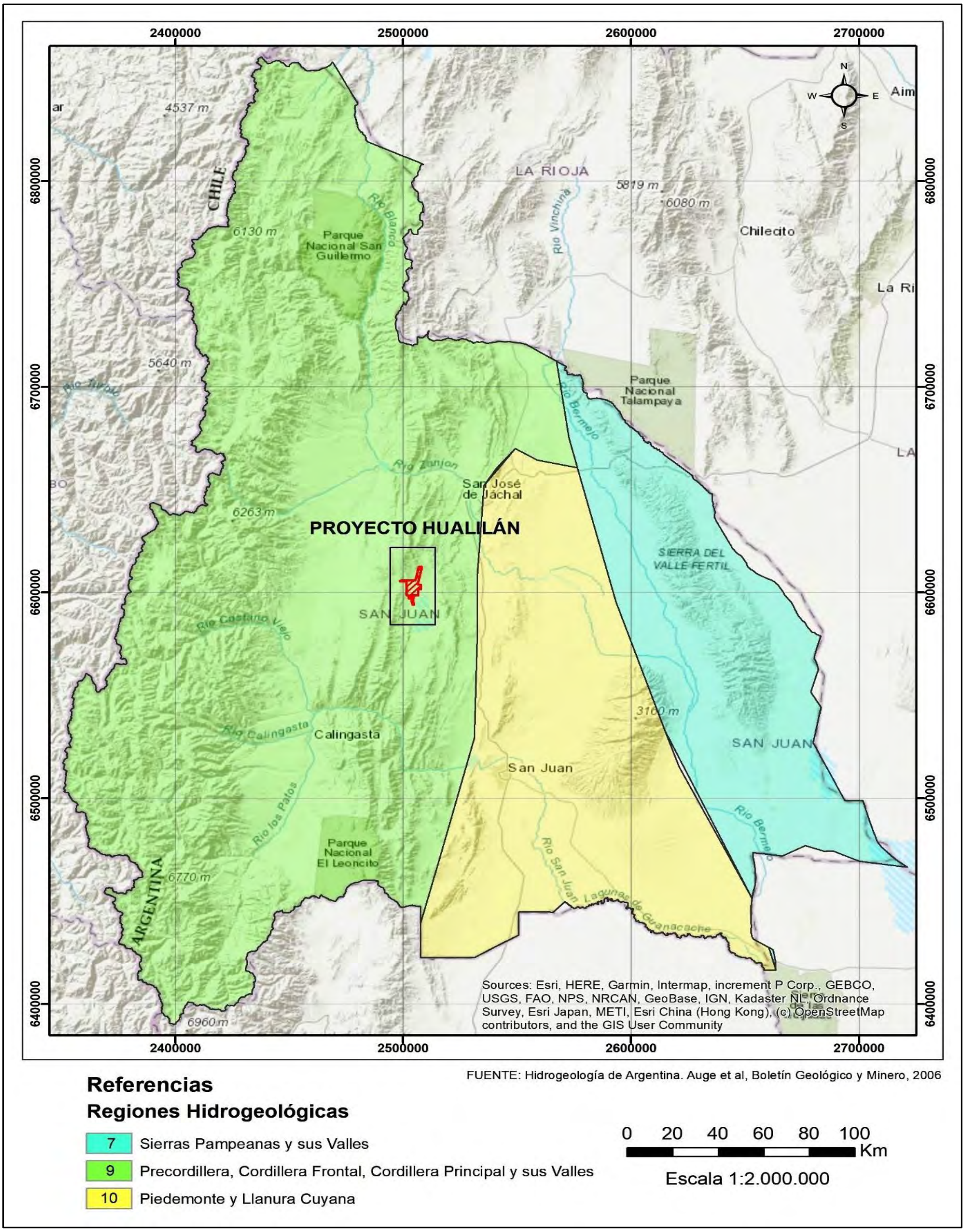


Figure 14-1: Hydrogeological Areas – San Juan Province

14.1.1.4 ***Gualilán Basin***

The project area is located in the Gualilán Basin. This is an intermontane tectonic depression of about 300 km^2 of extension, of tectonic origin and endorheic character, located between the La Invernada and Talacasto mountain ranges. It houses a groundwater basin with a surface area of 200 km^2 ((Furque, 1983), (INA / CRAS, 1998), (Damiani, 2016) and (INA / CRAS, 2021)).

In the area of the basin there are no populated cities nor agricultural-livestock enterprises of commercial value. However, in the surroundings of Cienaguita and Estancia La Ciénaga, in the southwest of the basin, there are a series of small farms cultivated with pastures for local consumption, which take advantage of the water provided by the existing springs in this sector of the Basin. At Estancia La Ciénaga, unsuccessful attempts have been made to plant forest trees (poplars), with the poplar trees usually affected by frost in the spring season.

The depositional environment corresponds to coalescent alluvial cones, with a foothill descent and silt-clay alluvial plain in its central-south zone. The drainage network that attends this intermontane basin is made up of innumerable temporary channels, inland due to rainfall, which usually presents a torrential nature and tends to concentrate in the summer season. These spill fields transport a significant volume of detrital sediments (solid waste) from the surrounding mountains.

This dynamic has originated a depositional environment of coalescent alluvial cones, with their respective lowering zone and central depression of silt-clay size, configuration typical of arid zone pockets. The evolution of this type of depocenters has a tendency, as clogging occurs and through a process of aggradation, that the portion occupied by fine sediments tends to increase, extending towards the edges.

In this geological framework, there would be possibilities of finding in depth, coarse-grained materials and aquifer behaviour in areas currently covered by impermeable silt-clay sediments. (INA / CRAS, 2021).

The regional geological information along with the geomorphological and tectonic features that characterise this and other intermontane depressions in the region allow us to assume the existence of a hydrogeological basement made up of two main lithological units below the Quaternary alluvial fill. In the area of the hydrogeological basin, the basement would be made up of two units, namely resistive basement and conductive basement.

The resistive basement is represented by calcareous and clastic sequences from the Lower to Middle Palaeozoic (limestone, shales, sandstone, etc.) and even an intrusive, of dacitic composition, assigned to the Miocene. Compaction and diagenetic changes, by eliminating or minimising the primary porosity of the sediments, determine the high resistivity of these lithological units.

The conductive basement is represented by the Neogene sediments that house mineralised waters and whose intercalations, which are rich in evaporites such as gypsum, give it high electrical conductivity.

The granulometry of the sediments participating in the alluvial filling of this basin decreases from the edges towards the centre. Superficially, there is a central zone of silt and clay, partially salinised, and another peripheral zone with coarse-grained materials (block, gravel, pebbles) resulting from the physical weathering of the various lithological types of the Palaeozoic and Neogene.

The subsoil geological information obtained from lithological profiles and well logging located in the southwestern sector of the basin, in the vicinity of National Route 149, indicate that the thickness of the alluvial fill is at least 140 m. It is composed mainly of silt and clay in the upper part. The lower part is gravel as well as pebbles and sand with intercalations of fine materials. This granulometric distribution favours the confinement of aquifers. The pumped flows vary between 80 to 200 m^3/h and the ascending static levels reach values close to 4-5 m deep.

The water is calcium-bicarbonated and sulfated-sodic. The electrical conductivity varies between 565 and 788 microsiemens/ cm, which allows it to be included in Class II, moderate saline hazard.

In order to identify and quantify the source of water supply for the mining project, a series of studies and investigations have been carried out in the Northern Third of the Pampa de Gualilán, which receives occasional contributions from the Agua del Molle River, from the NW, and from the Quebrada de La Cienaguita and the Cajón River to the N, also accounting for the occasional contributions from the Agua del Médano River from the E respectively.

The Northern Third of the Pampa de Gualilán is located at a higher structural level, as a consequence of a system of faults with a NW - SE strike recognisable from satellite images (INA / CRAS, 2021).

14.1.2 Water Supply Exploration

In the project area, and with the aim of defining the subsoil geology and the geometry of the aquifer, two geophysical prospecting campaigns were undertaken by carrying out Vertical Electrical Soundings (VES).

The first campaign (May 2021) was undertaken by INA/CRAS and included five (5) VESs and the second campaign (September 2022) was undertaken by Eduardo Morell (BSc in Geophysics) and included five (5) VES. Figure 14-2 shows the location of the VESs.

Based on the geophysical, surface geology and subsoil hydrogeological information, four Hydrogeological Units are recognised, with completely different behaviour in relation to groundwater.

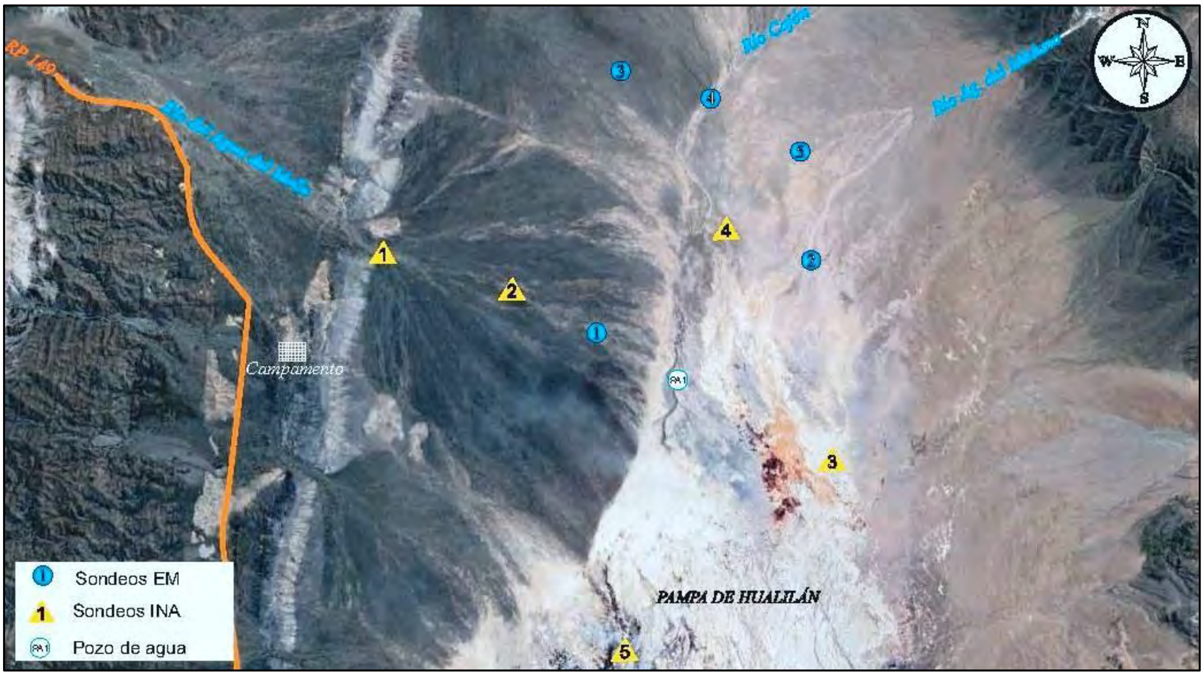


Figure 14-2: Location of Vertical Electric Surveys

The concept of a Hydrogeological Unit (HU) involves a subsoil unit identified by a characteristic range of electrical resistivity and the presence of consistent hydrogeological characteristics, associated with one or more geological formations of variable thickness and possible stratigraphic correlation.

UH1: correlates with the Upper Tertiary sediments with a predominance of pelitic rocks, consolidated sandstones, few levels of conglomerates and partly carriers of gypsum and other soluble salts, in correlation with the El Corral Formation or Mogna Formation. In hydrogeological terms, it has a Non-Aquifer behaviour and constitutes the conductive hydrogeological basement of the groundwater basin. It is characterised by resistivity values between 321 Ω.m. and 96 Ω.m.

UH1b: illuminated in well Nº 1 is clastic sediment of the Middle Paleozoic (Silurian - Devonian), behaving as Non-Aquiferous and constituting the resistive basement of certain sectors of the hydrogeological basin. The resistivities of this unit reach values greater than 800 Ω.m. up to 2,000 Ω.m.

UH2: it is found above UH1 and is correlates with medium-sized, permeable, Quaternary sediments, whose granulometry includes fine to medium gravel, boulders, and sand. This unit corresponds to the saturated sedimentary fill and represents the Free Aquifer. It reaches thicknesses of 80 m in VES2 and 160 m in VES3. The resistivity values vary between 18 Ω.m. to 22 Ω.m., except in the case of the VES4 where it climbs up to 164 Ω.m.

UH3: rests on UH2 with its upper limit being the water table. It is made up of medium-sized, permeable, Quaternary-age sediments, whose granulometry corresponds to fine to medium-sized gravel, boulders, and sand. This unit corresponds to the saturated Quaternary sedimentary fill and also represents the Free Aquifer. It reaches thicknesses between 50-60 m in the 1 - 1' profile (NW - SE course) and between 30 - 40 m in the 2 - 2' profile (N - S course). The resistivity values vary between 31 Ω.m. and 640 Ω.m, for which it is interpreted that this unit is saturated with fresher water (less saline).

UH4: this unit is located between the water table and the surface of the land. It is made up of Quaternary sediments of variable granulometry (from blocks to silt and clay). From the hydrogeological point of view, it corresponds to the Vadose or Unsaturated Zone. It presents a thickness that varies between 20 to 40 m and the resistivities oscillate between 15 Ω.m. and 669 Ω.m.

*14.1.2.1 **March 2021 Campaign***

From the data recorded in the 5 VESs, two profiles 1 - 1' and 2 - 2' were made.

Profile 1 - 1', approximately 5 km long, extending NW - SE, is VES 1, 2 and 3. In this section, the top of UH1 (conductive hydrogeological basement) would be found at a depth between 72 m in VES1 and 226 m below VES3. The static level along the cut is between 40 m and 15 m deep, becoming shallower towards the bottom of the Pampa de Gualilán. The aquifer would behave as free (subjected to atmospheric pressure) and is approximately 120 m thick in VES2 and 200 m thick in VES3.

The 2 – 2' profile, approximately 4 km long and N – S oriented, is made up of VES4 and 5. In this cut, the top of UH1 (Tertiary age hydrogeological basement) is found at a depth ranging between 153 m at the height of VES4 and 118 m at the height of VES5. The static groundwater level, inferred from geophysics, is 33 m deep at the height of VES4 and 16 m below VES5. The aquifer is free and is approximately 110 m thick saturated in VES4 and about 100 m saturated in VES5.

Based on the data of the profiles and the geological interpretation of the subsoil, INA/CRAS concluded that the Quaternary sediments constitute the most important aquifers in view of their high effective porosity and high permeability, recommending the execution of a borehole in the sector between VES2 and VES3 to the E of the fault where the greatest saturated thicknesses are recorded and more favourable conditions are verified in terms of surface recharge.

The results obtained from these geophysical prospecting campaigns are presented in Figure 14-3.

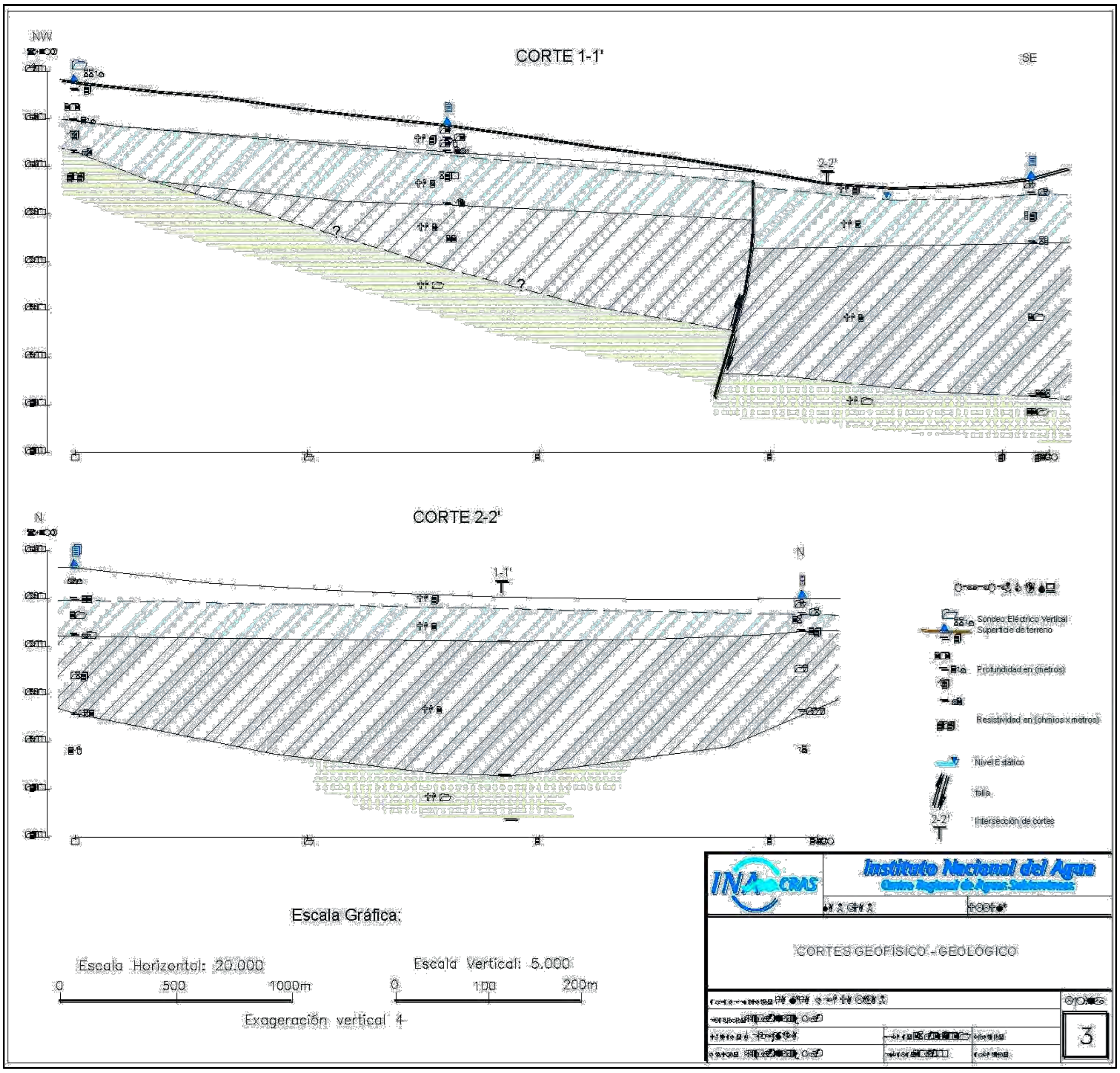


Figure 14-3: Geophysical Cross Section showing Hydrological Units

14.1.2.2 ***September 2022 Campaign***

The data recorded in the 5 VESs collected in late 2022 are detailed below.

HLLN 01 indicates the presence of the water level approximately towards 55 m. The aquiferous horizon, represented by three layers, shows average resistivity values of 77 Ω.m. and extends to approximately 130 m, depth from which the resistivity drops sharply to the order of 21 Ω.m. probably indicating the change to the UH1 and from 288 m would underlie the UH1b with resistivity of the order of 2,000 Ω.m.

HLLN 02 the aquifer horizon is represented by the third and fourth layers and would begin at approximately 44 m. It gives average resistivity values of 44 and 65 Ω.m. and it would extend to at least 250 m, the depth of investigation reached with the final openings of the device.

HLLN 03, although it gave unreliable results, since due to limitations in the field, the measurement had to be made under the high voltage line, which causes electrical interference (noise) in the readings, however, it was interpreted with an error greater than 30% the presence of the aquifer horizon towards 39 m and is represented by the third layer with average resistivity values of 87 Ω.m. It extends to approximately 86 m depth from which the resistivity drops to the order of 25 Ω.m. (UH1) and from 166 m the roof of UH1b would be found.

HLLN 04 indicates the presence of the water level at approximately 35 m. The aquiferous horizon, represented by layers 2 and 3, shows average resistivity values of 57 and 99 Ω.m. and extends to approximately 250 m, the device's maximum investigation depth, without reaching the basement.

HLLN 05 is not precisely defined at the water level. The aquiferous horizon is represented from the second layer. This horizon shows mean resistivity values of 82 Ω.m., the third layer represents UH2 with mean resistivity values of 49 Ω.m. and from 177 m the so-called UH3 would be found with average resistivity values of 73 Ω.m. The saturated Quaternary fill would also extend to at least 250 m.

Based on the VES data, Geophysics graduate Eduardo Morell concluded that there is a primary aquifer level made up of medium to coarse-sized sediments, with a varied proportion of fine-grained materials. In these sediments there is evidence of a tendency to increase the proportion of silts as it advances towards the South (in the area of the Northern Third of the Pampa de Gualilán).

In VES2, VES4 and VES5 it is very probable that the aquifer thickness reaches about 250 m depth and the hydrogeological basement was not detected. These sediments lie on a semiconductor horizon that acts as the base of the basin and that could be considered correlated with the tertiary lithostratigraphic units (El Corral Formation or Mogna Formation).

Based on the geological interpretation of the subsoil, the execution of a drill test located 1,000 m to the West of VES Nº 1 was recommended, where it is feasible to intercept saturated thicknesses close to 200 m and resistivities of the order of 40 to 100 Ω.m.

14.1.2.3 ***Water Exploration Drilling***

The well called PA-1 was located in the north-central sector of the Pampa de Gualilán (Department of Ullum, Province of San Juan) at the intersection of the geographical coordinates 68° 54' 33.42” LS; 30° 44' 20.38" LO at a height of 1,685 meters ASL.

Drilling by the Rotary method began once the work of preparing the land, location and leveling of the drilling machine, construction of water sumps and injection chute and preparation of the mud were completed.

The drilling machine used was a Drilltech DK25 rotary rig with a drilling capacity of 200 m, owned by the Company Nivel Construcciones ("Nivel"). Nivel was contracted to execute the work under the technical supervision of Lic[2]. Carlos Torres (MP Nº J-032) and Rubén Gianni (MP Nº J-066) from TEOTOP Geoelectric and Hydrogeology (Teotop, 2023).

The water to meet the requirements during the drilling work was brought from the springs of Estancia La Ciénaga by means of a tank truck. Bentonite was used to prepare the drilling mud that is used to stabilise the walls of the well, raise the cutting towards the surface, as a lubricant and cooling of the drilling tools, etc.

The drilling began in June 2022, using a triple diameter bit (12" - 25" - 27") up to a depth of 11 m. Subsequently, tubing was carried out with 11 m of 22" guide pipe, in order to prevent possible landslides. Once the previous operation was finished, casing was cemented and left 48 hours to set. Subsequently, the 8" pilot well continued up to a depth of 150 m.

During the drilling samples were extracted every 2 metres for logging as shown in Figure 14-4.

Figure 14-4: Samples from drillhole PA-1

14.1.2.4 ***Hydrogeological record of drillhole***

Between the soil surface and a depth of 40 m, the hole encountered fine and medium sand with few mid-sized sand clasts and fine gravel (gravel), with variable participation of silt. From 40 m to 48 m the materials increased their granulometry, corresponding to medium and

[2] *In South America the term Lic is used to denote university degree qualified.*

coarse sand with less mid-sized sands. The sediments intersected increase in grain size towards the base of this section with the base of this zone comprising quartz, sandstone and lesser limestone/calcite.

Between 48 m and 66 m, the well encountered fine, medium and coarse gravel, coarse sands and fine sands, and less of silt – clay materials. The clasts appear rounded - subrounded up to angular or splintery fragments of up to 1.5 cm as a result of the rupture of larger clasts. The lithologies correspond to quartz, sandstone, limestone/calcite, shales, igneous rocks and indeterminable lithics.

From 66 m to 127 m, the sediments comprise fine, medium and coarse gravel, coarse and fine sands with variable amounts of silt material; possibly less abundant between 90 m and 95 m and 110 m and 118 m. The clasts appear rounded - subrounded, up to angular or splintery fragments of up to 1.5 cm.

At a depth of 127 m, the pre-Quaternary basement of the groundwater basin was reached, which consists of limestone. This lithological type was maintained up to 150 m; final drilling depth. Following drilling electrical profiling of the well was undertaken by EM contractors, the operator being the Graduate in Geophysics Eduardo Morell and witnessed by TEOTOP. The profiled depth was 149.5 m. This log is shown in Figure 14-5.

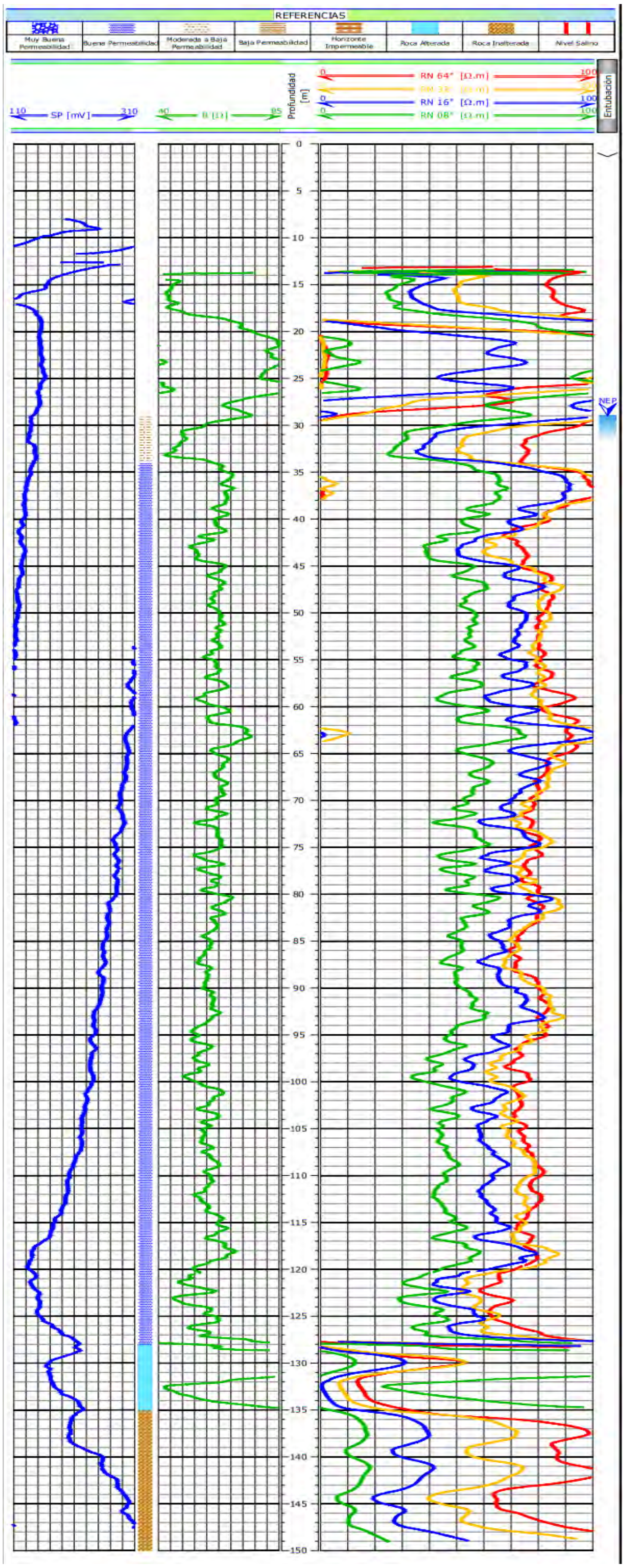


Figure 14-5: Log for Hole PA-1

With the help of the samples of the extracted materials, the profiling was analysed and the most appropriate piping design was decided, fixing the position of the blind pipes and filters. The well was cased using steel pipes and 12” galvanized steel continuous slot filters and prepared with rings for welding union between the different sections. It was piped from the ground surface to a depth of 126 m, with the following characteristics as shown in Figure 14-6:

- From the surface to a depth of 55 m, blind iron pipes 12” in diameter
- From 55 m to 65 m deep, galvanized iron filter pipes, 12” in diameter and 2 mm opening (10 m of filters)
- From 65 m to 73 m 12” blind iron pipe
- From 73 m to 83 m deep, galvanized iron filter pipes, 12” in diameter and 1.5 mm opening (10 m of filters)
- From 83 m to 90 m 12” blind iron pipe
- From 90 m to 100 m deep, galvanized iron filter pipes, 12” in diameter and 1.5 mm opening (10 m of filters)
- From 100 m to 105 m 12” blind iron pipe
- From 105 m to 120 m deep, galvanized iron filter pipes, 12” in diameter and 2 mm opening (15 m of filters)
- From 120 m to 126 m 12" blind iron pipe with 0.40 m "pencil tip".

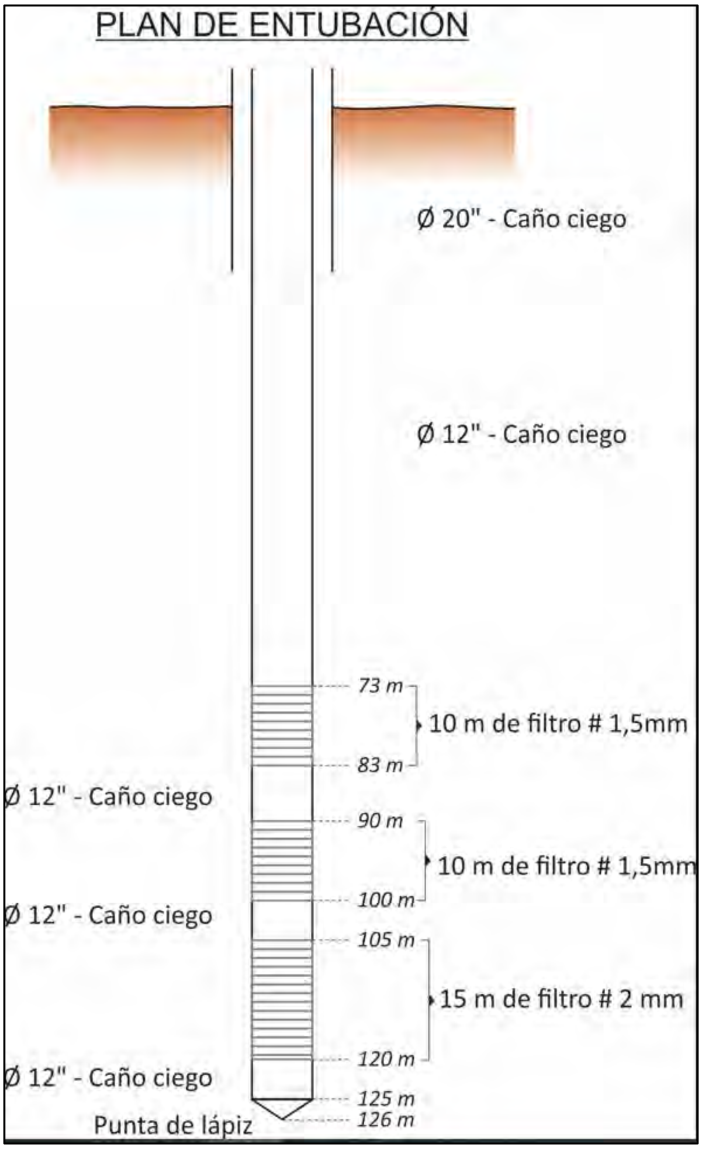


Figure 14-6: Location of filters Hole PA-1

A total of 45 m of filters were placed and the upper end of the casing was left about 0.60 m long above the wellhead, so that it protrudes above the level of the natural terrain and allows the following works to be carried out: cleaning, development and execution of pumping tests.

The annular space between the well walls and the casing was filled with gravel (gravel prefilter) from the bottom of the well to the surface, thus constituting a prefilter to contain fine materials and improve permeability in the vicinity of the filter sections. The gravel was introduced by gravity using the counter current circulation method and 12 gravel bags were used.

For the purposes of cleaning and developing the well, after etching, sodium tripolyphosphate was added in an aqueous solution, in order to disperse the clay particles naturally incorporated or added with injection (bentonite) and facilitate drilling mud removal. By breaking their agglutination, they allow their removal from the filter, the pre-filter (engraved) and formation material. The solution was left acting for four days.

Subsequently, the well was washed by adding clean water in order to dislodge as much mud as possible. Next, the piston and jet cleaning was performed for approximately 72 hours. This activity was carried out through a single manoeuvring since both were in solidarity, forming a single tool that was introduced into the well, cleaning alternately, using the piston in the blind pipe section and the jet in the filter area. The outlet flow at the jet's peaks was calibrated at a pressure of 9 bars in order to achieve a water speed of 37 m/s. All the works above are necessary to extract the greatest possible amount of sludge or bentonite injection, in order to leave the filters clean.

Finally, as the last action for the cleaning and development of the well, a 50 HP electric pump, 6" in diameter and a 6" elevation pipe was installed. This was located at a depth of 87 m, within the section of blind pipe located between 83 and 90 m.

The work was carried out with different flows, so that for a given flow the muddy or turbid water at the beginning, with benthic mud and fine materials, becomes clearer until it becomes transparent and without sedimentation of fines. The approximate development time was about 48 hours.

Pumping was then stopped and the groundwater level was allowed to recover for pumping tests to determine the specific flows or yields of the well for different exploitation flows, the loss coefficients, the efficiency of the well and the transmissibility of the aquifer.

*14.1.2.5 **Hole pumping tests***

In order to preliminarily evaluate the properties of the aquifer and the degree of completion of the well, a staggered pumping test of three steps of increasing flow rates was carried out with 60-minute intervals using a 6-inch electric pump, driven by a diesel engine, with a maximum pumping capacity of 74 m^3/h. Following the test a series of calculations were undertaken to determine the well efficiency, yield, permeability and descent.

14.1.2.5.1 Results of the staggered pumping test

In Table 14-1, the data of Relegations versus Time is shown for the three steps of pumping carried out and Figure 14-7 shows the curve of descents versus the time of the three pumping steps used.

Table 14-1: Staggered Pump Test Relegation vs Time Results

Time (min)	Level Drop (m)	Time (min)	Level Drop (m)	Time (min)	Level Drop (m)
2	37.43	62	38.91	123	39.8
3	38.32	64	38.89	126	39.84

Time (min)	Level Drop (m)	Time (min)	Level Drop (m)	Time (min)	Level Drop (m)
6	38.39	68	38.93	130	39.84
8	38.38	70	38.92	135	39.855
10	38.36	80	38.92	140	39.855
15	38.34	90	38.915	150	39.88
20	38.4	100	38.935	160	39.9
30	38.425	110	38.92	170	39.9
40	38.435	120	38.95	180	39.9
50	38.465				
60	38.495				
Q1 = 20 m^3/h		Q2 = 29 m^3/h		Q3 = 74 m^3/h	
Starting Level = 37.74 m					
Re1 = 35 $m^3/h/m$		Re2 = 32 $m^3/h/m$		Re3 = 29 $m^3/h/m$	

The static level indicated in Table 14-1 is measured from the reference point located 1.23 m from the ground surface, in order to facilitate field measurements. The static level with respect to ground level is 36.51 m.

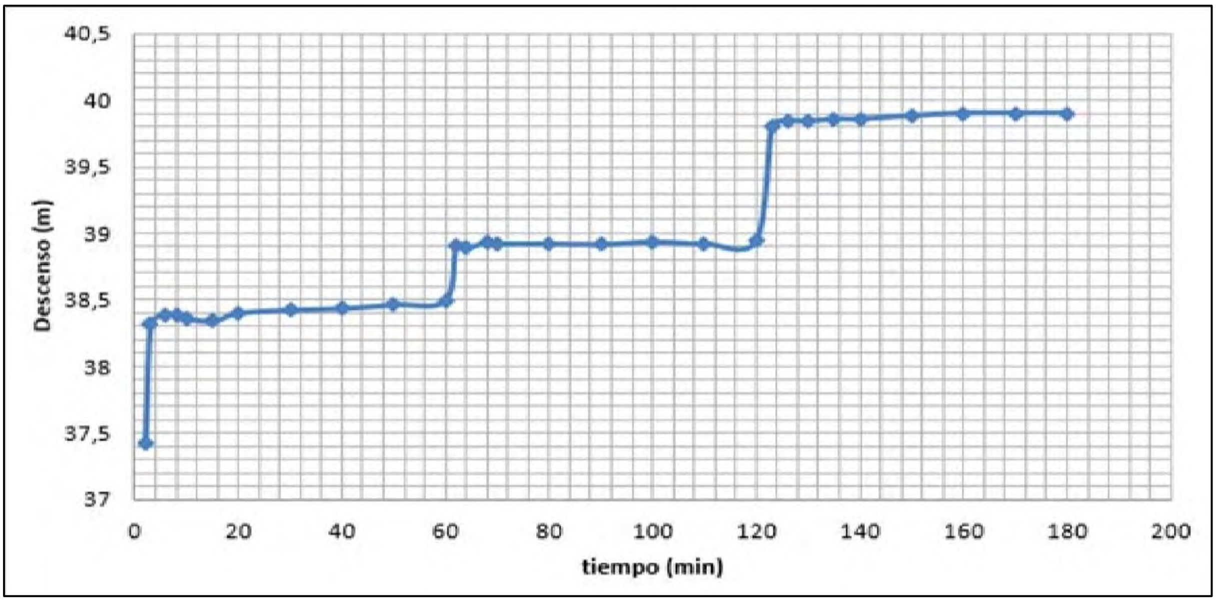


Figure 14-7: Curve of Descents vs. Time of the three pumping steps used

14.1.2.5.2 Determination of the Constructive (C) and Formational (F) Load Coefficients. Hantush Solution – Bierschenk Well Loss

The most adopted formula to express the descent in a well is the one proposed by Jacob (1938):

$$S = F \times Q + CQ^2 \qquad \textbf{(1) Characteristic equation}$$

The term FxQ expresses the decrease due to pressure losses due to the effect of friction that opposes the porous medium to the passage of a viscous fluid and due to the loss of validity of Darcy's Law in the vicinity of the filter due to the increase in speed in its proximity. In this case, the admissible value of the Reynolds Number is exceeded, invalidating Darcy's Law.

The coefficient F is called the Coefficient of Formational Load Losses and is expressed in days/m^2 and is variable over time. The term CQ^2 expresses the decrease caused by the construction characteristics of the well, the diameter of the elevation pipe and the penetration losses of the pump as in the case of electric pumps where the entry of water into it is partially restricted.

The coefficient C expresses the load losses or over-lowering caused by the above causes. Its unit of measurement is $days^2/m^5$.

The F and C coefficients were determined by applying the Aquifer Test V.3.6 software using the Hantush - Bierschenk Well Loss solution, from the staggered test of three increasing flows. The software output is shown in Figure 14-8.

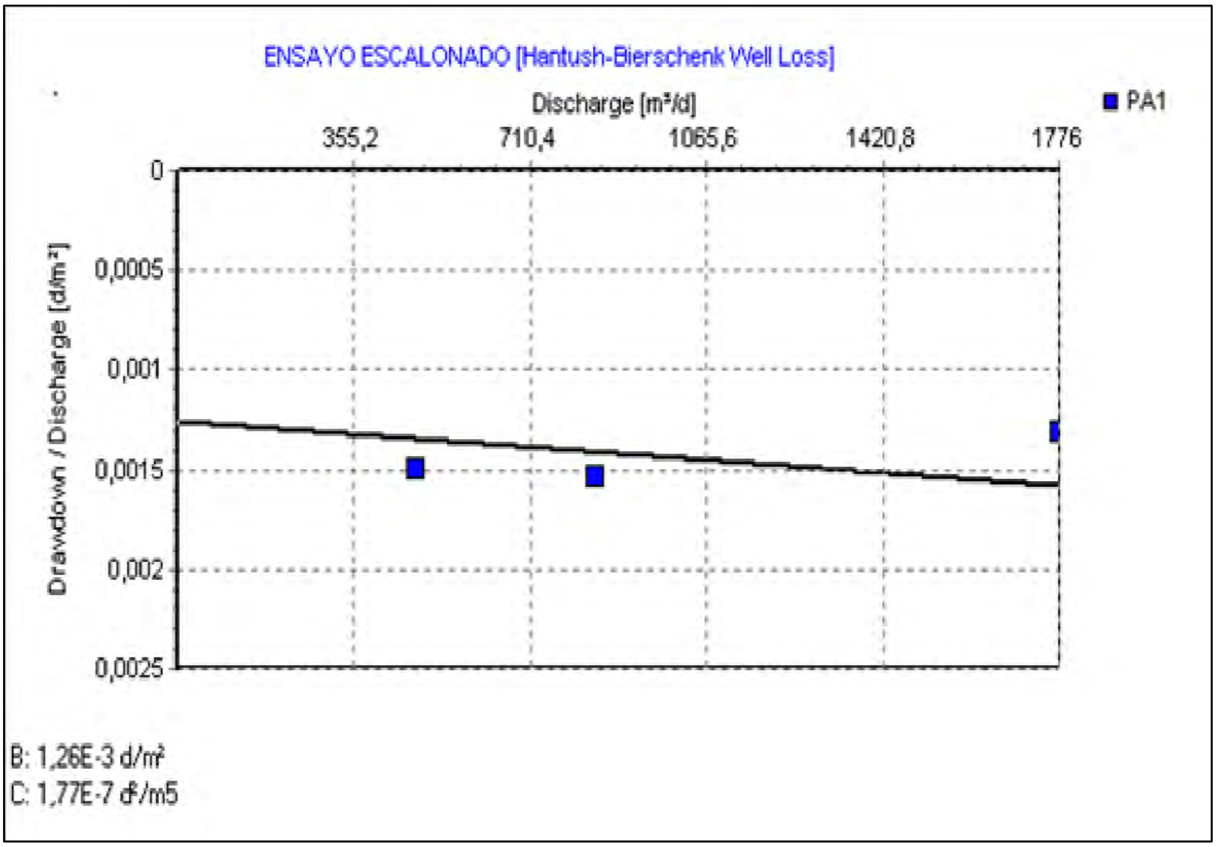


Figure 14-8: Staggered Test (Hantush - Bierschenk Well Loss)

Once these coefficients were determined, the value of the Efficiency of the well (Ef) was calculated for the three flows.

Finally, the Characteristic Equations (1) of the well results:

$$S\,(\text{drop}) = 1.26\,E^{-3}\,x\,Q + 1.77\,E^{-7}\,x\,Q^2$$

(2) The value of F is only valid for comparing wells located in the same geological formation. Regarding the C coefficient, Walton (1958) establishes as a guideline that, in well-constructed wells, if they are well developed, it is generally less than:

$$2.5\ E^{-7}\ \text{days}^2/\text{m}^5$$

if the value reaches:

$$5\ E^{-7}\ \text{days}^2/\text{m}^5$$

indicates the beginning of plugging in wells in use or deficient development time in new wells. The value obtained in the PA-1 well confirms that there was optimal cleaning and development. Equation (2) should not be applied to determine drawdowns.

14.1.2.5.3 Efficiency and Efficiency Curve vs. Flow

The Efficiency (percentage quotient between the theoretical decrease compared to the real decrease) is a valuable data for the construction of the well and allows comparing wells in the same geological formation, provided that equal flow data are taken.

The calculation of Ef starts from the following equation:

$$Ef = \frac{1}{1+\left(\frac{C}{F}\right)}\,x\ Qn\ (Q\ in\ m^3/day)$$

The systematic verification of the Efficiency (e.g. every three years) allows assessment for the early appearance of incrustation phenomena, corrosion, etc. which would indicate the need for preventive maintenance.

The three values of Ef were calculated for each flow rate step and were represented as a function of said flows, obtaining the relational curve Ef vs. Flow rates (Figure 14-9).

In the case of the PA-1 well, despite the fact that the value of C is optimal, everything indicates that the low efficiency for flows greater than 230 m^3/hour (57% for 250 m^3/hour, see Figure 14-9) is possibly related to a lower transmissive capacity of the aquifer or, most likely, to an immature development (low value of the F coefficient) which should improve over time. Normally newly built wells tend to increase their Ef after the first year of activity.

According to the test carried out, the well reaches remarkably high Re values, between 25 and 39 m^3/h/m. Although, during the test, the pump flow rate was reduced (subject to its maximum capacity: 74 m^3/h), the graphical representation of the Re vs. Flow, allows you to fit an exponential regression curve and then extrapolate it to the Re corresponding to a maximum flow of 230 m^3/h.

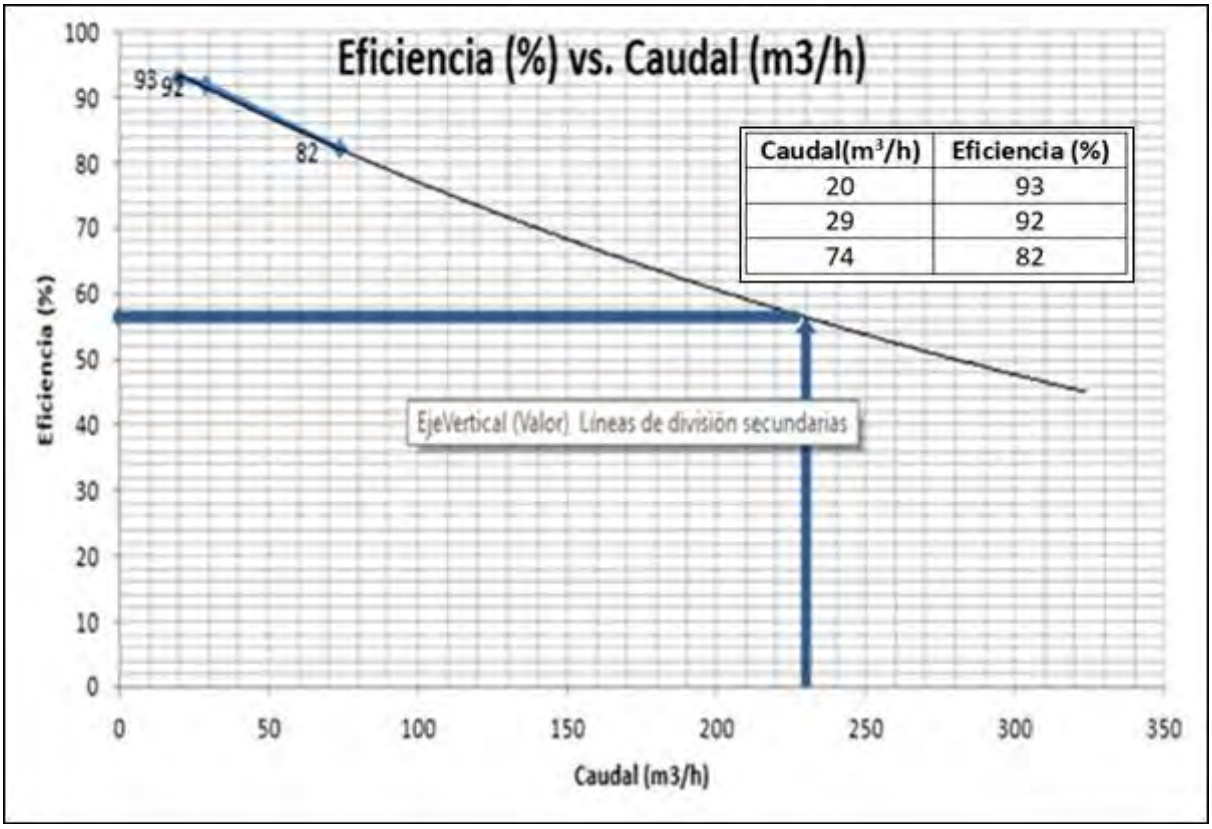


Figure 14-9: Efficiency versus Flow Rates

14.1.2.5.4 Specific Yield Versus Flow

The term specific performance (Re, in $m^3/h/m$) expresses the increase in flow that would be obtained for each additional metre of decrease in the dynamic level; it is not a linear function as evidenced in the relational curve of Figure 14-10, it is an exponential function.

The extrapolation of the regression curve for a flow of 230 m^3/h indicates an estimated Re of 25 $m^3/h/m$. This value allows us to infer that, with said flow, a drop in the static level of 10 m to 11 m would be obtained. At the maximum test flow the water is clear, without sand particles.

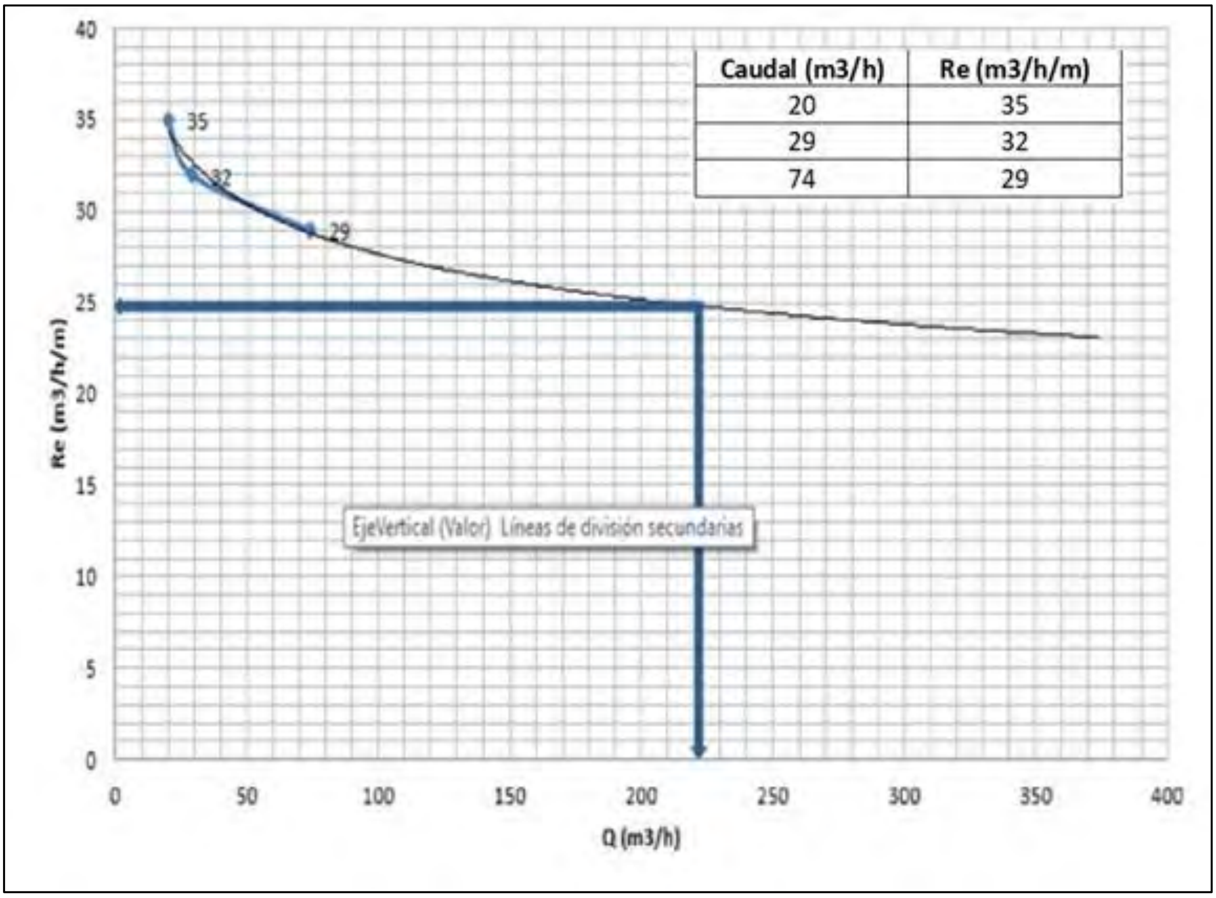


Figure 14-10: Specific Performance (Re) vs Flow (Q)

With a flow of 230 m^3/h, the pump works within the range of acceptable Ef (>57% see Figure 14-9, Efficiency Vs. Flows) and very high Re (25 m^3/h/m, as seen in Figure 14-10).

14.1.2.5.5 Determination of Transmissibility and Permeability

The Aquifer Test program enables the use of the measurements of a staggered test (variable or intermittent flow rates) to "estimate" the value of the Transmissibility (T, m^2/day) and known the total saturated thickness, the determination of the permeability "k" in m/days, The calculation of the T, in this case, is indicative and generally lower than the real one because the drops were measured in the pumping well itself in which it is known that drops greater than the real ones However, it is a fact, although conservative, important to count.

The value of T (see Figure 14-11) found through the STEPTEST (Cooper – Jacob) solution is:

$$T = 864 \text{ m}^2\text{/day}$$

Notes that at the time of carrying out the test, there was no observation well (PO).

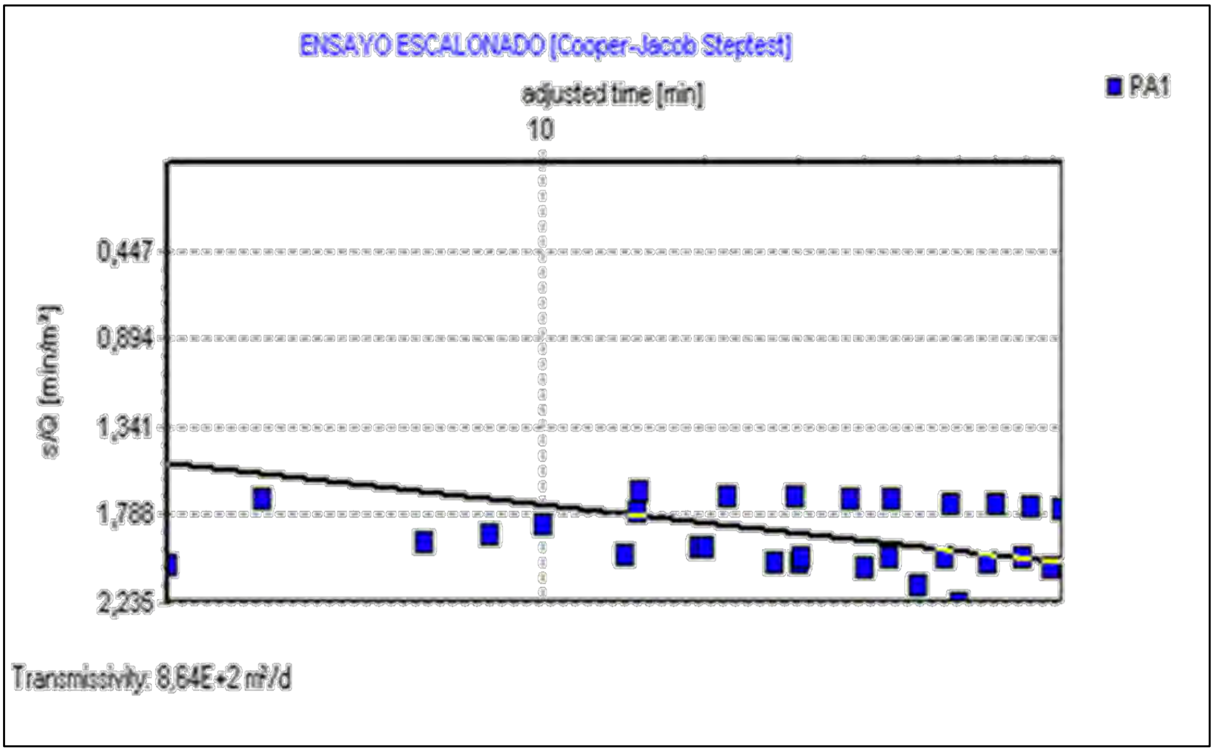


Figure 14-11: Cooper-Jacob Step Test

Considering that the well is totally penetrating, given it was possible to drill the hydrogeological basement at a depth of 127 m, made up of limestone from the Fm. San Juan, and that the static level is at 36.51 m, the saturated thickness (m) is 90 m.

Consequently, the permeability is: $\boldsymbol{k = \frac{T}{m} = \frac{864}{90} = 9.6\ m/day}$

14.1.2.5.6 Descent Prediction Test

The Aquifer Test software allows obtaining the presumption of the drawdowns that would be observed in the well for infinite pumping scenarios. In the case at hand, the following data was entered:

- T = Calculated by Cooper Jacob's solution= 864 m^2/day.
- Flow= The optimum estimated by the relational curves: 230 m^3/day.
- S = Storage coefficient. Estimated according to the sedimentary characteristics of the aquifer= 0.001.
- R = Distance at which it is desired to know the drawdown; in this case, practically, in the well itself: 0.1 m
- t, Pumping time: Arbitrary value of 10,000 minutes, approximately 7 continuous days.

Figure 14-12 corroborates that with a flow rate of 230 m^3/h and after 7 days of uninterrupted pumping, the static level inside the well would drop 10.7 m, or in other words, the dynamic level stabilises at approximately -47 m depth.

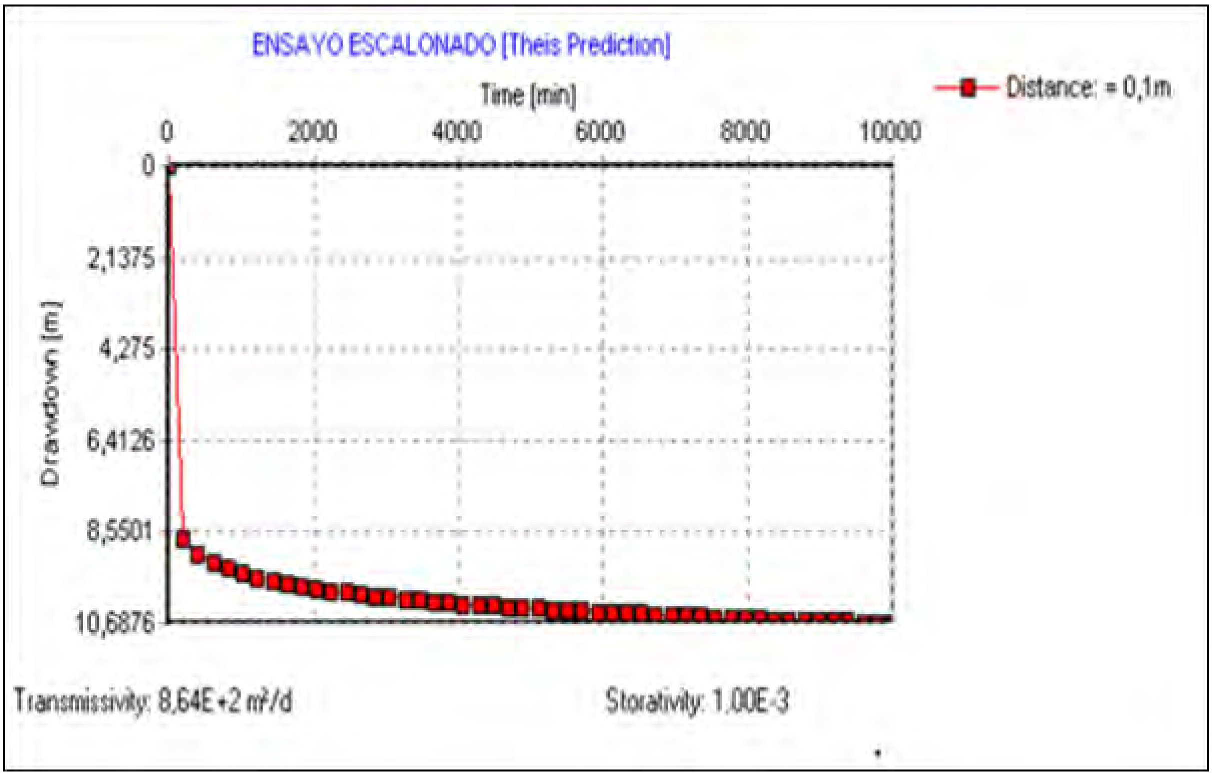


Figure 14-12: Staggered Test Theis Prediction

14.1.3 Pit Groundwater and Dewatering

In the open pit mining sector the level of the base of the RF39 pits is 1700 metres above sea level. This will be well above the estimated water table (1460 metres above sea level) which will avoid the requirement to dewater the pits.

14.1.4 Overall Site Water Balance

14.1.4.1 *Operation Phase – 1 Mtpa*

In the operation phase the EIA adds the following for water usage:

- 5.5 litres/second for dust suppression and other uses industrial.
- 0.41 litres/second water for camps and facilities. (This includes water consumption for domestic use in the camp at a rate of 150 litres/person/day and drinking water consumption for of 0.013 litres/second (at a rate of 5 litres/person/day) based on a 240 person camp
- 24.4 litres/second for process plant

This equates to 5.9 L/s for the camp (at 240 people) plus 24.4 L/s for the 1 Mtpa process plant.

14.1.4.2 *Construction Phase*

During construction phase an average consumption of 3 L/sec is estimated. This includes use for:

- Civil works for dust mitigation and contingent situations.
- Water consumption for domestic use in the camp is also expected to be around 0.6 litres/second (at a rate of 150 litres/person/day) and a consumption of water for human drinking of 0.02 litres/second (at a rate of 5 litres/person/day) based on 350 people.

14.2 Hydrology

This section details the baseline conditions of the existing water resources in the "project footprint" and hydrological study area of the Hualilan project, for which the physical and geographical environment of the region, surface and underground hydrography are described. It also indicates the current state of the quality of groundwater in the area under study.

14.2.1 Water Management Setting

The Hualilan project is located in the northwestern corner of the Pampa de Gualilán, in the area of the Central Precordillera. In hydrographic terms, the Pampa de Gualilán, which represents the denominated "hydrographic study area", constitutes an endorheic basin that receives occasional contributions from the drainage network of the mountain ranges that delimit it on its four flanks.

These contributions are represented by the basins of the Quebradas de La Invernada (to the West), Quebrada de La Cienaguita (to the North West), the Agua del Médano River (to the Eeast) and La Crucecita (to the South) respectively.

14.2.1.1 *Quebrada de La Invernada Basin*

The Quebrada de La Invernada Basin develops between the Sierras de la Invernada to the East and Tigre to the West, with their sources located on the Morterito hill.

The main collector, which receives the name of Quebrada de la Invernada, has an almost straight sub-meridional course heading north and lies on the western flank of the homonymous mountain range. At the north end of the mountain range, it is intercepted by a ravine, through which the route of National Route 149 runs, receiving at this point the contributions of the Portezuelo del Colorado, Los Caracoles and the Agua del Molle River. At the height of the El Molle hill, this collector is called Agua del Molle River and runs in a practically easterly direction, with the northern end of the Pampa de Gualilán as the local base level. It is a temporary channel, which only provides flows in the summer. As a result of rainfall events of an exceptional nature, it can provide water to the Ciénaga de Gualilán.

14.2.1.2 *Quebrada de La Cienaguita Basin*

The Quebrada de La Cienaguita Basin drains the western slope of Cerro Blanco and the northeastern portion of Cordón del Peñón. This channel, which usually has a semi-permanent flow in its sources, infiltrates at the outlet of the mountain front, and its occasional contribution can circulate at a subsurface level at the northern end of the Pampa de Gualilán.

14.2.1.3 *Basin of the Agua del Médano River*

The Basin of the Agua del Médano River drains the upper sectors of the Horqueta hill of 2,293 metres above sea level. Due to the marked topographic unevenness with respect to the local base level, the channels tend to deposit in the northeastern vertex of the Pampa de Gualilán. To the south of the Agua del Médano River, the Aguada Azul River is located, which drains the North end of the Sierra de Talacasto and whose occasional flows end up infiltrating at the outlet of the mountain front without reaching the pampas.

14.2.1.4 *La Crucecita Basin*

The La Crucecita Basin comprises the runoff that drains the northern flank of the La Crucecita Sierra and ends up infiltrating the alluvial deposits of the Bajada de Gualilán.

As with the Ciénaga de Gualilán, located in the extreme southeast of the pampa, it occasionally constitutes a lagoon during the summer season for only a short period of time, due to the combination of intense sun and the strong winds, which, as a whole, cause a rapid evaporation of the water.

A special note applies to the La Ciénaga and Cienaguita springs that emerge at the foot of the Lower Paleozoic calcareous sedimentary cord, located to the West of the Pampa de Gualilán, which is affected by a fault on its eastern flank. This fracture, responsible for the uplift of the calcareous cord, would act as a hydrogeological barrier to the subterranean flow coming from the West.

In Cienaguita, four springs have been recognized, aligned in a practically southern direction, which dispense flows between 12 l/s and 19 l/s. In Table 14-2, a detail of the location and flows discharged by these springs is presented according to information provided by Challenger.

In hydrochemical terms, these are waters with a neutral pH (8.1 - 8.4 pH) and electrical conductivity varying between 721 and 725 microsiemens/cm, being included in Class II (moderate saline hazard).

Table 14-2: Location and Flow Rates of Springs

Spring	Latitude S	Longitude W	Q (l/s)
1	30° 49’ 53,49"	68° 56’ 51,86"	18.0
2	30° 49’ 56,74"	68° 56’ 51,86"	17.5
3	30° 49’ 59,48"	68° 56’ 44,78"	19.0
4	30° 50,70’ 11"	68° 56,41’ 48,2"	12.0

At La Ciénaga, there is another spring located at the intersection of the geographical coordinates 30° 51' 45.43" Lat. South; 68° 56’ 42.67” Long. West, which distributes a flow of the order of 13 l/s). The water has a neutral pH (8.6 pH) and the electrical conductivity measured was 724 microsiemens/cm, being included in Class II (moderate saline hazard).

14.2.2 Hydrology for the project area

From the hydrological point of view, the "hydrologic study area" is an endorheic basin, that is a closed drainage basin in which the precipitations that fall in it do not flow and can only leave the superficial hydrological system by evaporation or infiltration.

In terms of the portion of the territory to be occupied by the facilities that will integrate the mining-industrial complex (“project footprint”), the contribution basin covers an area of 317 km^2 and has three rivers of a temporary or non-permanent nature, namely:

- The La Invernada River, which runs through the homonymous ravine, from the south and
- The Agua del Molle and Colorado Rivers that collect the contributions that come from the northern part of the basin. See Figure 14-13.

Figure 14-13: Hydrologic basin for the Project area

14.2.2.1 ***Delimitation of basins***

A Digital Elevation Model (DEM) is a graphical representation of a continuous surface, usually referred to as a land surface. Slope maps, slope aspect, shadow relief, perspective views, runoff models, etc., are some of the products derived from the topographic analysis of a DEM.

From DEM, a data set is obtained that describes the drainage patterns of the basins and allows the delimitation of these and the drainage network itself. These products are the databases for the hydrological project and are listed below: flow direction, accumulated flow, channel definition, channel segmentation and basin delimitation.

Some parameters are obtained automatically using the GvSIG software, while the data necessary for the indices are extracted from tables generated from the rasterized topography maps (from the digital terrain model or DTM), from basins and from rivers; and later treated in a spreadsheet in the Excel program. See Figure 14-13 and Figure 14-14.

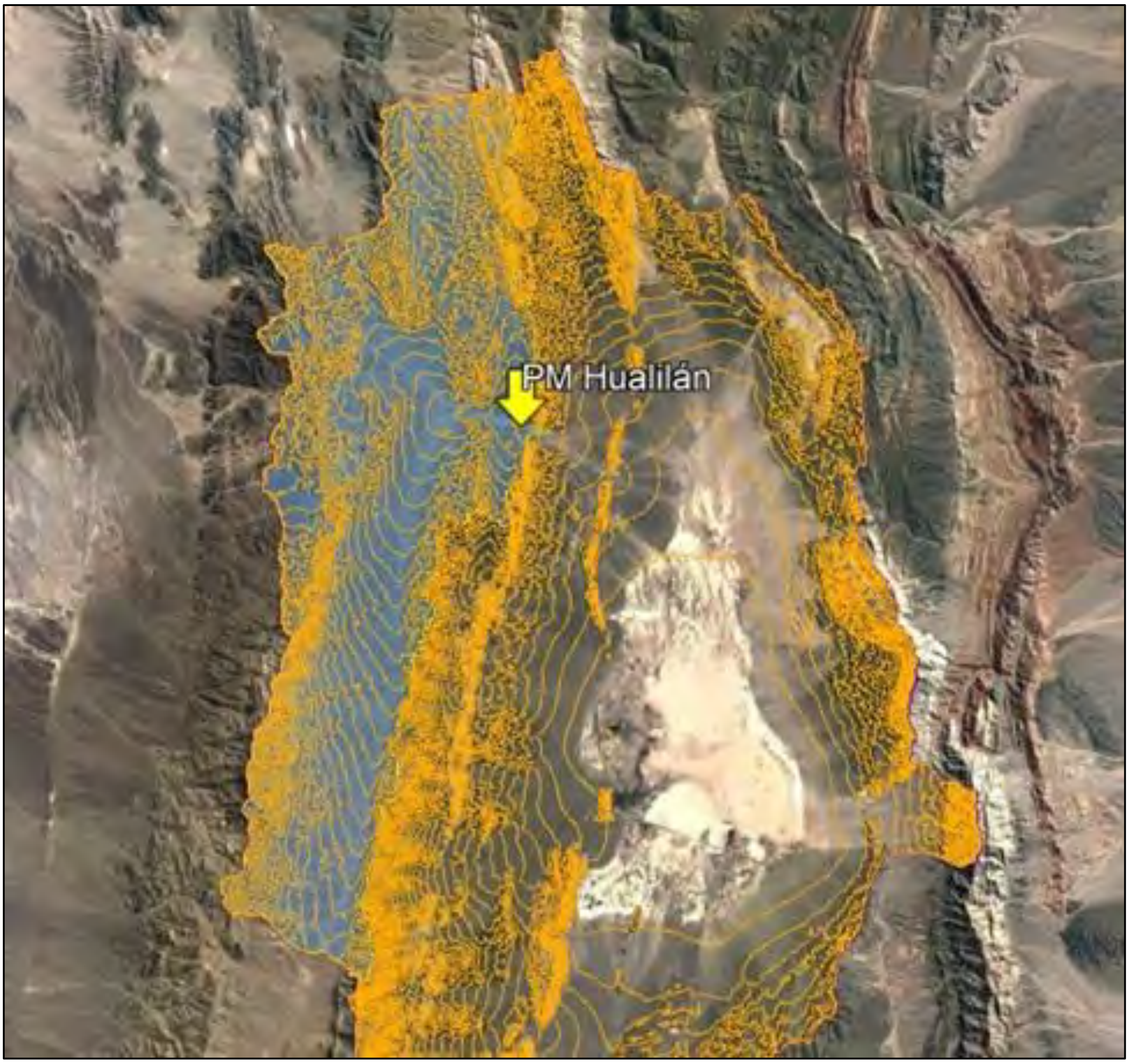


Figure 14-14: Topographic curves (50 m) of the regional basins involved in Hualilan

The physical characteristics of a basin depend on the morphology (shape, relief, drainage network, etc.), the types of soil, the vegetation cover, the geology, the land uses, etc. These characteristics have a decisive influence on the hydrological response of the basin.

The basic geomorphological parameters to establish a hydrological affinity between comparable basins are: areas, perimeter, maximum and minimum height and slope, compactness index, radius of elongation, hypsometric curve, altimetric frequency curve, average height, length of the main channel, slope Mean Mainstream, Weighted Mainstream Slope, Equivalent Rectangle, Slope Index, Slope, Total Mean Slope, Massive or Martone Coefficient, Orographic Coefficient.

These geomorphological parameters can be easily calculated using geographic information systems (GIS) that can then be integrated into hydrological models.

Table 14-3 details the main morphometric parameters of the basins. The methodology applied for the purpose of determining the basic and derived parameters is described in the Environmental Impact Assessment.

Table 14-3: Main Parameters of the Regional Basins

Basin	Area (km^2)	Longitude (m)	Upper level (m asl)	Lower level (m asl)	Δ H (m)	Slope (%)
A	91	24.2	3,236	2,056	1180	4.88
B	151	31.5	3,157	2,056	1101	3.49
C	65	22.7	3,222	1,997	1225	5.41
D	2	2.4	2,163	1,996	167	6.87
E	8	5.3	2,281	1,961	320	6.03
F	2	5.4	2,079	1,783	297	5.49
G	1	3.7	1,990	1,788	202	5.42
H	3	5.0	2,121	1,755	366	7.34

*14.2.2.2 **Hydrological study***

14.2.2.2.1 Background

Based on the collected meteorological information, the average and extreme climatic characteristics prevailing in the region can be accurately inferred. Local stations for data collection are shown in Figure 14-15.

Figure 14-15: Location of pluviometric stations

The Department of Hydraulics of the Province of San Juan, served for several years the meteorological stations "La Toma" (31° 29' South latitude; 68° 46' West longitude; altitude 805 m asl.; period of observations with missing data 1968 - 1994) and “Km 47.3” (31° 32’ South latitude; 68° 52’ West longitude; altitude 945 m asl.; observation period 1989-2000, 12 years).

The first, exclusively climatic, was located in the vicinity of Aviadores Españoles street, about 1.3 km southwest of its intersection with Hermogenes Ruiz, in the town of Ullúm, close to the Punta Negra diversion dam; the second, meteorological and hydrometric, was located in the San Juan River canyon, at 47.3 kilometre point of Provincial Route No. 12. Both have high-value records.

The climate is the result of numerous and complex factors that affect and define the characteristics, magnitudes, and temporal and spatial distribution of the intervening variables, among which it is worth mentioning, orography, location and displacement of high and low pressure centres, heights with respect to sea level, etc. Due to this, in general the prevailing climate in an area is the result of regional characteristics. It should be borne in mind that, according to (Wiesner, 1970), regions of "meteorological homogeneity" are defined "as areas where all points experience events through the same storm mechanisms and total air movements, but not necessarily with the same change in humidity or with equal frequency.

Consequently, the definition of such zones depends mainly on the location and extent of the air masses. In particular, it must consider storm fronts, the properties of the upper air, the topography, its location with respect to general circulation and the internal factors that cause rain" (Caamaño Nelli and Dasso, 2003).

According to these concepts, it is valid to extend the mean and extreme values of the climatic variables registered in the stations "La Toma" and "Km. 47.3" to the entire study area. On the basis of climatic information in general and rainfall in particular, the bibliography consulted on the subject and the knowledge of the region, the main climatic characteristics of the area are described below.

The corresponding climatic statistics are summarised in Table 14-4.

Table 14-4: Key Climatic Statistics from Km 47.3 and La Toma

Location	Km 47.3	La Toma
Average Annual Precipitation	128 mm	106 mm
Average Annual Temperature	19.2ºC	18.8 ºC
Average Annual Maximum Temperature	25.1 ºC	24.5 ºC
Average Annual Minimum Temperature	13.4 ºC	10.5 ºC
Absolute Maximum Temperature	42.5 ºC	42.1 ºC
Absolute Minimum Temperature	-2.9 ºC	-9.6 ºC
Average Annual Relative Humidity	46%	46%
Average Annual Steam Tension	10.3 mb	12.7 mb
Average Temperature Dew Point	5.9 ºC	8.8 ºC
Average Annual Evaporation	1,540 mm	1,923 mm
Average Annual Daily Evaporation	4.2 mm	5.3 mm
Frequency Days with Precipitation	44	15

For the determination of the probable maximum precipitations in the surroundings of the pluviometric stations in the region, the station Km 47.3 has been chosen, as it is the most unfavourable (Table 14-5).

Table 14-5: Rainfall Station Km 47.3 Data (1989/2008)

Date	Maximum rainfall (mm per day)
18/11/1989	27.4
6/2/1990	15.8
12/3/1991	25.8
30/8/1992	15.6
5/3/1993	18.5
25/2/1994	32.9
28/9/1995	11.3
20/11/1996	44
1/1/1997	10.8
16/2/1998	13.6
31/12/1999	20.2
21/2/2000	17.8
2001	24.8
2002	19.1
2005	45.7
2006	34.1
5/4/2007	7.4
27/1/2008	59.6

14.2.2.2.2 Characteristics of rainfall

According to the results of the analysis carried out on the available pluviometry information and in coincidence with what happens in general in the arid region of the Northwest of Argentina, the maximum rains prone to cause considerable floods occur in the summer period between December and March. These are usually generated by convective processes, associated with condensation by expansion due to frontal and orographic causes, a phenomenon that usually occurs during days of high temperature and significant solar radiation.

The precipitations in question are characterised by their short durations, high intensities and reduced coverage areas. Furthermore, in most cases they are preceded and accompanied by thunder and lightning. The floods that they eventually generate present alluvial hydrographs with short base times and significant maximum or “peak” flows. In short, rain-runoff events are characterised because they occur suddenly, with a marked torrential nature and in general with very high solid costs (amount of sediments carried by the flow).

In summary, the analysis of the existing information and the experience indicate that, in the region, normally the rains that originate the avenues with the highest maximum flows, are the summer ones of short duration and consequently high intensity; in general, those whose duration is less than sixty minutes and exceptionally up to ninety minutes. The floods are maximised, when intense rains occur at a time when the infiltration rates are reduced, mainly due to previous precipitations that reduce the moisture deficit of the land and the environment.

In the absence of rainfall records, the Intensity-Duration-Frequency (IDF) curves can be made based on the analysis of statistical methods of annual maximum daily precipitations observed in pluviometry stations in the region. The probable maximum intensities for durations of rains less than 24 hours are then calculated by means of some semi-empirical expression.

It should be clarified that "annual maximum daily precipitation" is understood to be the maximum height of water recorded in twenty-four hours in a given year. These are obtained from the values recorded by automatic stations or from the monthly sheets made by the observers of the rain gauges, who in most cases only make daily readings without generally consigning the type of precipitation.

The present study and pluviometry analysis is carried out on the basis of the data duly obtained in the aforementioned stations.

Once the available information was available and systematised, the annual maximum daily rainfall database received statistical treatment using the "Gumbel probabilistic method" in order to calculate the Probable Maximum Rainfall (PMP) corresponding to different return periods.

As a conclusion of what was previously exposed regarding the subject, it is inferred without a doubt that the rains most likely to cause considerable floods are those of short duration and high intensity; in particular, when they occur when the infiltration rates are low, mainly due to previous rains that reduce the moisture deficits of the land and the environment.

14.2.2.2.3 Statistical analysis of rainfall information

If rainfall records of a representative number of years were available, the accumulated precipitation heights could be plotted in ordinates and time in abscissas, denominating instantaneous intensity of rain (i) in a given interval Δt, when this tends to zero, a:

$$i = \lim \frac{\Delta h}{\Delta t}, \text{when } \Delta t \to 0$$

If a given time interval is considered for a given rain, for example 10 minutes, there will be an average value of the intensity (I) for said interval; similarly, there will be mean values of I for other intervals of 5, 20, 30 minutes, etc. Analysing the values of (I) obtained, for all the rains recorded in a period of N years, the values of probable maximum intensities I can be obtained for different recurrences and durations, which on average can be equated or exceeded only once during the period of return (Tr) considered in each case.

The statistical analysis of the maximum annual precipitations registered in the stations in question can be carried out by assigning probabilities to the precipitations by the Weibull formula. The procedure for applying this consists of ordering the data in descending order (from highest to lowest and independent of the date of occurrence of the precipitation) and associating a return period with each value given by the following expression:

$$T_r = \frac{(N+1)}{c}$$

Where:

c = order number of the maximum annual precipitation corresponding to the decreasing ordering (independent of the date of occurrence).

N = total number of years of the record.

Tr = return period in years.

The analytical adjustment of the data by means of the Gumbel probabilistic method allows obtaining a precipitation value associated with a given return period. The Gumbel expression, derived from the cumulative distribution function, is:

$$P_T = \frac{-ln\left\{-ln\left\{1-\frac{1}{T}\right\}\right\}}{a} + \mu$$

Where:

PT = annual maximum daily precipitation (in mm) corresponding to a return period (Tr).

Tr = return period in years.

µ; a = parameters of the Gumbel distribution that are calculated using the following formulas:

$$Sx^2 = \frac{\sum(Xi - Xm)^2}{(N-1)}$$

Where:

Xm = arithmetic average of the sample (in mm.).

Sx = standard deviation of the sample (in mm.)

Yn = mean value of the reduced variable

Sn = standard deviation of the reduced variable

The parameters of the reduced variable (Yn; Sn) are only a function of the sample size or extension of the data record (N). They are obtained from a table provided by the manuals in which the Gumbel distribution appears; the parameters vary between the following extreme values:

Yn = 0.50 and Sn = 0.95 (for N = greater 10 years)

Yn = 0.57 and Sn = 1.28 (for N greater than 100 years)

The method described above has allowed for different recurrences that in hydrology is called "Probable Maximum Precipitation" or simply "PMP" associated with a given recurrence. If desired, the distribution of the maximum annual monthly rainfall observed could be plotted in a semi-logarithmic representation and the adjustment line resulting from applying the Gumbel probabilistic model can be obtained.

The data could also have been processed using the lognormal method, in order to determine the probable maximum values corresponding to different return periods. As a result of this methodology, the graph of the logarithmic-normal probability distribution of the annual maximum daily rainfall could be obtained.

Due to the fact that there is no rainfall information available that allows an analysis of the intensity of rains lasting less than 24 hours, which would allow directly obtaining the Intensity - Duration - Frequency (IDF) curves corresponding to different return periods, these can only be obtained by applying semi-empirical methods.

The result of the statistical treatment applied to the data from Station Km 47.3 can be seen in Table 14-6 and Table 14-7.

Table 14-6: Rainfall Frequency at Station Km 47.3

Order	Rainfall	Return period	$(x_i - x_m)^2$
1	59.6	19.00	3552.16
2	45.7	9.50	2088.49
3	44.0	6.33	1936.00
4	34.1	4.75	1162.81
5	32.9	3.80	1082.41
6	27.4	3.17	750.76
7	25.8	2.71	665.04
8	24.8	2.38	615.04
9	20.2	2.11	408.04
10	19.1	1.90	364.81
11	18.5	1.73	342.25
12	17.8	1.58	316.84
13	15.8	1.46	249.64
14	15.6	1.36	243.36
15	13.6	1.27	184.96
16	11.3	1.19	127.69
17	10.8	1.12	116.64
18	7.4	1.06	54.76

Table 14-7: Maximum Probable Precipitation at Rainfall Station Km 47.3

Return Period (Years)	Maximum Probable Precipitation (mm)
5	37.8
10	47.8
15	53.5
20	57.4
25	60.4
30	62.9
40	66.8
50	69.8
100	79.1
200	88.4

14.2.2.2.4 Intensity-Duration-Frequency curves (IDF)

Grunsky's formula (and other similar ones) allow for analytically determining the probable maximum intensities corresponding to rains of different durations and for different recurrences; in the present case through the simple application of the expression:

$$I_t = I_{24} \times (24 / t)^{0.5}$$

As already mentioned, the "design rains" or PMP to be considered in the hydrological sizing of flood channelling and evacuation works, are those whose durations are equal to the concentration times of the basins. These are then calculated using the anticipated formulas and with the averaged values for each basin, the different probable maximum intensities are obtained from the IDF curves.

In the hydrological study, the calculations of the maximum probable design rainfall were made on the basis of the maximum intensities obtained in the probabilistic analysis carried out on the station data (Figure 14-16).

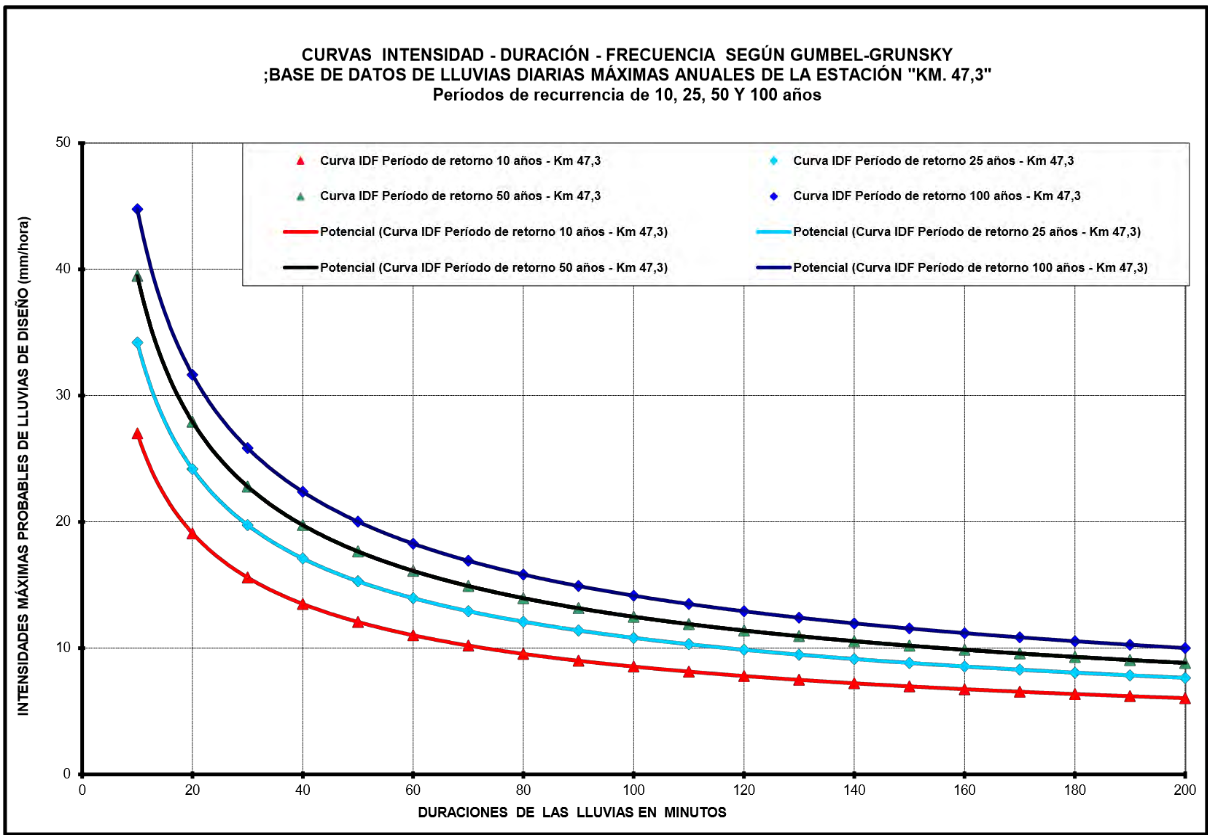


Figure 14-16: Intensity - Duration – Frequency curves for Station Km 47.3

14.2.2.2.5 Determination of probable maximum flows

Probable maximum flows were determined for return periods of 25, 50 and 100 years using the Generalised Rational, Témez and Soil Conservation Service Methods. The results are shown in Table 14-8 to Table 14-10.

Table 14-8: Determination of probable maximum flows for a return period of 25 years

							P_{max} T_r 25yrs 60.4 mm	M.G.R.	Témez	S.C.S			Q weighted
Basin	**Area (km^2)**	**Main Channel (km)**	**Δh (m)**	**i (m/m)**	**Time (hours)**	**Tc min**	**I mm**	**Q_{max} $T_{r\ 25\ yrs}$ (m^3/s)**	**Q_{max} $T_{r\ 25\ yrs}$ (m^3/s)**	**T_c (hours)**	**T high (hours)**	**Q_{max} T_r 25 yrs (m^3/s)**	**$T_{r\ 25\ yrs}$ (m^3/s)**
A	90.80	24.20	1,180	0.049	6.00	360	5.0	37.60	36.99	6.00	4.6	54.7	43.9
B	150.60	31.50	1,101	0.035	7.81	469	4.4	42.25	51.82	7.81	5.7	56.7	50.6
C	65.20	22.65	1,224	0.054	5.59	336	5.2	31.70	29.66	5.59	4.4	47.0	37.0
D	1.52	2.44	167	0.069	0.98	59	12.4	2.50	2.40	0.98	1.6	4.3	3.2
E	8.29	5.31	320	0.060	1.82	109	9.1	9.63	7.48	1.82	2.1	17.0	12.1
F	2.49	5.40	296	0.055	1.88	113	9.0	2.94	3.34	1.88	2.1	5.2	3.9
G	1.29	3.73	202	0.054	1.42	85	10.3	1.77	2.15	1.42	1.9	3.1	2.4
H	3.42	4.99	366	0.073	1.67	100	9.5	4.26	4.13	1.67	2.0	7.5	5.5

Table 14-9: Determination of probable maximum flows for a return period of 50 years

							P_{max} T_r 50yrs 69.8 mm	M.G.R.	Témez	S.C.S			Q weighted
Basin	Area (km^2)	Main Channel (km)	Δh (m)	i (m/m)	Time (hours)	Tc min	I mm	Q_{max} $T_{r\ 50\ yrs}$ (m^3/s)	Q_{max} $T_{r\ 50\ yrs}$ (m^3/s)	T_c (hours)	T high (hours)	Q_{max} $T_{r\ 50\ yrs}$ (m^3/s)	$T_{r\ 50\ yrs}$ (m^3/s)
A	90.80	24.20	1,180	0.049	6.00	360	5.8	43.45	42.75	6.00	4.6	71.8	54.4
B	150.60	31.50	1,101	0.035	7.81	469	5.1	48.83	59.89	7.81	5.7	74.5	62.0
C	65.20	22.65	1,224	0.054	5.59	336	6.1	36.64	34.27	5.59	4.4	61.7	45.9
D	1.52	2.44	167	0.069	0.98	59	14.4	2.88	2.77	0.98	1.6	5.6	4.0
E	8.29	5.31	320	0.060	1.82	109	10.6	11.13	8.61	1.82	2.1	22.3	15.2
F	2.49	5.40	296	0.055	1.88	113	10.4	3.40	3.86	1.88	2.1	6.8	4.9
G	1.29	3.73	202	0.054	1.42	85	12.0	2.04	2.48	1.42	1.9	4.1	3.0
H	3.42	4.99	366	0.073	1.67	100	11.0	4.93	4.77	1.67	2.0	9.9	6.9

Table 14-10: Determination of probable maximum flows for a return period of 100 years

							P_{max} T_r 100yrs 79.1 mm	M.G.R.	Témez	S.C.S			Q weighted
Basin	Area (km^2)	Main Channel (km)	Δh (m)	i (m/m)	Time (hours)	Tc min	I mm	Q_{max} T_r 100 yrs (m^3/s)	Q_{max} $T_{r\ 100\ yrs}$ (m^3/s)	T_c (hours)	T high (hours)	Q_{max} T_r 100 yrs (m^3/s)	$T_{r\ 100\ yrs}$ (m^3/s)
A	90.80	24.20	1,180	0.049	6.00	360	6.6	49.24	48.44	6.00	4.6	89.7	65.4
B	150.60	31.50	1,101	0.035	7.81	469	5.8	55.33	67.86	7.81	5.7	93.0	73.8
C	65.20	22.65	1,224	0.054	5.59	336	6.8	41.52	38.84	5.59	4.4	77.1	55.3
D	1.52	2.44	167	0.069	0.98	59	16.3	3.27	3.14	0.98	1.6	7.0	4.8
E	8.29	5.31	320	0.060	1.82	109	12.0	12.62	9.79	1.82	2.1	27.8	18.5
F	2.49	5.40	296	0.055	1.88	113	11.8	3.85	4.37	1.88	2.1	8.5	5.9
G	1.29	3.73	202	0.054	1.42	85	13.6	2.31	2.82	1.42	1.9	5.1	3.6
H	3.42	4.99	366	0.073	1.67	100	12.5	5.58	5.41	1.67	2.0	12.3	8.4

The results were plotted to show the independent hydrographs of each of the basins for the return period of 100 years (Figure 14-17).

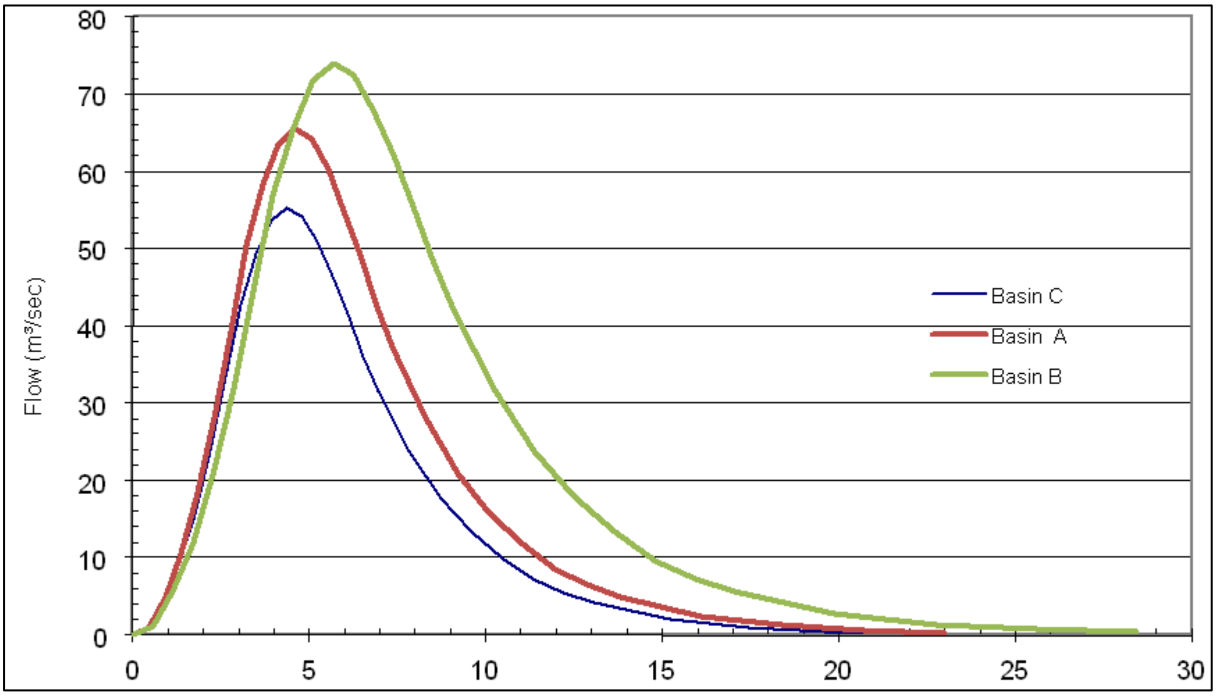


Figure 14-17: Hydrographs for basins A, B and C for T_r:100 years

14.2.3 Surface Drainage and Flood Control

The natural drainage is from north-west to the south-east with the main floodway flowing east through the Gap area between the Central and Northern pits. This existing floodway crosses the proposed location of the treatment plant and ROM pad.

The project design contemplates a western flood diversion bund extending approximately 2.2 kilometres as shown in (Figure 14-18). This bund will be constructed from approximately 65,000 t of the initial unconsolidated cover in the starter open pits. This flood control measures have been designed to accommodate a 1 in 100 year rainfall event which is 76 mm.

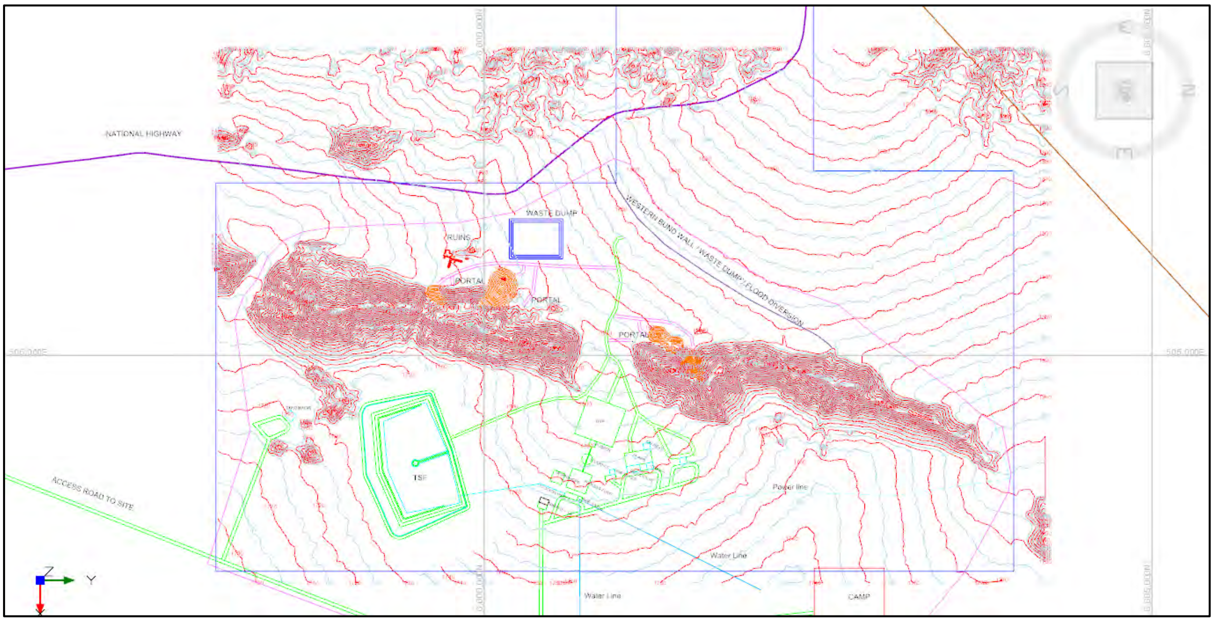


Figure 14-18: Proposed flood diversion

15 MINING

15.1 Introduction

At the commencement of the project a trade-off study was conducted to determine the most appropriate mining method (open pit, underground or a combination of both) that would deliver the greatest economic benefit. The evaluation concluded that underground mining would be the best option, however the waste material obtained by this method would not be sufficient for the construction of the tailings dam, therefore the open pit would be used to obtain the material and take advantage of the surface economic mineral instead of just using the waste rock from borrow pit material. The results of this trade-off are in Appendix 3 – Define the Process.

A second trade-off was performed with the aim of selecting the optimal pit size that meets the tailings requirement, the best underground cut-off grade that allows for the recovery of the highest-grade, and which achieves the greatest economic benefit. The results obtained in this trade off can be reviewed in Appendix 4 – OP and UG Combined.

The pit size and underground cut-off grade chosen in the winning scenario of the second trade off were used as the basis for the development of this section and estimation of the Potential Mineral Inventory (PMI).

The open pit consists of three pits, North, Central, and South, which will be mined using conventional excavator and truck techniques. The underground mining will be carried out below each pit using the longhole open stopping method and includes a 30 m crown pillar between the open pit floor and the upper most underground stope. The mine plan was designed to feed the processing plant at a rate of 1.0 Mt per year or 2,800 tpd.

The total mine life is expected to be 8 years with one year of pre-production and 7 years of production. Six months prior to the pre-production year the open pit mining contractor will mobilise to commence pioneering works for the pits and commence establishing roads and the tailings storage facility (TSF).

During the pre-production year, the PMI extracted from the pit will be stockpiled and the waste will be sent to TSF for embankment construction, which will initially have capacity for two years of processing production. Underground mining will also commence at the start of the pre-production year in the north and south zones. In the first year of production, the plant will be fed with PMI from the high grade stockpiles, open pit and underground. From the second year onwards, the plant will be supplied with PMI primarily coming from the underground mine.

15.2 Geotechnical data review

The geotechnical study described in Section 13 (AKL Ingenieria y Geomecanica Ltda., 2023) was carried out at the same time as this Scoping Study and considers that the mining method will be only by open pit, therefore the evaluation considers a larger pit size.

From the trade-offs done (Appendix 3 – Define the Process and Appendix 4 – OP and UG Combined), a smaller pit size was selected to allow early start-up of underground mining, and more conservative angles were assumed for pit optimisation, see Table 15-1 below. During the Pre-Feasibility Study an update will be made to the pit optimisation considering the geotechnical information described in Section 13 and the impact of the cover material.

Table 15-1: Overall angles used for open pit optimisation

Sector	OSA (°)
East	50
West	45

Based on the available geotechnical data, and on similar styled mines, the following stoping assumptions were used for the purposes of the Underground Mining Scoping Study:

- Level spacing (floor to floor) - 20 m
- Strike length (regardless of stope width) - 20 m
- Minimum stoping width – 2.5 m
- Minimum crown pillar of 30 m below the designed open pits.

Additional pillar requirements were not considered for the purposes of the study, due to the paste backfill system removing the need for them.

A 25 m sublevel system was considered to be impractical due to the dip of the orebody requiring stoping blocks to be >25 m high, on dip.

15.3 Hydrogeology and Hydrology review

The hydrogeological information presented in this section was taken from Chapter 2 and Chapter 4 of the Environmental Impact Assessment developed in 2023 by San Juan Mining Consultants for Challenger Exploration (Consultores Mineros de San Juan, 2023).

To identify and quantify the water supply source for the mining project, a series of studies and investigations have been carried out in the northern third of the Pampa de Hualilan, which receives occasional contributions from the Agua del Molle River to the northwest, from the Quebrada de La Cienaguita and the Cajon River to the north, as well as occasional contributions from the Agua del Medano River to the east.

The northern third of the Pampa de Hualilan is located at a higher structural level than the rest of the basin, because of a system of NW - SE faults recognizable from satellite images (INA / CRAS, 2021).

In this area and to define the subsoil geology and the geometry of the aquifer, two geophysical prospecting campaigns were carried out by means of Vertical Electrical Sounding (VES). The first campaign (May 2021) was overseen by INA/CRAS and included five VES and the second campaign (September 2022) was overseen by Eduardo Morell and included five VES.

According to the information presented, the water table is at a level of 1460 m ASL, which does not affect the work to be carried out in the pit. However, MP recommends for the next stage of the (PFS), to carry out a new hydrogeological study to determine the impact on the underground workings. More detailed information can be found in Section 14.

15.4 Open Pit

Open pit mining considers three pits (North, Central and South) which will be mined by conventional techniques using excavators and trucks.

Mineralised material from the pits will be sent to the processing plant or stockpile as required. The competent waste material will be hauled directly to the TSF for the construction of the embankments.

15.4.1 Block model

The resource block model, described in Section 12.6, comprised 10 m x 20 m x 10 m blocks with 2.5 m x 5.0 m x 2.5 m sub-cells surrounding the mineralised body. For the analysis developed by open pit, the block model was regularised to 5 m x 5 m x 5 m. In the regularisation the PMI tonnage decreased by 3.9% due to some inferred category blocks becoming undefined in the re-blocking, and the gold grade is diluted from 0.847 g/t to 0.781 g/t giving a result of 11.4% less gold metal content in the block model.

Table 15-2 and Figure 15-1 show the change in gold metal content on regularisation and the variation in PMI. These were generated with the material that could potentially be mineralised material, i.e. material inferred and indicated from the whole block model.

Table 15-2: Block model regularisation

Block model	COG Applied (g/t)	PMI (Mt)	Au Grade (g/t)	Au Contained (koz)	Change in Metal (%)	Change in PMI(%)
Resource Model	0.15	91.4	0.847	2,490		
5 x 5 x 5 reblocking	0.15	87.8	0.781	2,205	-11.4%	-3.9%

Note: Analysis performed on the whole block model

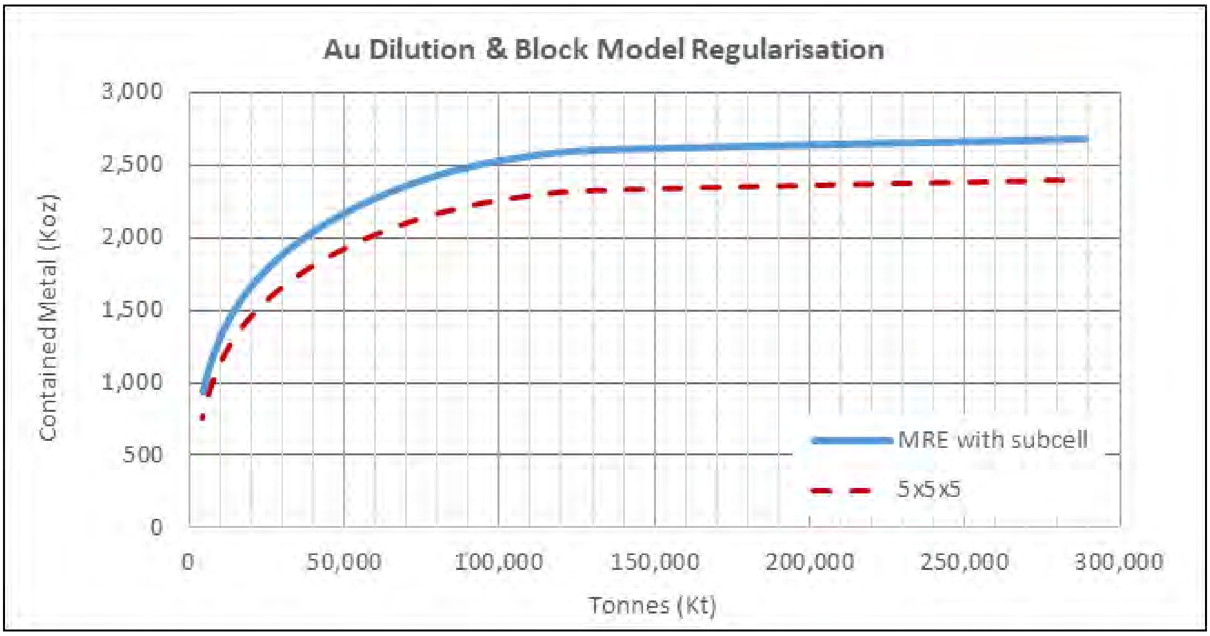


Figure 15-1: Block model regularisation

Mining Plus considers that the variation in tonnage and metal is within the acceptable range, therefore it was determined to continue the development of the study with the regularised model at 5 m x 5 m x 5 m. Table 15-3 summarises the block model dimension, extent and block size.

Table 15-3: Block model framework

Description	East	North	Elevation
Model Origin	503750	6599100	1100
Maximum Extension	505300	6602100	2000
Block size (m)	5	5	5
Block count (m)	310	600	180

Block model items used for open pit evaluation are presented in Table 15-4.

Table 15-4: Open pit block model items

Item	Unit	Description
AU	g/t	Gold grade
AG	g/t	Silver grade
PB	%	Lead grade
ZN	%	Zn grade
RSCAT	unit	Resource category
SG	m3/t	Density
OXIDE	unit	Rock type
SLOP2	°	Geotechnical sector
PTOPO	%	Percentage below topo

The block model considers three (3) rock types as in Table 15-5. The cover material is considered as topsoil.

Table 15-5: Description of rock type codes

OXIDE	Description
0	Undefined
1	Cover
2	Oxide
3	Fresh

15.4.2 Pit optimisation

Pit optimisation analysis was run to understand potential economics of extraction by open pit methods. The pit optimisation was made using the Geovia Whittle® software package and Lerchs-Grossmann algorithm.

A previous trade off analysis was developed to determine the appropriate metallurgical process and obtain the best cash flow considering a processing rate of 1 Mtpa. This analysis is described in Appendix 3 – Define the Process.

The metals considered during the optimisation analysis were gold, silver and zinc. The optimisation was made considering Measured, Indicated and Inferred Resources, with the parameters applied summarised in Table 15-6.

Table 15-6: Pit Optimisation parameters

Parameter	Unit	Values		
		If Au <1.5 g/t	If Au ≥ 1.5 g/t & Zn <1.5%	If Au ≥ 1.5 g/t & Zn ≥ 1.5%
Metal price				
Au	US$/oz	1700	1700	1700
Ag	US$/oz	20	20	20
Zn	US$/lb	1.3	1.3	1.3
Mining recovery and dilution				
Mining Recovery	%	95.0%	95.0%	95.0%
Mining Dilution	%	-	-	-
Mining cost				
PMI (average)	US$/t mined	2.5	2.5	2.5
Waste (average)	US$/t mined	2.5	2.5	2.5
Cover material	US$/t mined	2.0	2.0	2.0
Incremental cost	US$/t mined	0.04 Down /0.02 Up 10 m		
Processing cost				
Processing cost	US$/t proc	8.99	11.8	15.95

Sustaining capex	US$/t proc	0.5	0.5	0.5
Sustaining tailings	US$/t proc	1.0	1.0	1.0
G&A				
G&A cost	US$/t proc	3.91	3.91	3.91
Metallurgical recovery				
Au	%	86.2%	94.5%	94.5%
Ag	%	77.8%	90.5%	90.5%
Zn	%	-	-	85.0%
Selling cost				
Selling Cost Au				
Au Payable	%	94.1%	94.1%	94.1%
Transport & marketing	US$/oz	50.0	-	-
Royalty	US$/oz	60.0	60.0	60.0
Selling cost Ag				
Ag Payable	%	82.9%	82.9%	82.9%
Royalty	%	3.0%	3.0%	3.0%
Selling cost ZN				
Zn Payable	%	-	-	85.0%
Royalty	%	-	-	3.0%
Taxes				
Tax on Concentrate	%	8%	8%	8%
Tax community	%	1.5%	1.5%	1.5%
Cutoff Grade				
Economic cut-off grade	g/t	0.45	0.47	0.56

The overall slope angles used for pit shell optimisation are shown in Table 15-1.

The pit with the highest NPV is Revenue Factor RF 0.96, see Table 15-7 and Figure 15-2. However, from the evaluations carried out in trade off 1 (Appendix 3 – Define the Process), it was determined that the most optimal mining method is underground and the pit would be used for the extraction of the material required for the construction of the tailings dam and provide PMI in the earliest years. The optimum pit size that can meet this requirement and, in conjunction with underground mining, gives the best cash flow is the RF 0.39 pit. This analysis can be reviewed in Appendix 4 – OP and UG Combined.

Table 15-7: Whittle pit optimisation

Pit	Revenue Factor	Best Case (M US$)	Specific Case (M US$)	Worst Case (M US$)	PMI (Mt)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Waste (Mt)	Total (Mt)	SR	LOM
1	0.31	151.6	151.6	151.6	1.0	4.00	25.06	1.90	0.26	3.9	4.9	4.01	1.0
2	0.33	152.5	152.5	152.5	1.0	3.97	24.99	1.90	0.26	4.0	5.0	4.06	1.0
3	0.35	163.0	162.9	162.9	1.1	3.84	24.34	1.77	0.25	5.6	6.7	4.94	1.1
4	0.37	163.7	163.6	163.6	1.1	3.83	24.23	1.76	0.25	5.7	6.8	4.98	1.1
5	0.39	177.5	176.8	176.8	1.4	3.58	22.96	1.56	0.23	8.4	9.8	6.02	1.4

Pit	Revenue Factor	Best Case (M US$)	Specific Case (M US$)	Worst Case (M US$)	PMI (Mt)	Au (g/t)	Ag (g/t)	Zn (%)	Pb (%)	Waste (Mt)	Total (Mt)	SR	LOM
6	0.41	180.1	179.2	179.2	1.4	3.53	22.76	1.53	0.23	8.9	10.3	6.15	1.4
7	0.43	216.3	211.7	211.7	2.2	3.13	17.10	1.20	0.17	17.7	19.9	8.01	2.2
8	0.45	216.9	212.2	212.2	2.2	3.12	17.06	1.19	0.17	17.8	20.1	7.99	2.2
9	0.47	218.8	214.0	213.7	2.3	3.07	16.74	1.16	0.17	18.3	20.6	7.95	2.3
10	0.49	221.4	216.4	215.8	2.4	3.03	16.35	1.16	0.16	19.1	21.5	8.01	2.4
11	0.51	223.5	218.3	217.4	2.5	2.96	15.77	1.12	0.16	19.6	22.1	7.86	2.5
12	0.53	224.0	218.7	217.8	2.5	2.94	15.69	1.12	0.16	19.7	22.2	7.83	2.5
13	0.55	225.4	220.0	218.8	2.6	2.90	15.59	1.11	0.16	20.2	22.8	7.83	2.6
14	0.57	226.3	220.8	219.4	2.6	2.88	15.49	1.10	0.16	20.6	23.3	7.84	2.6
15	0.59	230.2	224.2	221.9	2.8	2.80	14.91	1.06	0.16	22.8	25.6	8.10	2.8
16	0.61	230.6	224.6	222.2	2.8	2.79	14.82	1.06	0.16	23.0	25.9	8.11	2.8
17	0.63	232.0	225.8	222.8	2.9	2.75	14.71	1.04	0.15	23.8	26.8	8.15	2.9
18	0.65	233.3	226.9	223.6	3.0	2.73	14.53	1.03	0.15	24.8	27.8	8.29	3.0
19	0.67	250.7	243.5	233.8	3.6	2.68	13.78	1.04	0.14	39.9	43.5	10.97	3.6
20	0.69	253.1	245.5	233.9	3.8	2.62	13.21	1.00	0.13	42.0	45.9	10.96	3.8
21	0.71	253.8	246.2	232.9	4.0	2.57	13.11	0.98	0.13	42.6	46.6	10.79	4.0
22	0.73	255.6	247.4	231.5	4.2	2.46	12.59	0.93	0.13	44.0	48.2	10.39	4.2
23	0.75	258.3	248.3	228.0	4.7	2.30	11.79	0.85	0.12	46.0	50.8	9.77	4.7
24	0.77	259.2	248.8	227.0	4.9	2.26	11.54	0.84	0.12	47.1	52.0	9.70	4.9
25	0.79	259.7	249.0	226.5	4.9	2.25	11.52	0.83	0.12	48.0	52.9	9.73	4.9
26	0.81	260.4	249.4	226.1	5.0	2.23	11.48	0.82	0.12	49.3	54.3	9.79	5.0
27	0.83	261.2	249.4	225.1	5.2	2.18	11.20	0.80	0.12	50.4	55.7	9.64	5.2
28	0.85	261.5	249.3	224.2	5.3	2.16	11.09	0.79	0.12	51.0	56.4	9.55	5.3
29	0.87	264.4	250.1	212.9	6.3	1.97	11.24	0.73	0.11	59.1	65.5	9.31	6.3
30	0.89	264.6	250.3	212.2	6.4	1.96	11.23	0.73	0.11	59.8	66.3	9.34	6.4
31	0.91	265.0	250.5	209.8	6.6	1.93	11.13	0.72	0.11	62.6	69.2	9.43	6.6
32	0.93	265.1	250.5	208.4	6.8	1.91	11.00	0.71	0.11	63.1	69.9	9.33	6.8
33	0.95	265.1	250.5	207.7	6.8	1.90	10.98	0.71	0.11	63.9	70.8	9.37	6.8
34	0.97	265.5	249.8	198.8	7.6	1.78	10.17	0.67	0.11	68.1	75.7	8.94	7.6
35	0.98	265.5	249.7	198.5	7.6	1.78	10.16	0.66	0.11	68.1	75.8	8.93	7.6
36	0.99	265.5	249.6	197.8	7.7	1.77	10.13	0.66	0.11	68.7	76.4	8.93	7.7
37	1.00	265.5	249.6	197.3	7.7	1.77	10.12	0.66	0.11	68.8	76.5	8.90	7.7

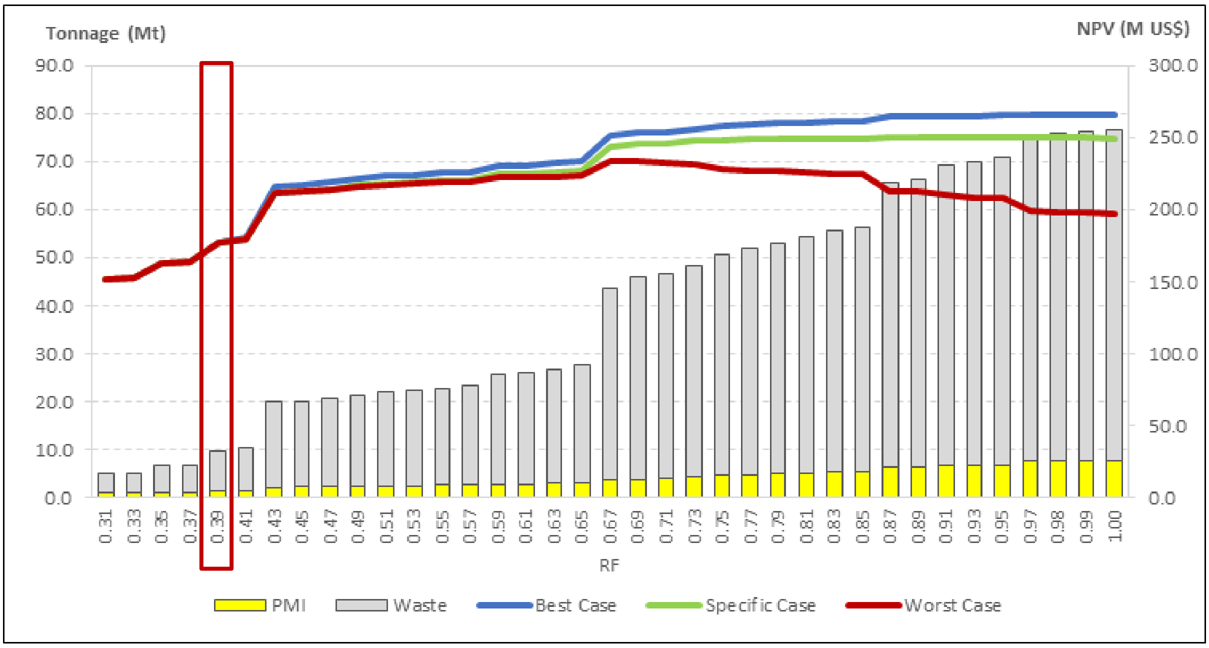


Figure 15-2: Pit by Pit Graph

The material content within the RF39 pit shell, considering only potential mineable areas and mining factors, is shown in Table 15-8.

Table 15-8: Material content in pit shell RF 39

Item	Unit	Pit shell RF 39
PMI	kt	1,311
Au	g/t	3.41
Ag	g/t	22.31
Zn	%	1.52
Pb	%	0.23
Waste	kt	9,552
Total	**kt**	**10,863**

Waste material from the pit will be used for the construction of roads, ROM pad and the tailings storage facility, material not required for construction will be placed on the waste dump. The cover material is considered non-competent and will be sent to the waste dump. The distribution of waste within the pit shell is shown in Table 15-9.

Table 15-9: Waste material content in pit shell RF 39

Waste	Unit	Pit shell RF 39
Cover	kt	791
Fresh	kt	1,403
Oxide	kt	2,815
Not defined	kt	4,474
Total waste	**kt**	**9,552**

Pit shell RF39 consists of three small pits: North, Central and South. Figure 15-3 shows a plan view of the pit shell while Figure 15-4, Figure 15-5 and Figure 15-6 show section views for each pit.

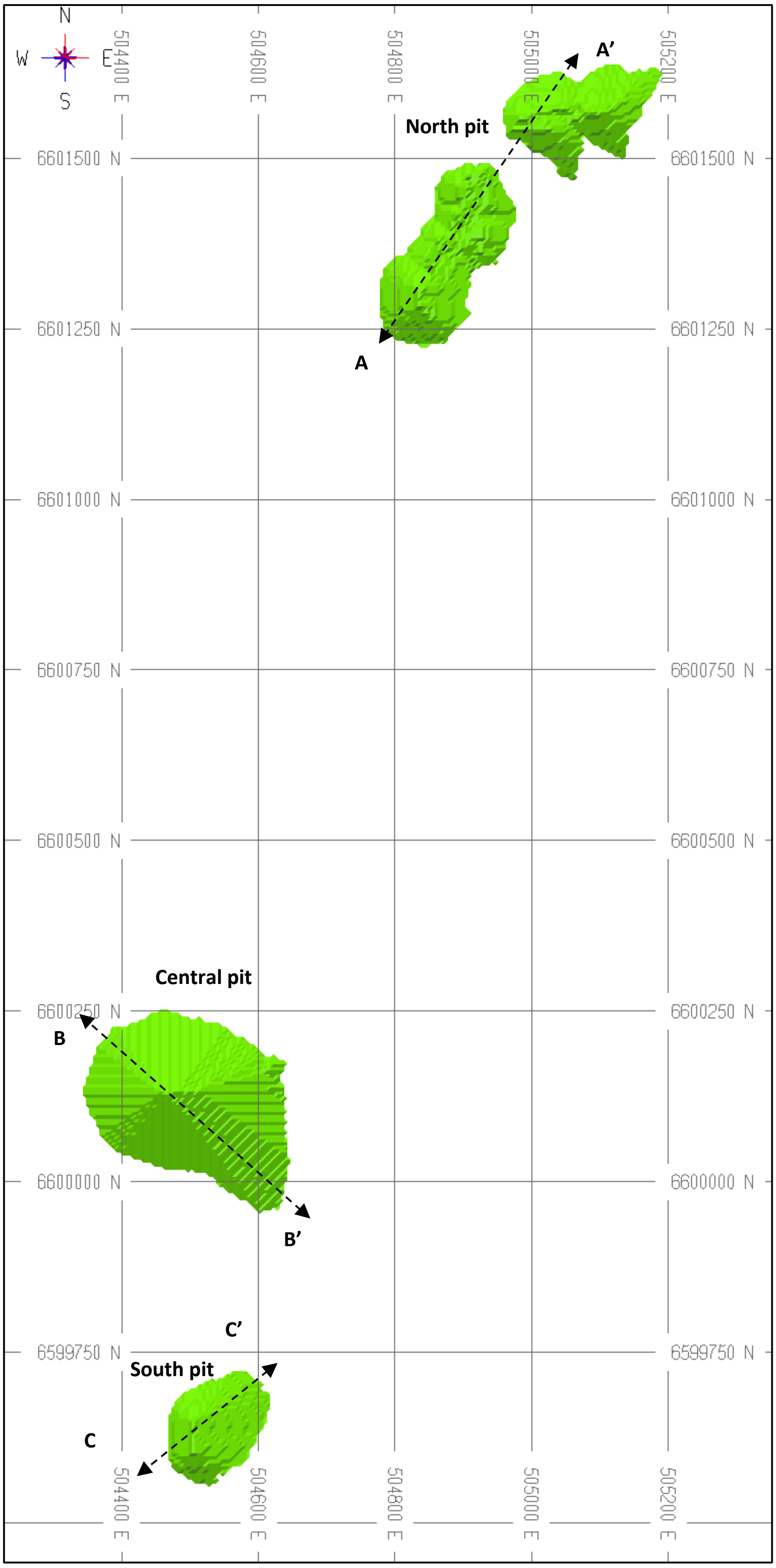


Figure 15-3: Plan view of pit shell RF 39

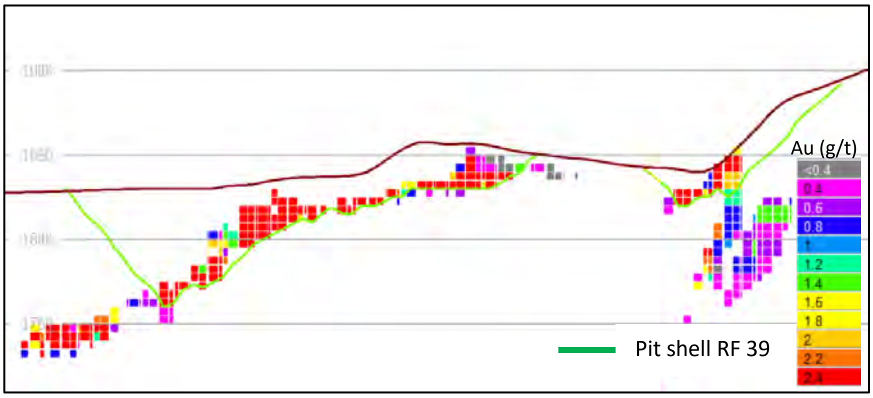


Figure 15-4: North pit shell – Section AA'

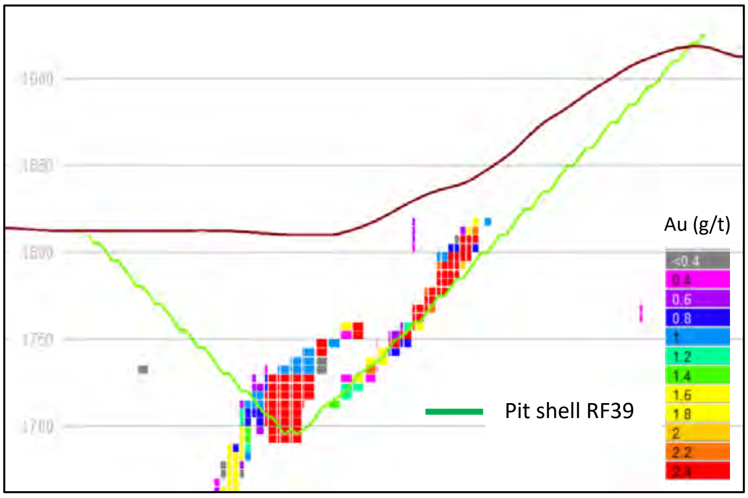


Figure 15-5: Central pit shell – Section BB'

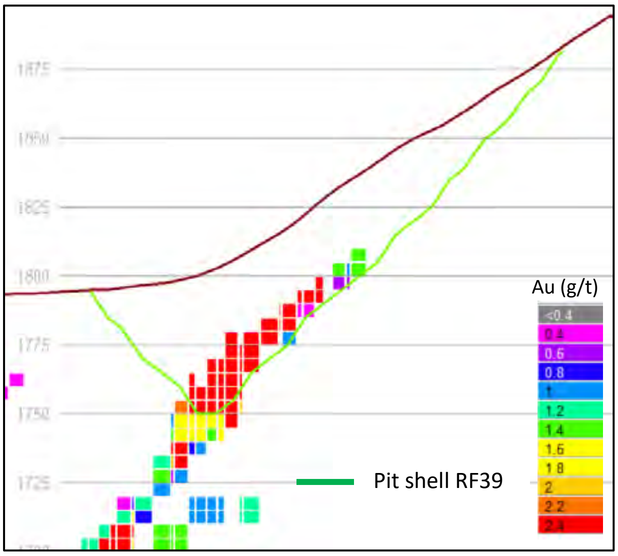


Figure 15-6: South pit shell – Section CC'

15.4.3 Pit Mine Schedule

Pit mine schedule was made from pit shells optimization, the following assumptions were considered:

- A mining recovery of 95% and waste increment of 20% were applied to the pit material to simulate pit design. Additionally, the grades were penalized by a factor of 3.5%.
- 3.6 Mt of material will be mined from the pit in year 0, and from year 1 to year 3, the pit will be mined at a rate of 2.4 Mt.
- In year 0 (pre-production), approximately 500 kt of ore will be stockpiled, while waste will be directed predominantly to the TSF embankment.
- Cover material is not considered appropriate for TSF embankment construction.

Table 15-10 shows the open pit mine schedule for Hualilan Project.

Table 15-10: Hualilan Scheduled Open Pit Mine Physicals

Open Pit	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Total
PMI (kt)	488	351	148	300	24				1,311
Au (g/t)	3.88	3.54	2.57	2.89	3.65				3.41
Ag (g/t)	22.87	19.40	24.85	23.68	20.59				22.31
Pb (%)	0.29	0.18	0.22	0.19	0.21				0.23
Zn (%)	1.30	2.78	1.05	0.69	0.84				1.52
AuEq (g/t)	4.42	4.39	3.10	3.27	4.03				3.99
Waste	3,106	2,048	2,250	2,099	50				9,552
Total mined	3,594	2,398	2,398	2,398	74				10,863

15.5 Underground

15.5.1 Introduction

Mining Plus finalised an underground mine design and schedule for Hualilan in October 2023. This was conducted to a scoping study level of detail. The purpose of this work was to create a mine design and schedule that was both cost effective and able to mine approximately 1 Mt per annum of ore as well as ~100 koz of gold per annum, on a consistent basis.

15.5.1.1 *Chosen Underground Mining Method*

The mining method chosen for the underground mining of the Hualian project was longhole open stoping (LHOS) with paste fill. This was determined by Mining Plus to be suitable based upon the available geotechnical information, the geology of the orebody and similar styled underground mining projects around the world.

A bottom-up retreat schedule was created using a maximum four level lift system. The number of levels in each lift was designed by MP and depended upon geometry and location of the orebody. One to four level systems have been designed for all three underground mines. Sub level open stoping with paste fill gives a level of flexibility to the mining sequence.

The mining method is shown for a three-level system in Figure 15-7.

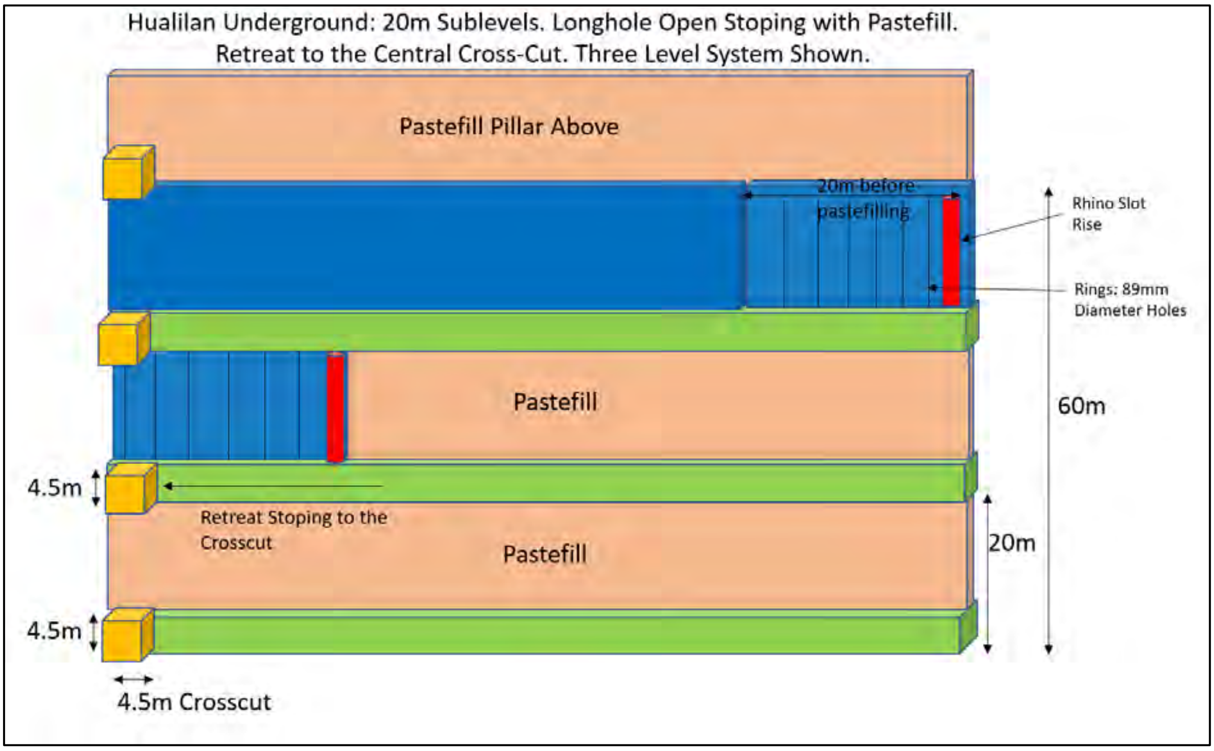


Figure 15-7: Mining Method (Three Level Example)

A stope dilution of 10% and stope recovery of 95% were estimated for this method and included as part of the scheduling phase.

15.5.1.2 ***Redesign of Previous Work***

Initially, Mining Plus conducted a mine design and schedule utilising four main declines and a transverse open stoping mining system. This was completed in July. This design is shown in Figure 15-8.

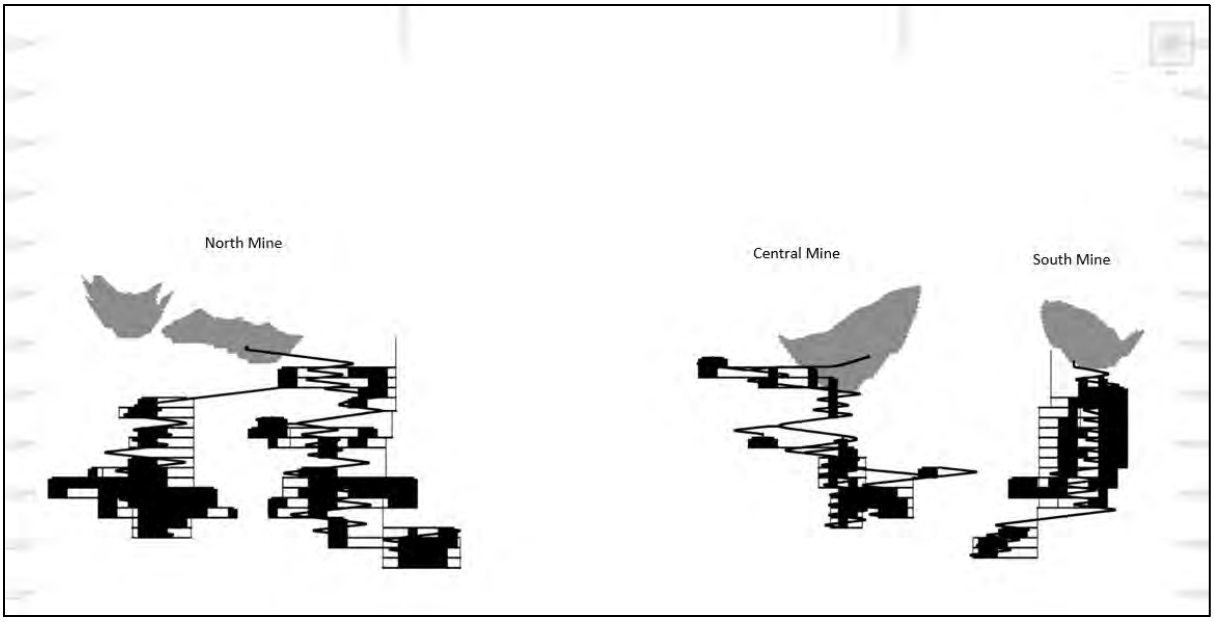


Figure 15-8: July Mine Design. Long section View of North, Central and South Mines

However, upon review, while the mining schedule was able to achieve the required production targets (1 Mtpa ore and ~100 koz pa), the transverse system requires more development metres than a more traditional longitudinal retreat mining system. This is shown in Figure 15-9.

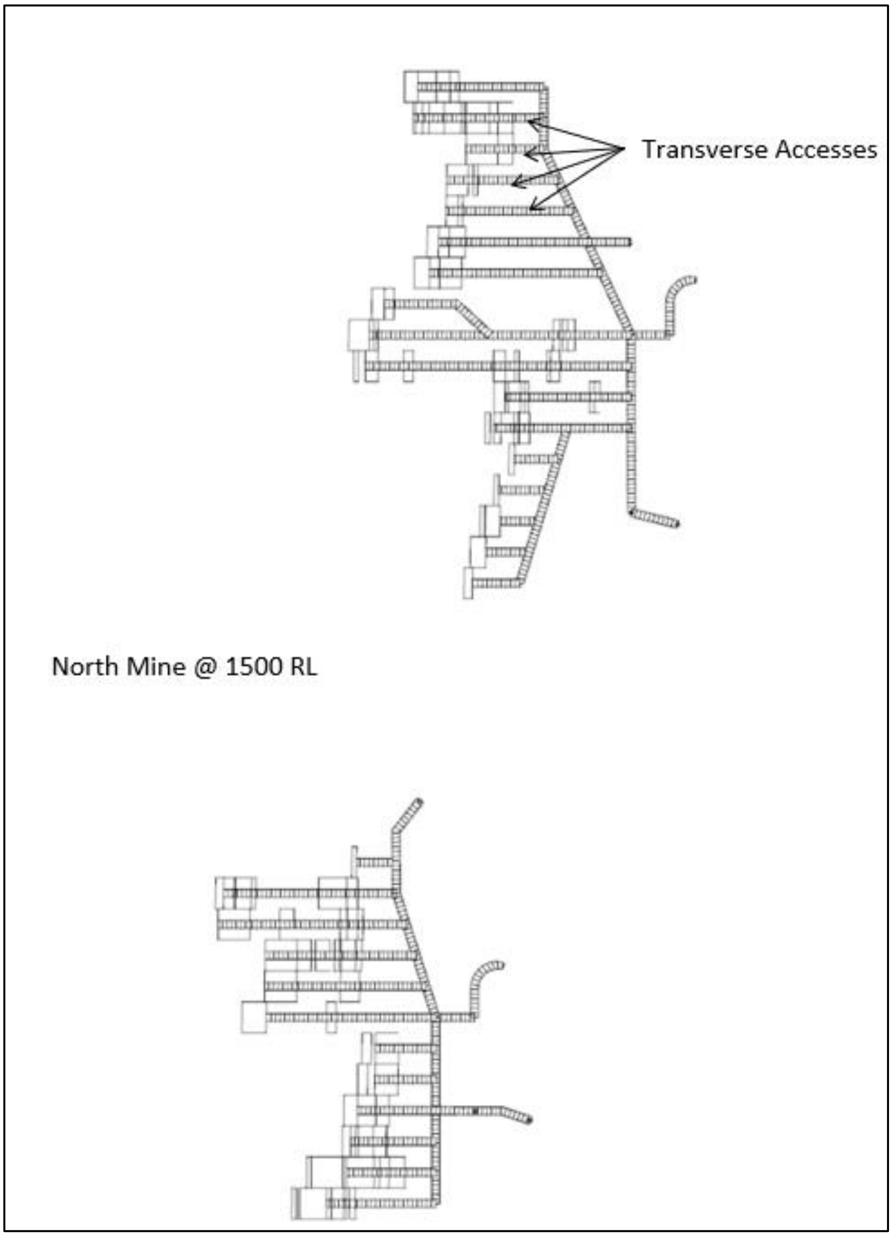


Figure 15-9: July Mine Design. Plan View of North Mine @ 1500RL

Hence, MP was requested to redesign the mine using a longitudinal retreat system, to see if the production guidance could still be met. This updated mine design reduced the number of primary declines to three (from four), enabled a longitudinal retreat system and confirmed that an open stoping system using paste fill be adopted, as the base case mining underground mining method for the Underground Mining Scoping Study. Additionally, all underground portals were removed from the open pit area to decouple the schedule and allow for earlier underground mining (if wanted). This was a critical factor in the trade off studies that negatively affected the NPV of the combined open pit & underground option; UG mining was not able to commence prior to year 7 and created a material impact to NPV. The resultant mine design is shown in Figure 15-10.

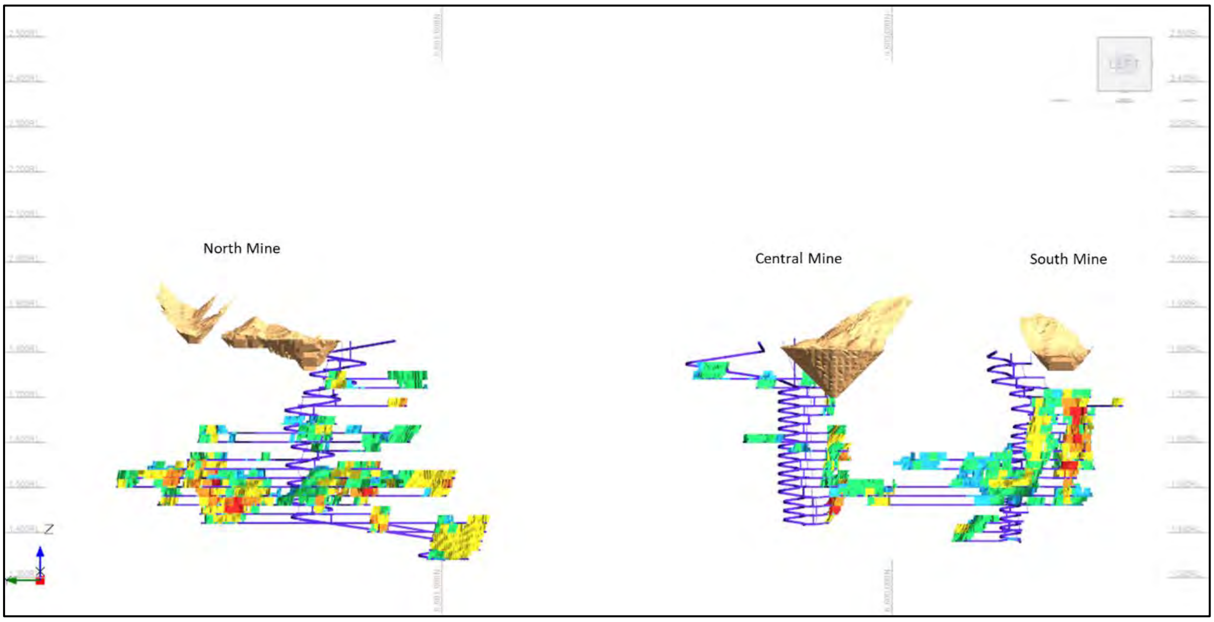


Figure 15-10: Updated Mine Design. Long section View of North, Central and South Mines

15.5.2 Mineable Shape Optimiser

The original block model "Hu_BM_MRE_20230310_lith_dipdir_dip_attrib" was provided in Datamine format for use in the open pit and underground study. This block model was revised (made smaller with ore dip and dip direction added) for MSO evaluation to the version "mp_mso2_23DIP".

Five different frameworks were utilised to account for the different orebody orientations. The three original frameworks are shown in Figure 15-11:

1. Magnata and Sanchez - 080° strike.
2. Between Magnata and Sanchez strike is 030° (and just north of Sanchez).
3. South of Magnata strike is 010°
4. North-South strike (YZ)
5. East-West strike (XZ)

Strike increments were limited to 5 m to accurately account for tonnages, and then combined manually into 20 m strike lengths in Deswik CAD for simplifying the scheduling.

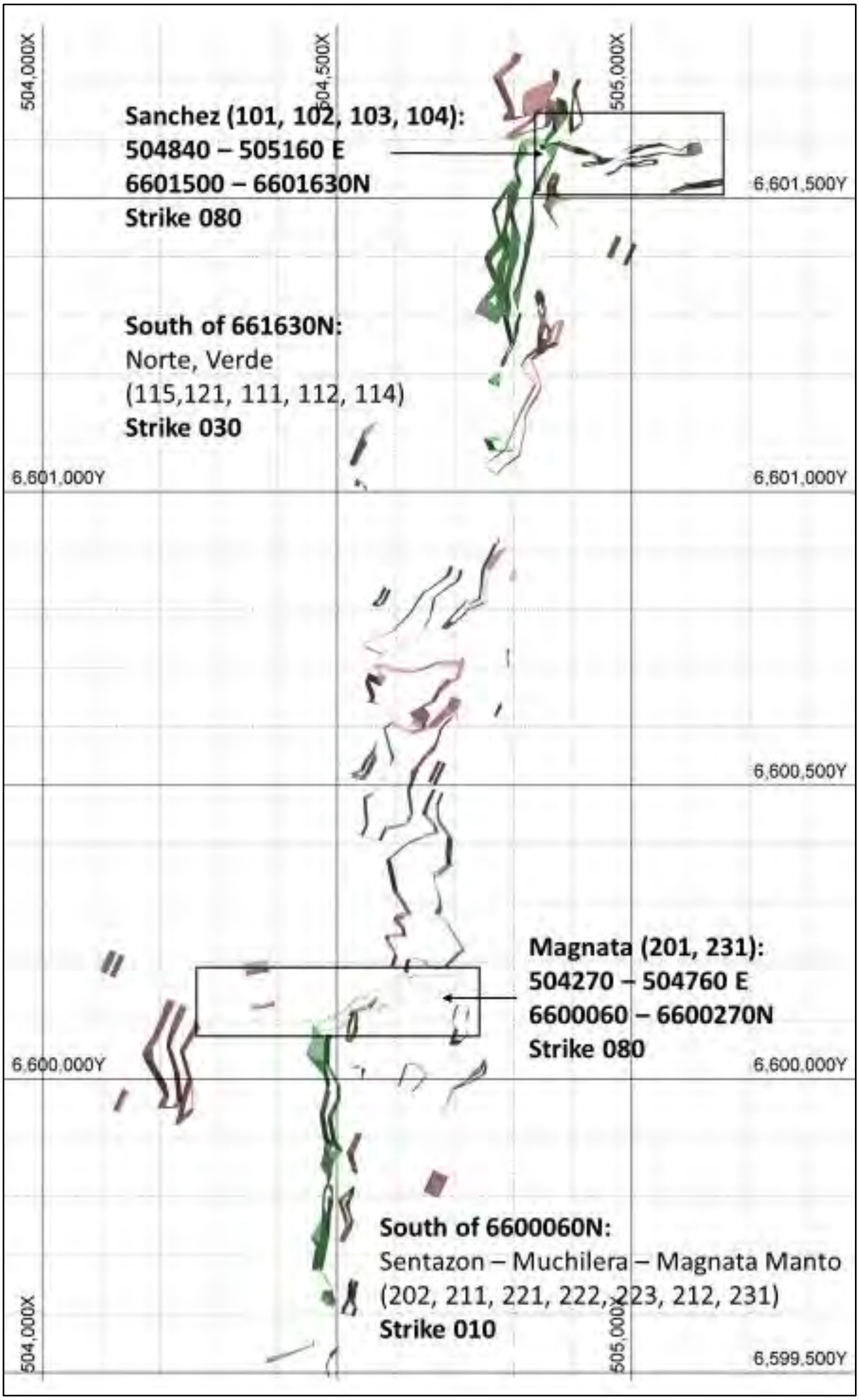


Figure 15-11: Stope Strikes by Domain

A summary of the MSO settings used for Hualilan project are summarised in Table 15-11.

Table 15-11: MSO Settings

MSO Settings	Value	Unit
Block Model	mp_mso2_23DIP.dm	
Layout framework Rotation	Various: 0, 10, 30, 80, 90	deg
undiluted cut-off grade	2.37	g/t AuEq
Default rock density	2.6	t/m^3
Resource categories	2 and 3	
Shape Settings	**Value**	**Unit**
Length	5	m
Level Interval	20	m
Minimum stope strike angle	-45	deg
Maximum stope strike angle	45	deg
Minimum Pillar Between Parallel Stopes	0.001	m
Minimum Mining Width	2.5	m
Maximum Mining Width	20	m

15.5.3 Underground Mine Design

Stope shapes were generated from the surface, 1780 RL, down to level 240, 1340 RL, using a 2.37g/t AuEq cut-off grade. Due to the long strike of the mineralisation of 2.5 km, the areas were broken down into three mine areas: North, Central and South.

A 30 m crown pillar was designed underneath each of the optimised shells of the open pits to the top level of the underground development. The stopes will be mined using a top-down sub level approach with bottom up longhole open stopes with pate fill within the sub levels with paste fill. Initially the number of declines was limited to two. One for the Northern Mine, and a single decline for the Central and South Mines. However, with the large strike lengths needed for the Central and Southern Mine Single Decline option, this option was discarded in favour of a Central and Southern Decline option.

Figure 15-12 shows a plan view of the final mine design by area.

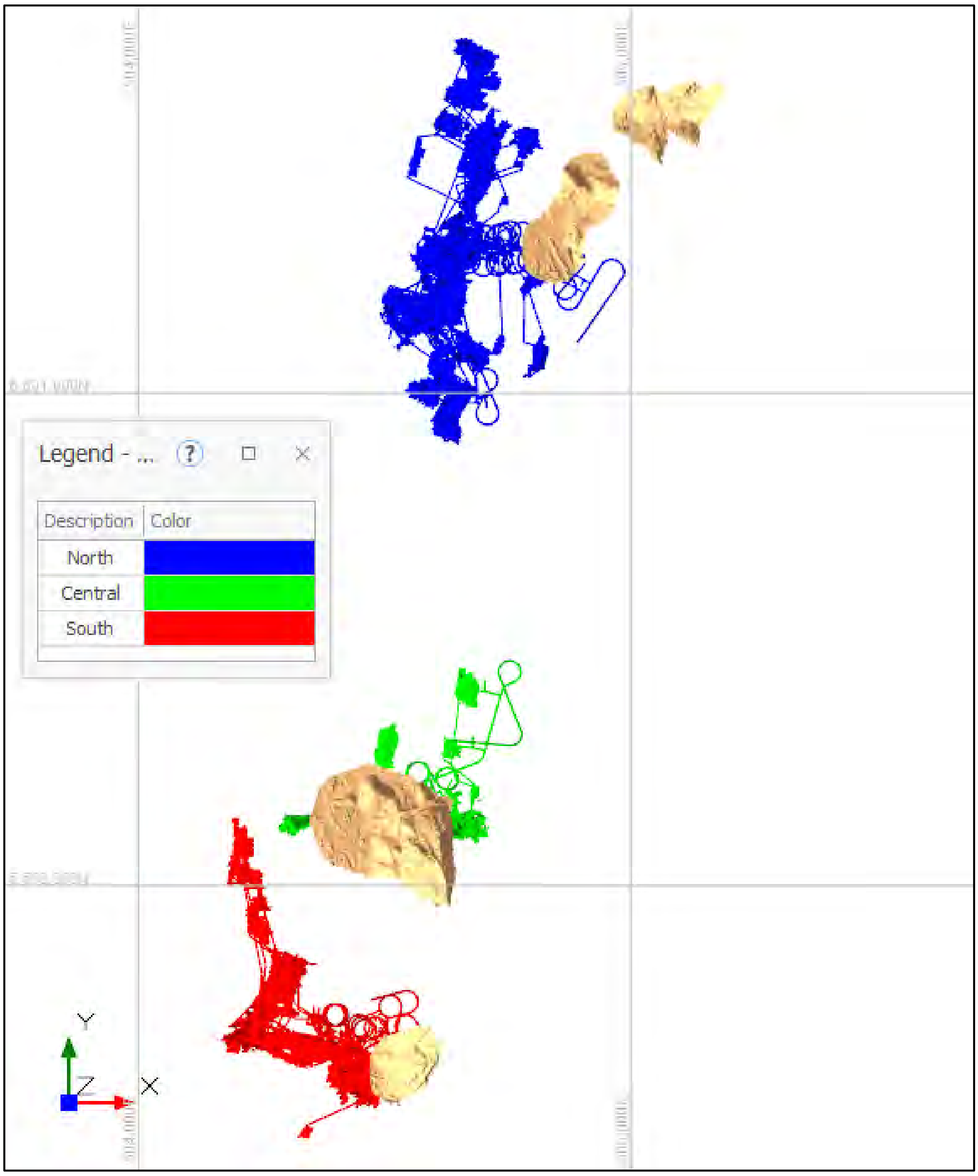


Figure 15-12: Mine Design by Mine Area at Hualilan

15.5.3.1 ***Development Design***

All the underground mine accesses to the underground workings are done via portals located at the surface. The primary declines were designed to suit the level spacing and orebody geometry with a 1:7 gradient. Each mine operates independently, with the declines used for all ore and waste haulage and personnel transport to the surface. The dimensions used for the capital and operating development is shown in Table 15-12.

Table 15-12: Mine Design Attributes, Profiles, Sizes and Shapes

Design Description	ACTTYPE	Profile	COLOUR	CAPOP	EXTYPE	Width	Height	Shape	Arch Radius
Decline	**Ramp**	A	149	CAP	LDEV	5.5	5.5	Arch	1.8

Crosscut 1st 25m	**XCUT**	A	10	CAP	LDEV	5.5	5.5	Arch	1.8
Crosscut 2nd 25 m	**XCUT**	B	39	OP	LDEV	4.5	4.5	Square	
Stockpile	**SP**	A	31	CAP	LDEV	5.5	5.5	Arch	1.8
Return Air Drive	**RAD**	B	0	CAP	LDEV	4.5	4.5	Square	
Escapeway Access	**EWD**	B	80	CAP	LDEV	4.5	4.5	Square	
Ore Drive	**OD**	B	95	OP	LDEV	4.5	4.5	Square	
Return Air Rise L/H	**RAR**	X	0	CAP	VDEV	3.5	3.5	Square	-
Return Air Rise R/B	**RAR**	Y	0	CAP	VDEV	3.5		Circle	-
Escapeway Rise	**EWR**	Z	80	CAP	VDEV	1.1		Circle	-
Uphole Stope	**STOPE**		17	OP	PROD				

15.5.3.2 *Level Design*

Each level was designed based on an up-hole, open stoping mining method in which the stoping retreats to a central access. The ore drives were designed at 4.5 mW x 4.5 mH with 20 m level spacing. Figure 15-13 depicts an example of a typical level.

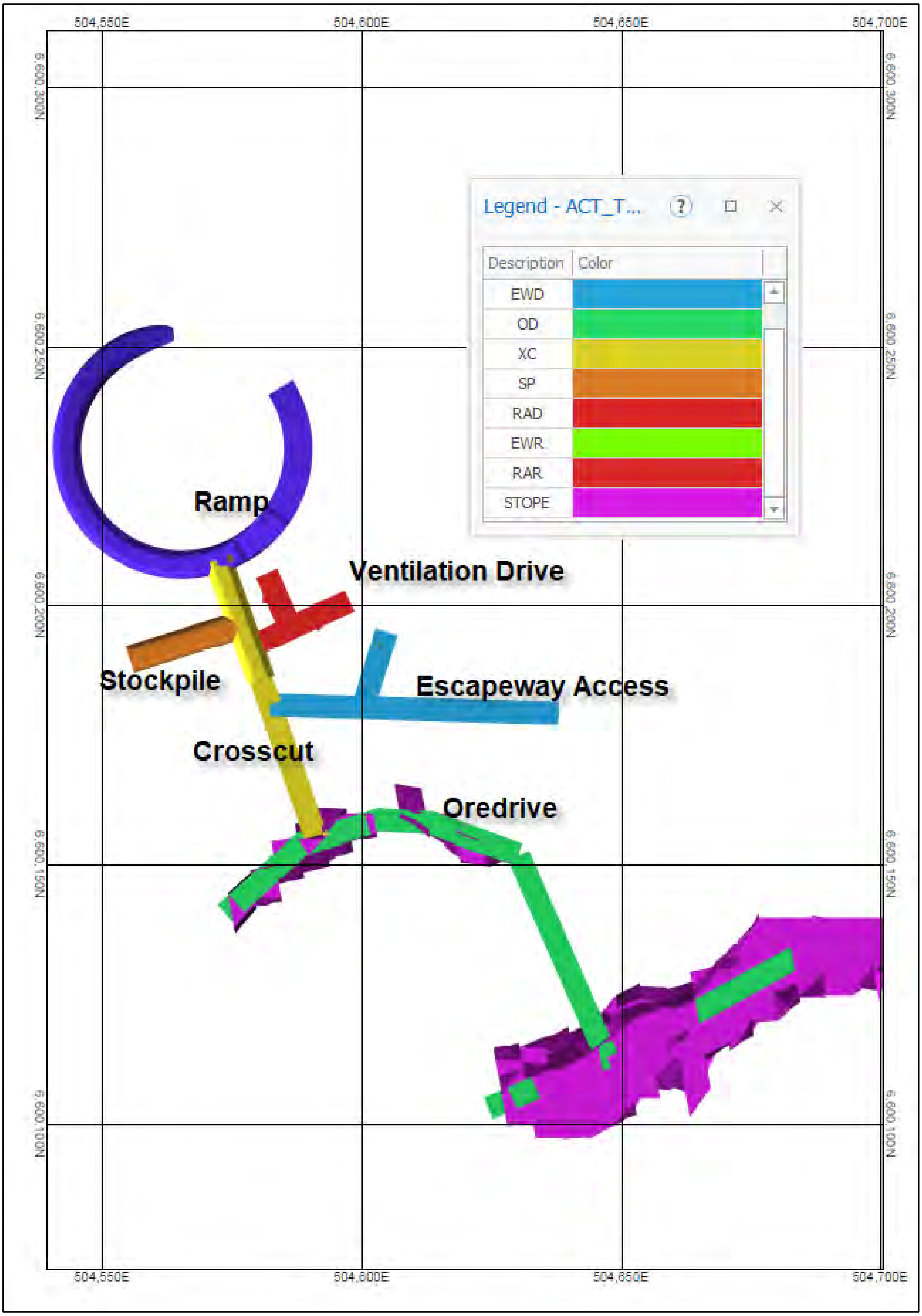


Figure 15-13: Typical Underground Hualilan Level

15.5.3.3 ***Capital Infrastructure Development Design***

All three mines use a similar primary ventilation system that consists of:

- An intake decline from the surface (5.5 mW x 5.5 mH)
- A series of return air raises connecting the levels to the surface (3.5 m x 3.5 m)

The design follows a conventional primary exhaust strategy, with decline fresh air supplying levels ventilated with auxiliary fans, then returning up regulator-controlled exhaust raises to the primary ventilation fans.

A second means of egress from the underground mine has been designed for each mine to enable personnel to exit the mine via a separate access other than the decline, if necessary. To ensure that all personnel can reach a connection drive, ladderways will be installed off the decline via small, sub-vertical rises.

The ladderways will be constructed from galvanised steel in a modular form and will include protective caging and rest platforms. Design, fabrication and installation of the ladderways will be carried out by a contractor that specialises in installation of underground infrastructure.

The following infrastructure development was not designed as it is not required for this level of study. However, they are required, and the physical metres and tonnes associated were factored and accounted for in the scheduling outputs:

- Sumps and pump stations
- Drive Stripping (flitches, fan stripping etc.)
- Decline Stockpiles

15.5.3.4 ***Dewatering***

Graded level development was not considered in the mine design for simplicity, which for this level of study is appropriate i.e. all level development has been designed flat. However, allowances for development physicals were made in the schedule by factoring the capital development. A dewatering design should be conducted in the next phase of study.

15.5.3.5 ***Uneconomic Analysis/Pseudoflow***

A Pseudoflow analysis was completed on the underground design to remove uneconomic areas that sit above the stope cut-off grade. Pseudoflow was used to calculate whether a stope is economic based on the development in addition to the stoping costs required to access that stope. Costs for the analysis were calculated using the provided unit rate costs. Revenue factor and costs used for this analysis are summarised in Table 15-13.

Table 15-13: Revenue factor and costs for Hualilan Pseudoflow Analysis

Costs	Value	Unit
Revenue Factor	1,472.2	US$/oz (AuEq)
Profile A (5.5m x 5.5m arch)	4,000	US$/m
Profile B (4.5m x 4.5m arch)	3,300	US$/m
Profile X (3.5m x 3.5m square)	2,500	US$/m
Profile Y (3.5mD circle)	2,500	US$/m
Profile Z (1.1mD circle)	2,500	US$/m
Stoping	37.62	US$/t
G&A cost	3.91	US$/t
Processing cost	13.11	US$/t

All stopes were linked to development to calculate which stopes were profitable. This requires an output revenue factor of less than one. Results of the Pseudoflow analysis by mine are shown in Figure 15-14, Figure 15-15 and Figure 15-16. All stopes with a value greater than 1 were excluded from the schedule (colours yellow to red).

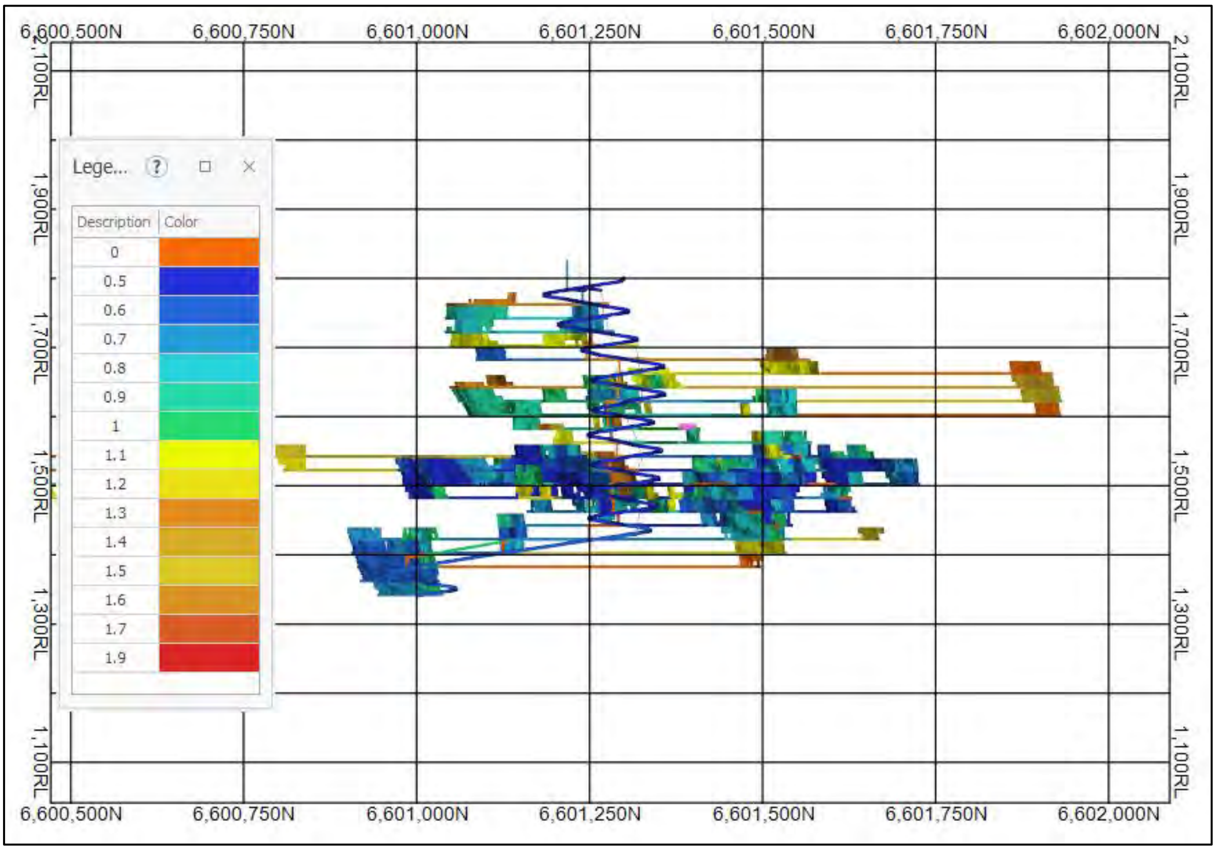


Figure 15-14: Hualilan North Mine Pseudoflow Outputs Showing Revenue Factor

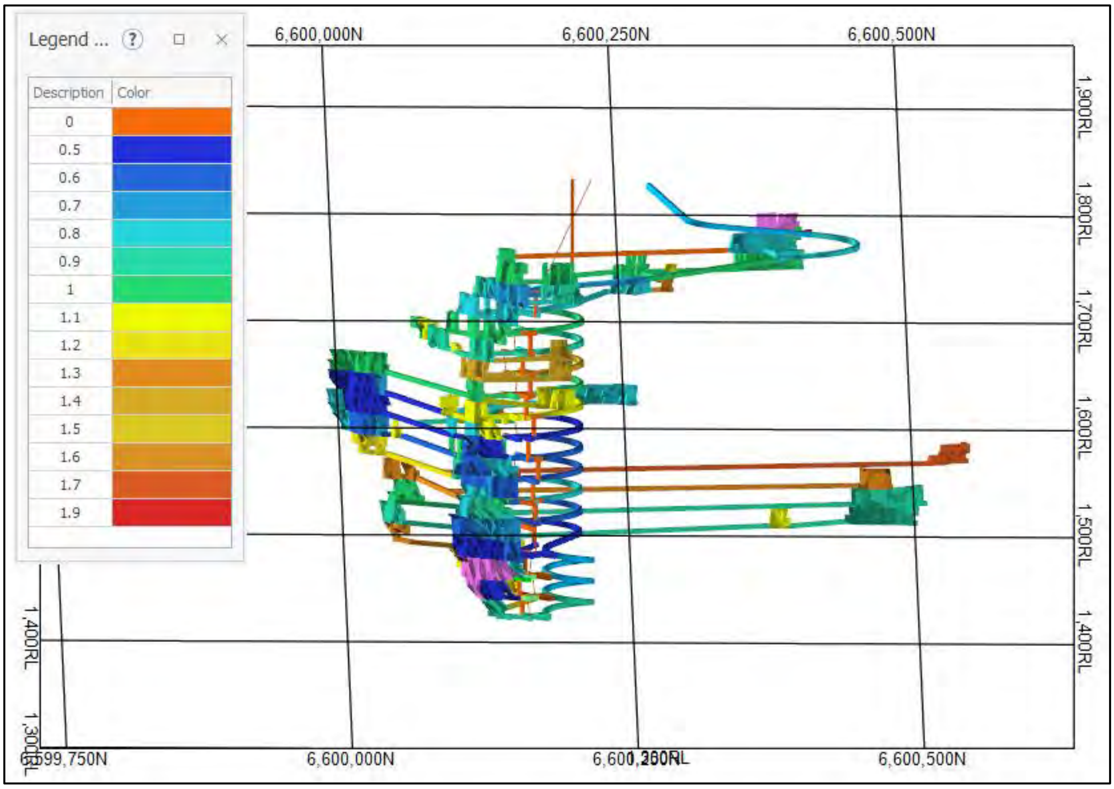


Figure 15-15: Hualilan Central Mine Pseudoflow Outputs Showing Revenue Factor

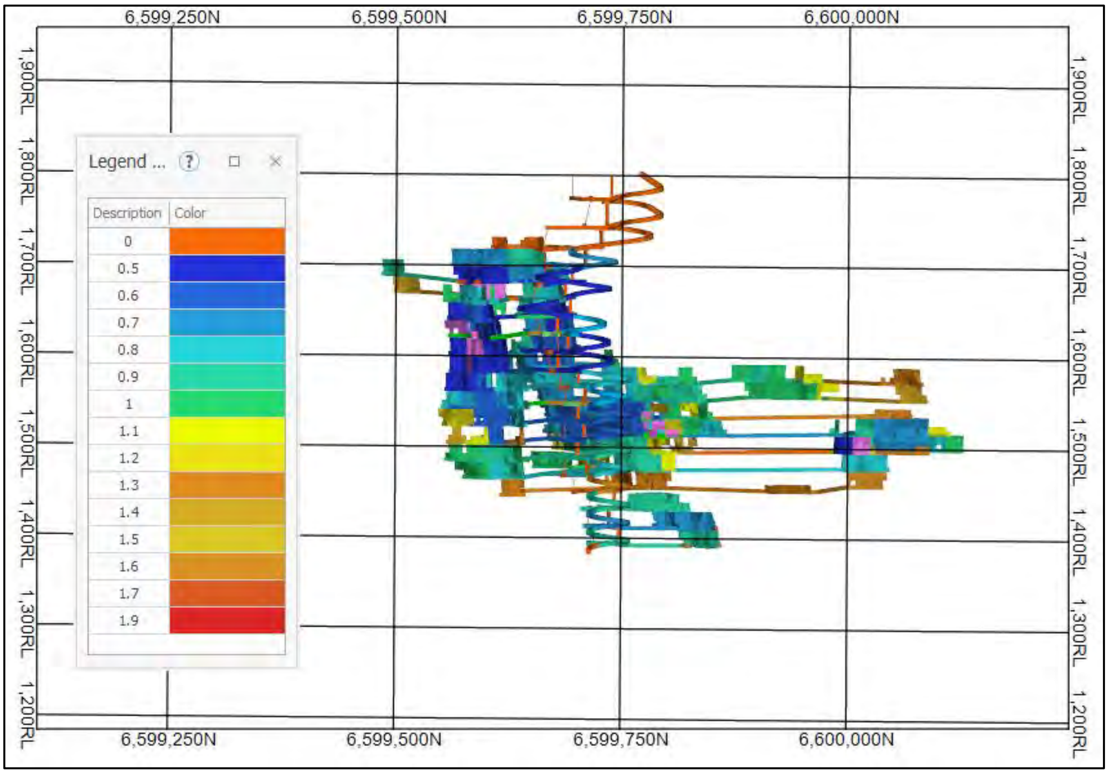


Figure 15-16: Hualilan South Mine Pseudoflow Outputs Showing Revenue Factor

15.5.3.6 *Final Mine Design*

Based on the Pseudoflow analysis, uneconomic areas were removed from the design and a final mine design was created for scheduling as shown in Figure 15-17, Figure 15-18 and Figure 15-19.

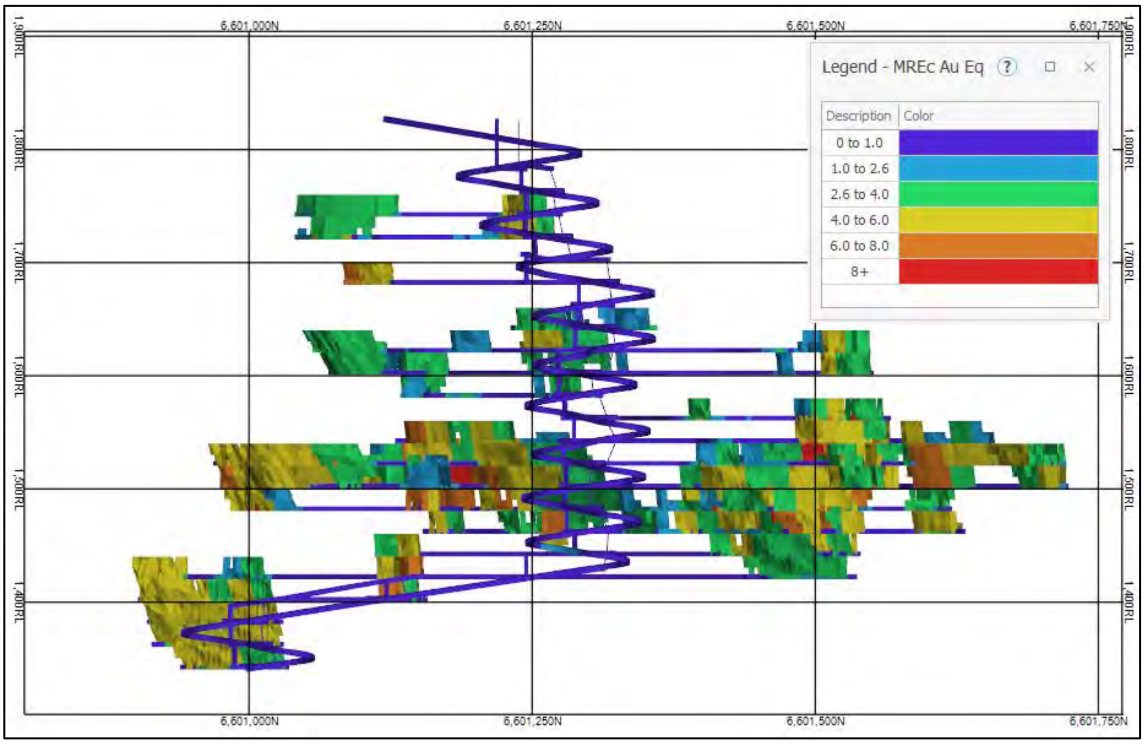


Figure 15-17: Hualilan Final Mine Design – North Mine

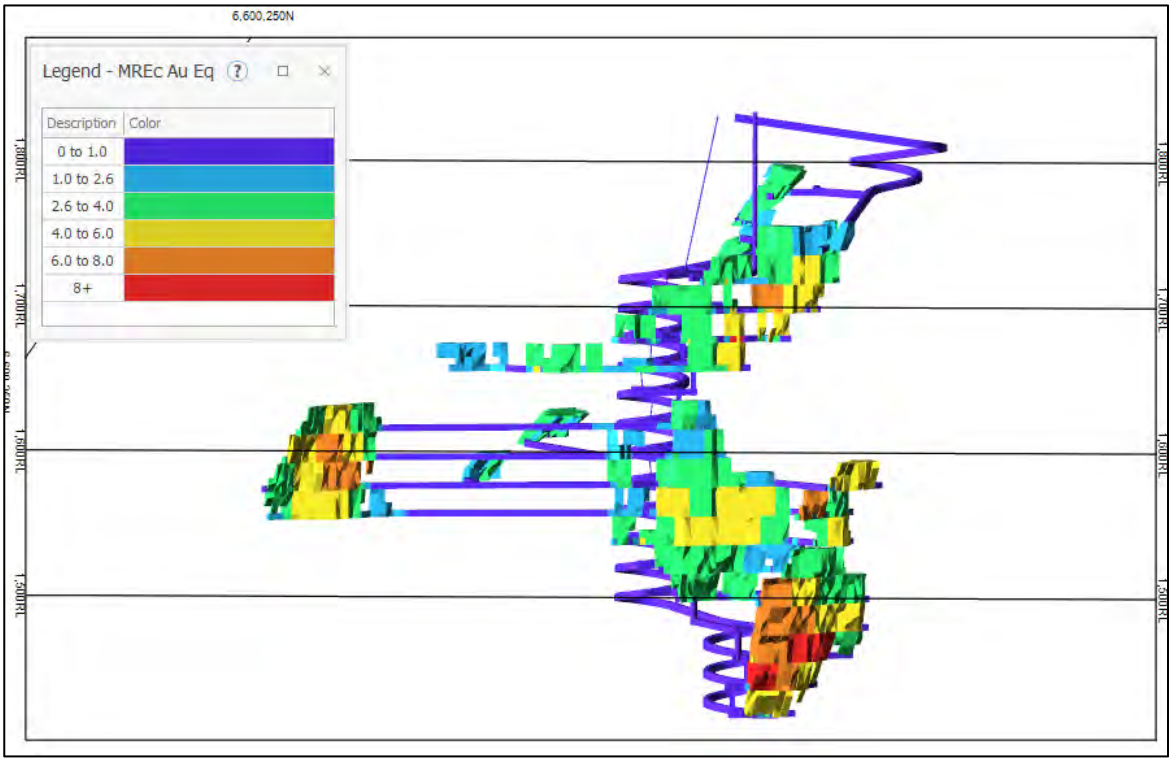


Figure 15-18: Hualilan Final Mine Design – Central Mine

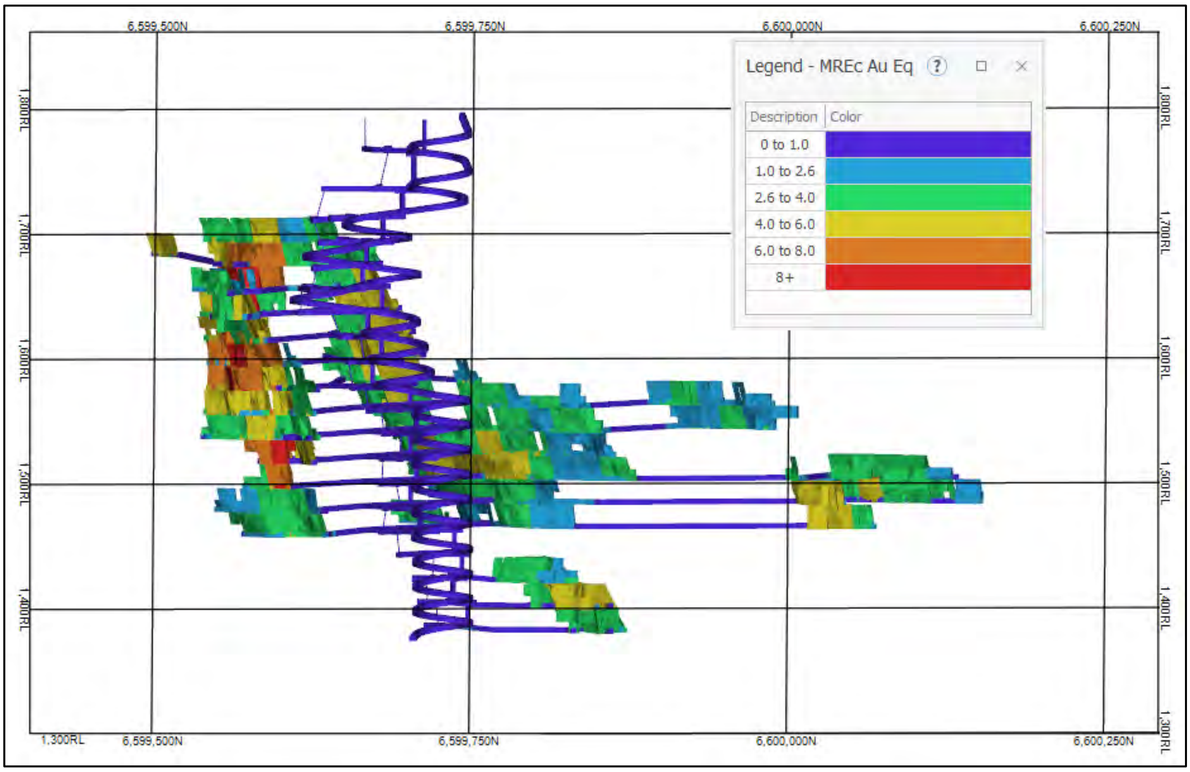


Figure 15-19: Hualilan Final Mine Design – South Mine

15.5.4 Underground Mine Schedule

The purpose of the underground mine schedule work was to create a schedule that was able to mine approximately 1 Mt per annum of ore as well as ~100 koz of gold per annum, on a consistent basis. Mining Plus tried to prioritise higher grade material early in the schedule, where possible.

15.5.4.1 *Production Schedule Rates*

Capital lateral development metres were factored by 1.15 to allow extra development accounting for sumps, stripping and cuddies. Ore was considered anything above 1.5 g/t AuEq. A mining recovery factor of 95% was applied to longhole stoping to account for losses attributed to mining adjacent to and below paste fill. A summary of the schedule mining rates and stoping factors used in the study is shown in Table 15-14.

Table 15-14: Hualilan Project Schedule Rates and Stoping Factors

Schedule rates	Value	Unit
Ramp Development	180	m/month
Other Lateral Development	100-120	m/month
Boxhole Raise (Rhino)	10	m/day
Escapeway (Rhino)	150	m/month
Ventilation Raise	150	m/month
Long Hole Drilling	300	m/day
Stoping rates and factors		
Stope Long Hole Drilling	300	m/day
Recovery factor for Stoping	95	%
Dilution factor for Stoping	10	%
Slot Drive	5	m/stope
Backfill Development	8	m/stope
Stope Bogging	1,500	MrT/d

Tonne Kilometre (TKM) calculations were performed based on haulage distance calculated from each material haulage level to portal level for each mine with additional distance added from portal to the ROM and Waste Dump being 1,000 m.

In addition to slot drives and backfill drives development, the following delays were added to the production schedule to account for backfill activities: Backfill preparation – 2 days; Backfill curing delay – 14 days.

*15.5.4.2 **Further Considerations***

The following additional considerations were applied to the schedule:

- The North and South mines start 1 year earlier than the Central mine. The north mine is the largest and requires the most time to mine. The South mine has high grades available sooner than the other underground mines.
- A maximum of 2 jumbos were scheduled for the North and South Mines and a single jumbo scheduled for the central mine.
- The underground mines were limited to approximately 1 Mtpa ore. Additionally, they are scheduled to productively co-exist with the open pit schedule and required processing schedule.

*15.5.4.3 **Scheduled Mining Areas***

15.5.4.3.1 North Mine

The areas mined in the North Mine by year are shown from Figure 15-20 to Figure 15-27.

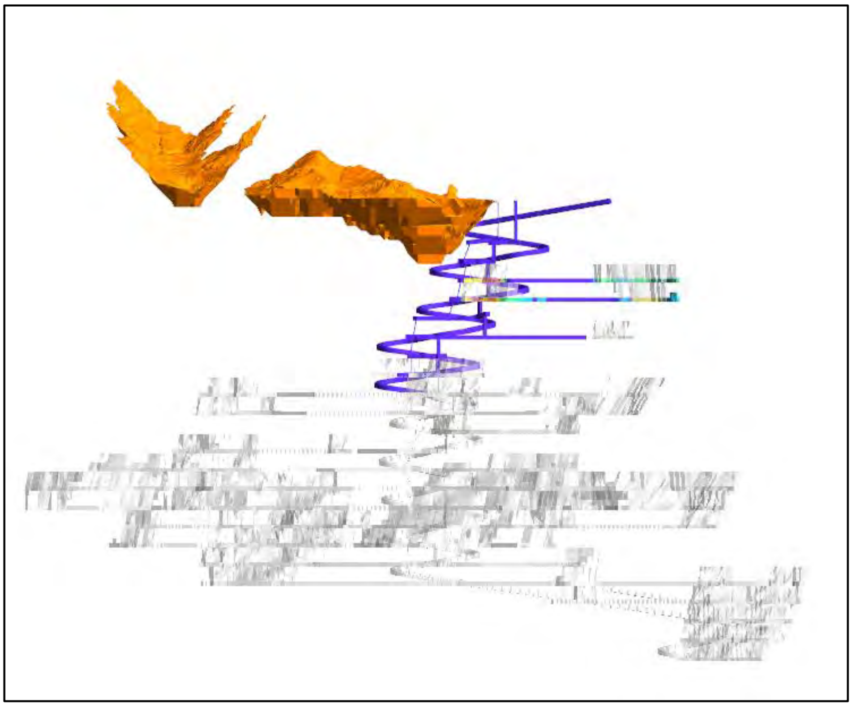


Figure 15-20: North Mine Schedule. Long-section View. Year 0 Mined Areas

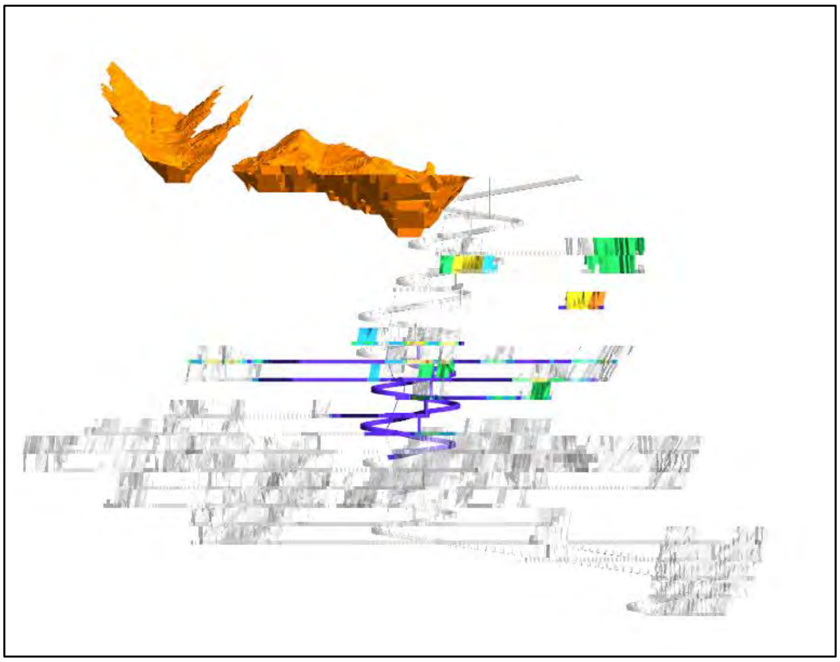


Figure 15-21: North Mine Schedule. Long-section View. Year 1 Mined Areas

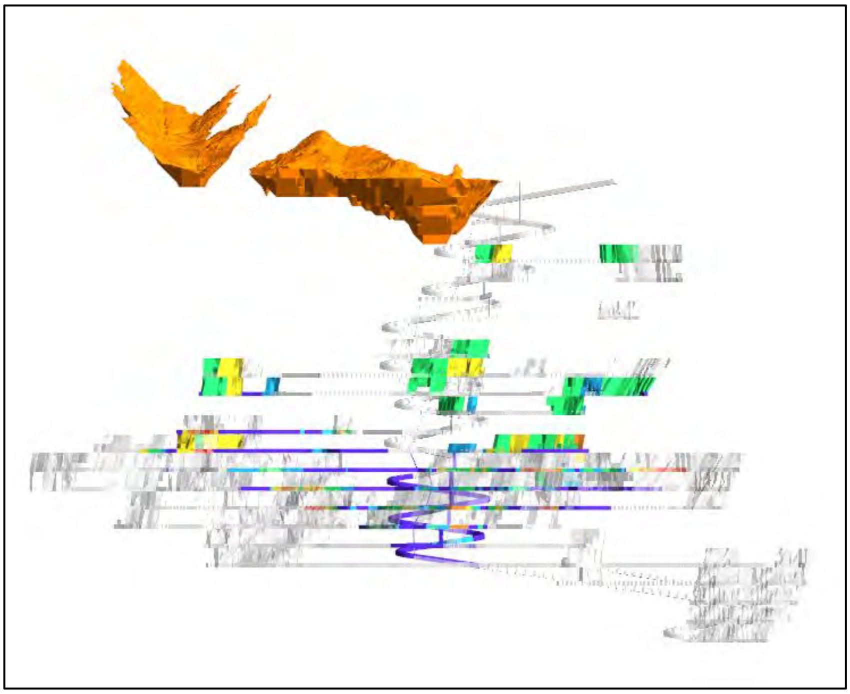


Figure 15-22: North Mine Schedule. Long-section View. Year 2 Mined Areas

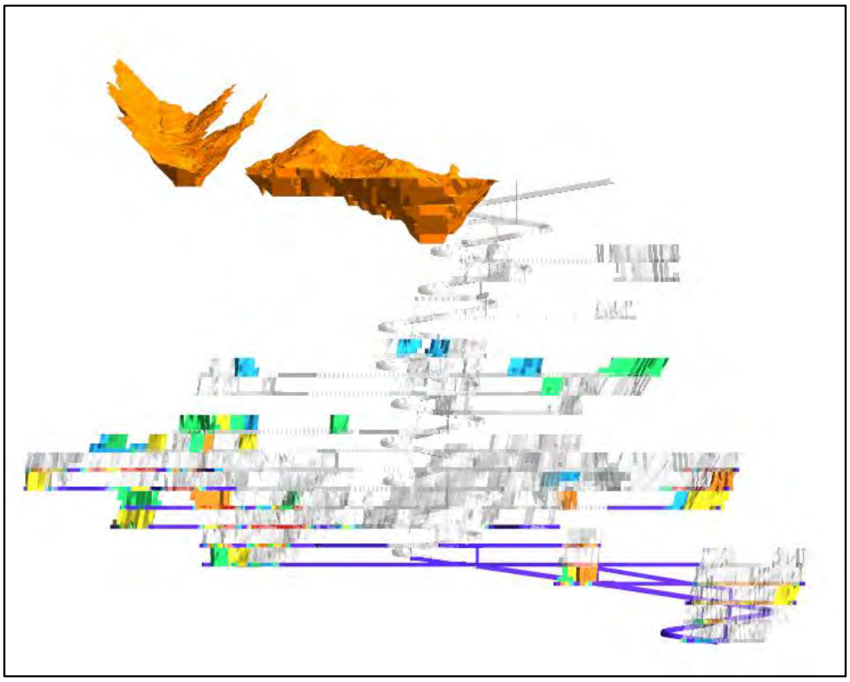


Figure 15-23: North Mine Schedule. Long-section View. Year 3 Mined Areas

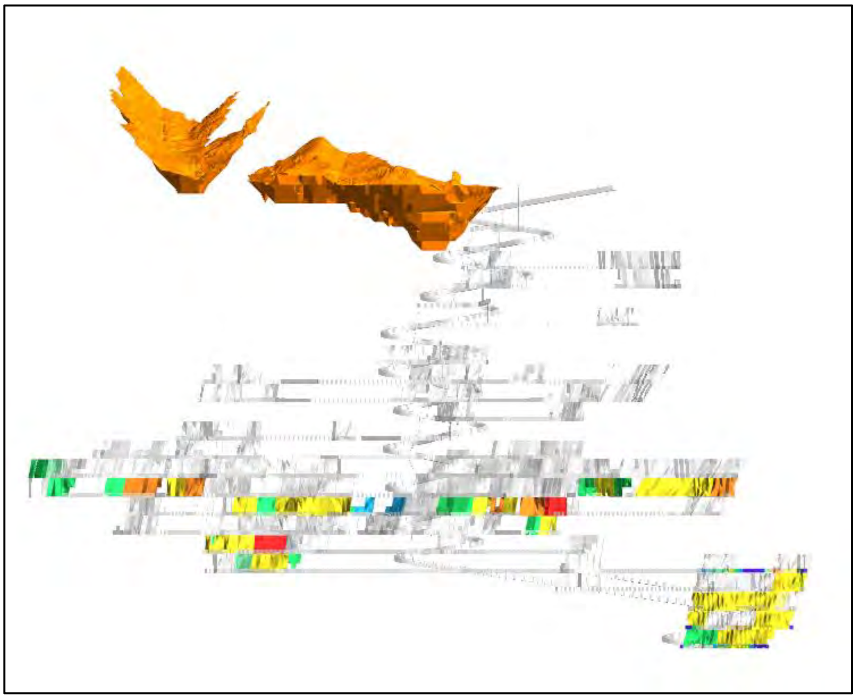


Figure 15-24: North Mine Schedule. Long-section View. Year 4 Mined Areas

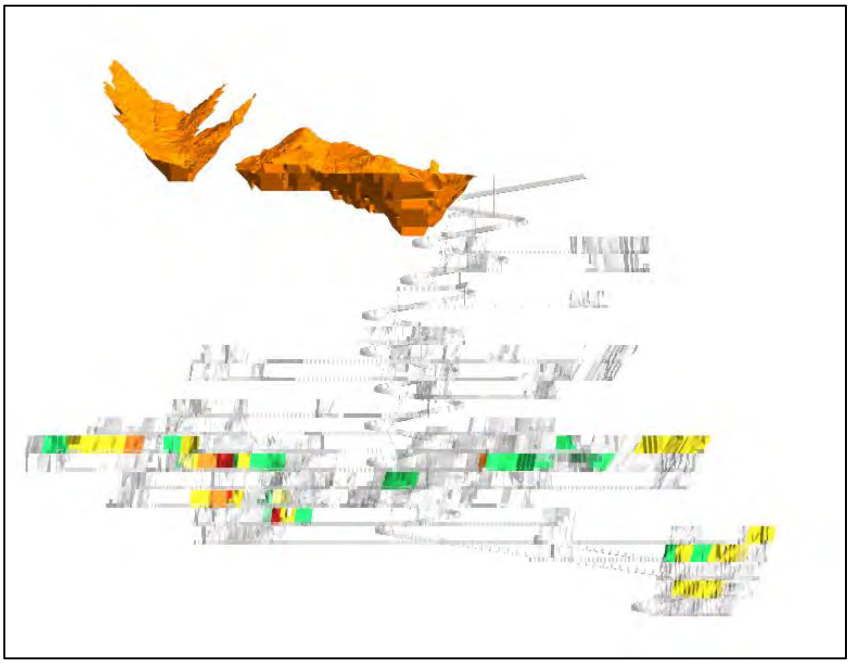


Figure 15-25: North Mine Schedule. Long-section View. Year 5 Mined Areas

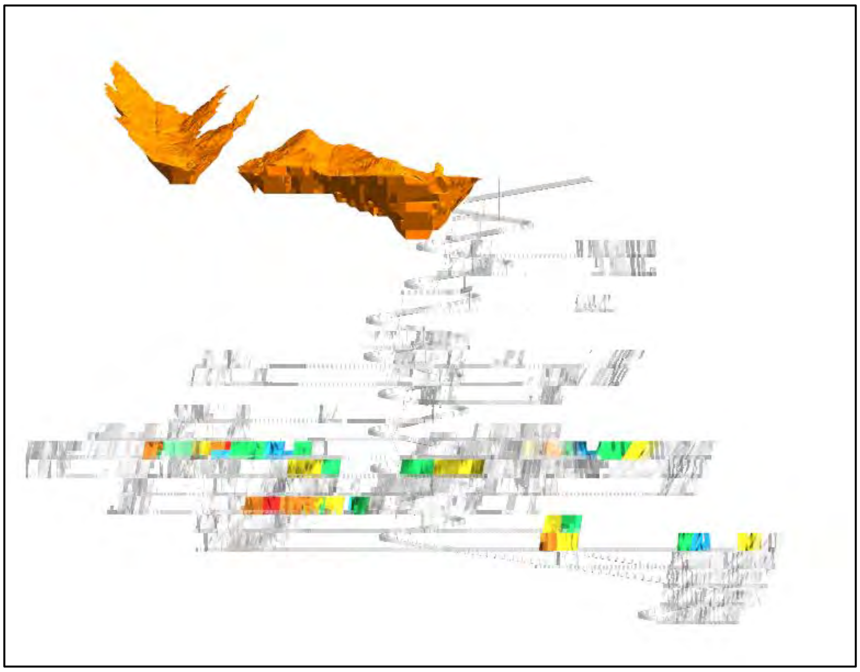


Figure 15-26: North Mine Schedule. Long-section View. Year 6 Mined Areas

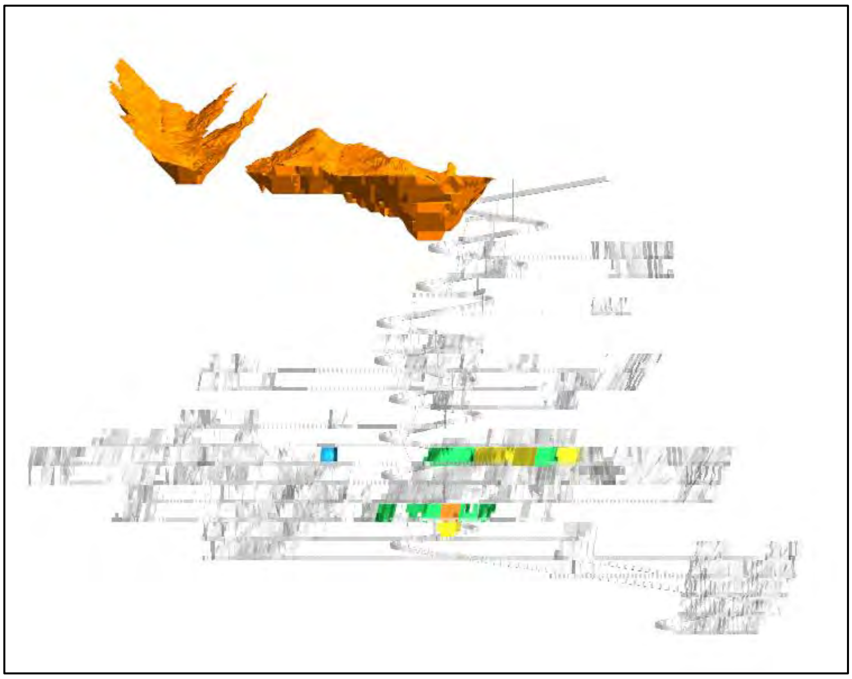


Figure 15-27: North Mine Schedule. Long-section View. Year 7 Mined Areas

15.5.4.3.2 Central Mine

The areas mined in the Central Mine by year are shown from Figure 15-28 to Figure 15-35.

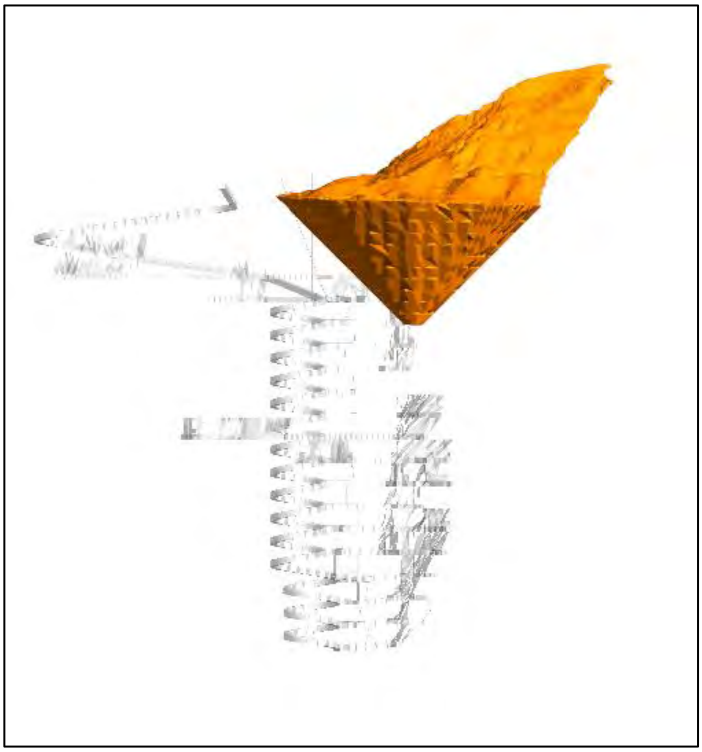


Figure 15-28: Central Mine Schedule. Long-section View. Year 0 Mined Areas

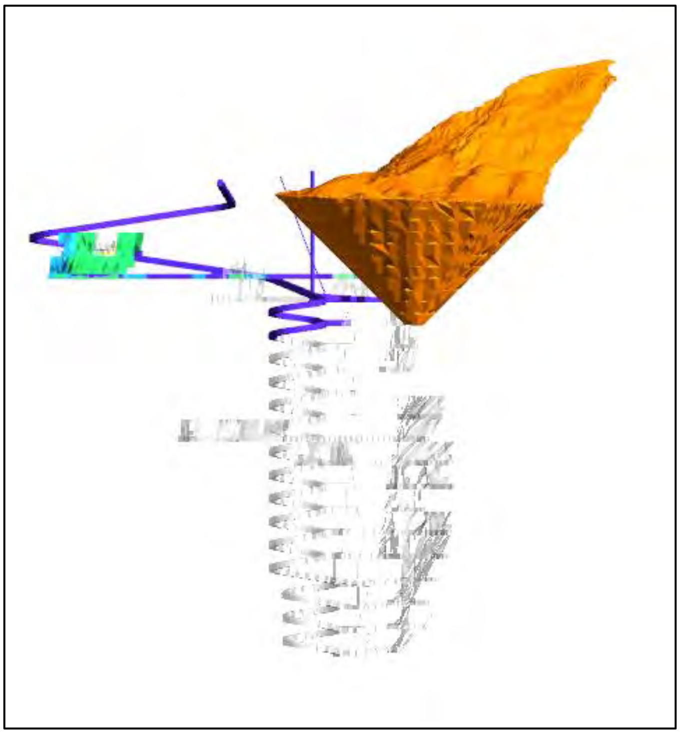


Figure 15-29: Central Mine Schedule. Long-section View. Year 1 Mined Areas

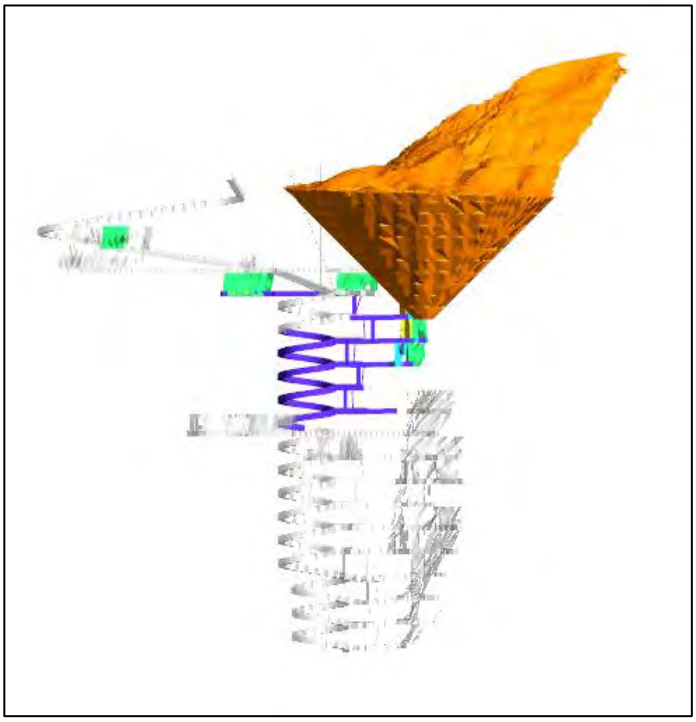


Figure 15-30: Central Mine Schedule. Long-section View. Year 2 Mined Areas

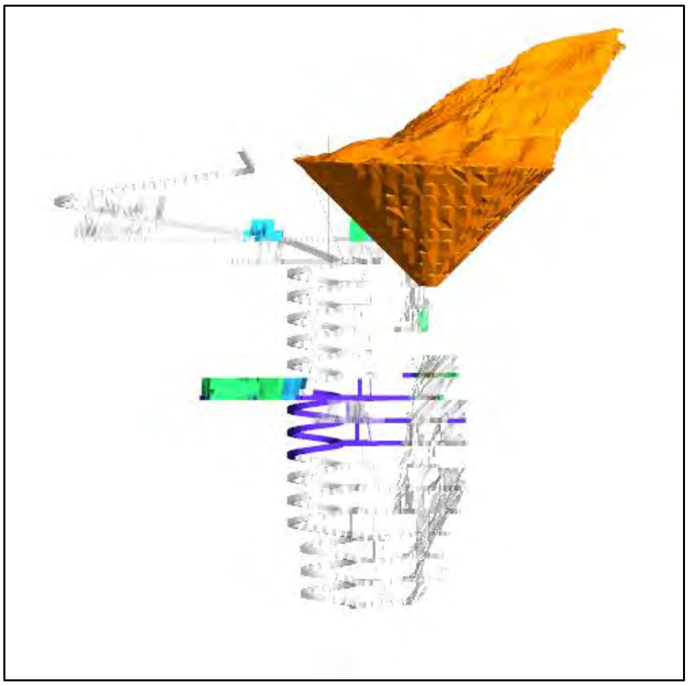


Figure 15-31: Central Mine Schedule. Long-section View. Year 3 Mined Areas

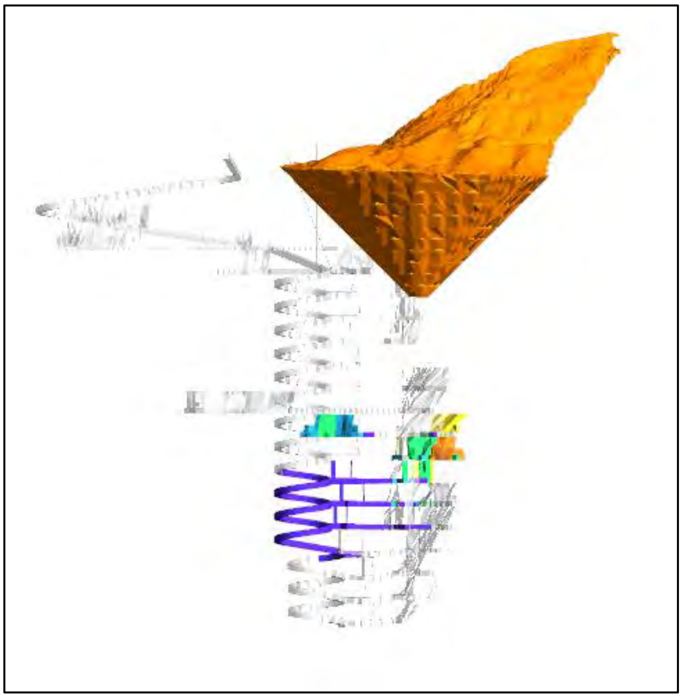


Figure 15-32: Central Mine Schedule. Long-section View. Year 4 Mined Areas

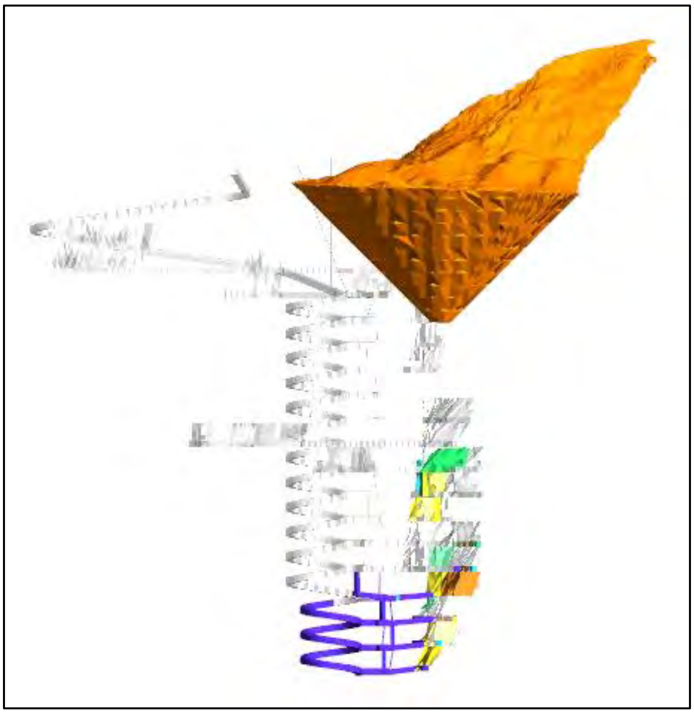


Figure 15-33: Central Mine Schedule. Long-section View. Year 5 Mined Areas

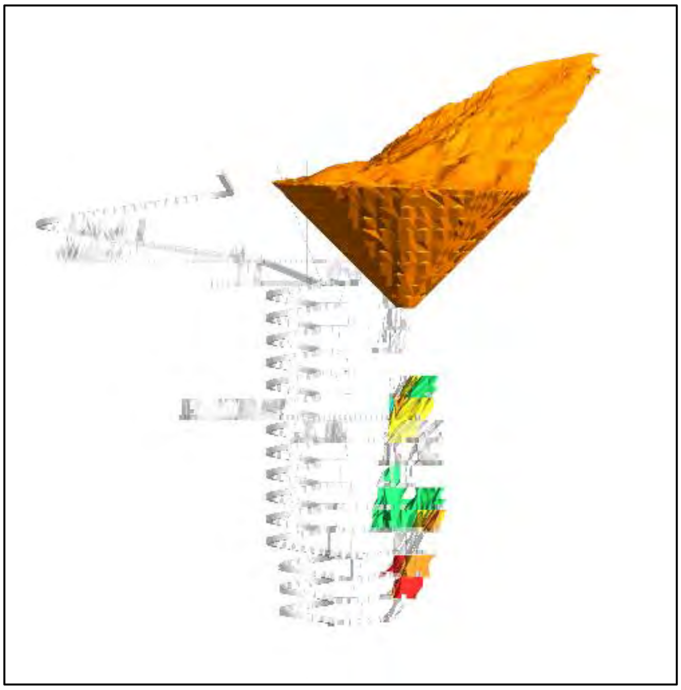


Figure 15-34: Central Mine Schedule. Long-section View. Year 6 Mined Areas

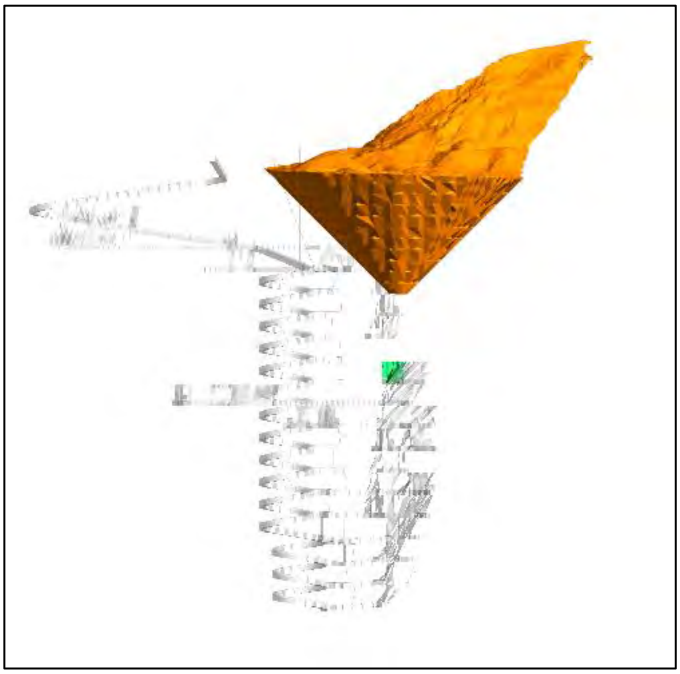


Figure 15-35: Central Mine Schedule. Long-section View. Year 7 Mined Areas

15.5.4.3.3 South Mine

The areas mined in the South Mine by year are shown from Figure 15-36 to Figure 15-43.

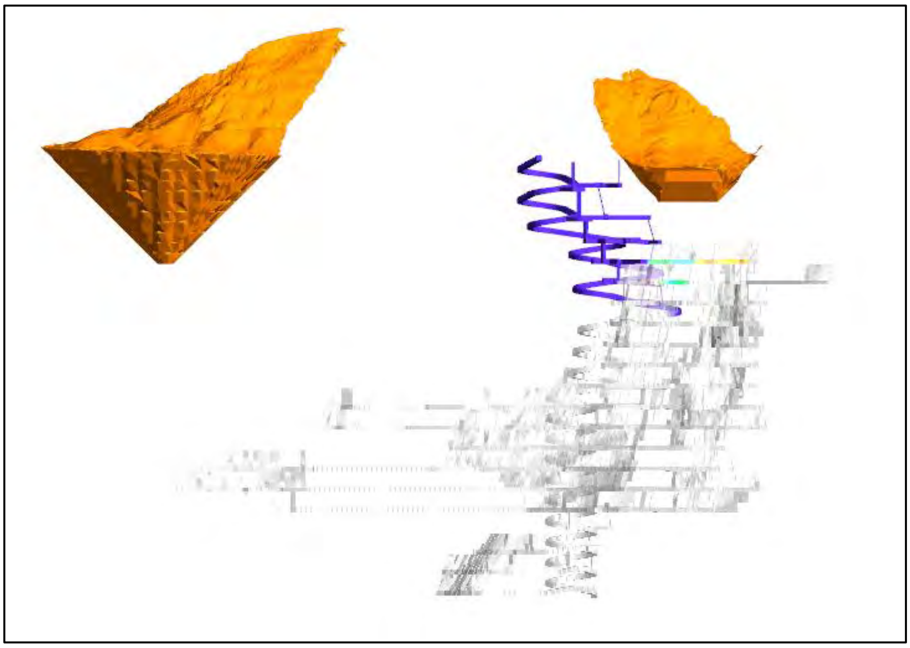


Figure 15-36: South Mine Schedule. Long-section View. Year 0 Mined Areas

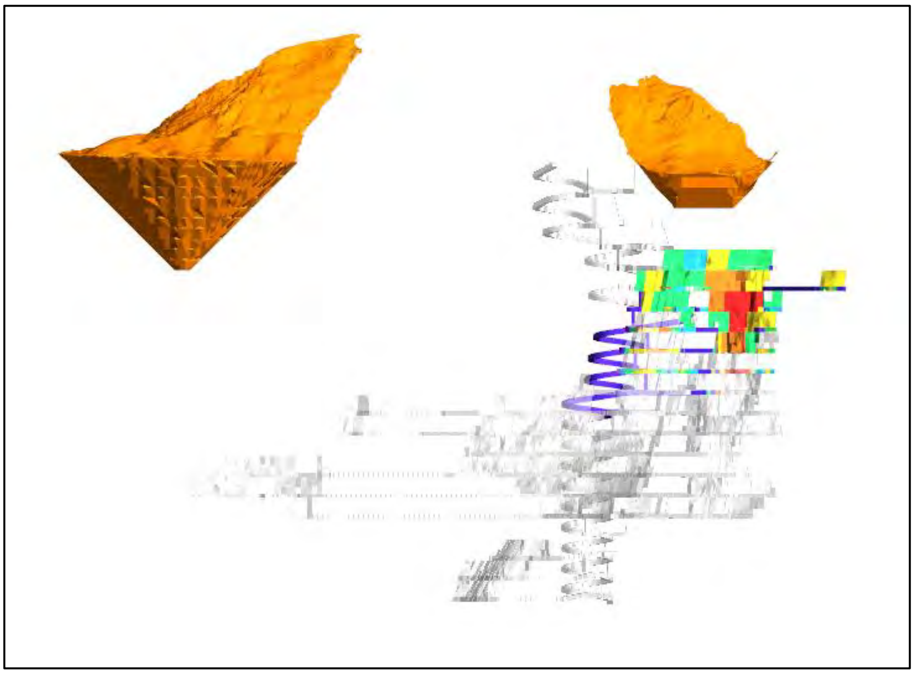


Figure 15-37: South Mine Schedule. Long-section View. Year 1 Mined Areas

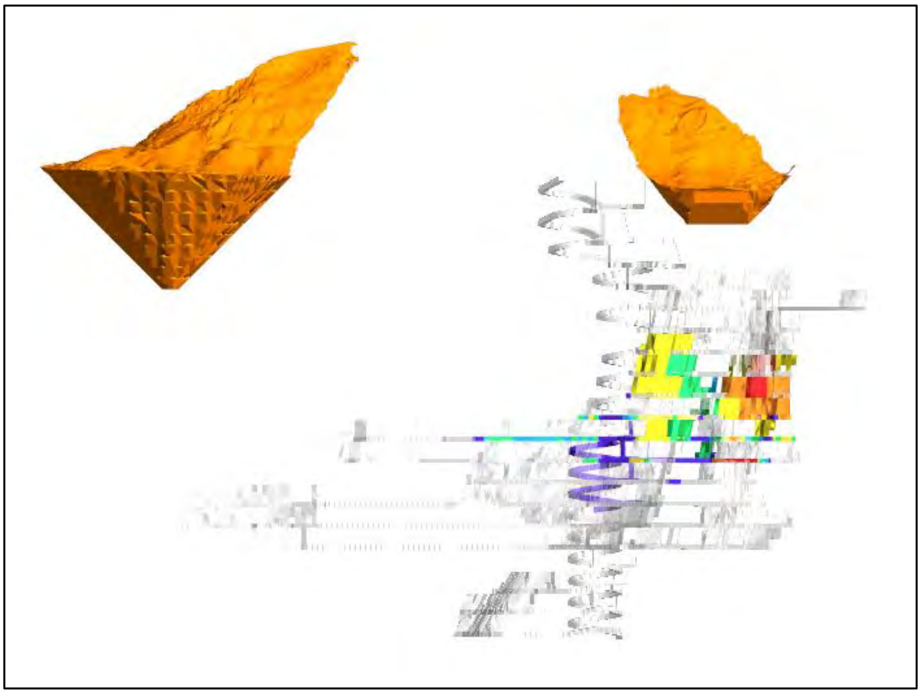


Figure 15-38: South Mine Schedule. Long-section View. Year 2 Mined Areas

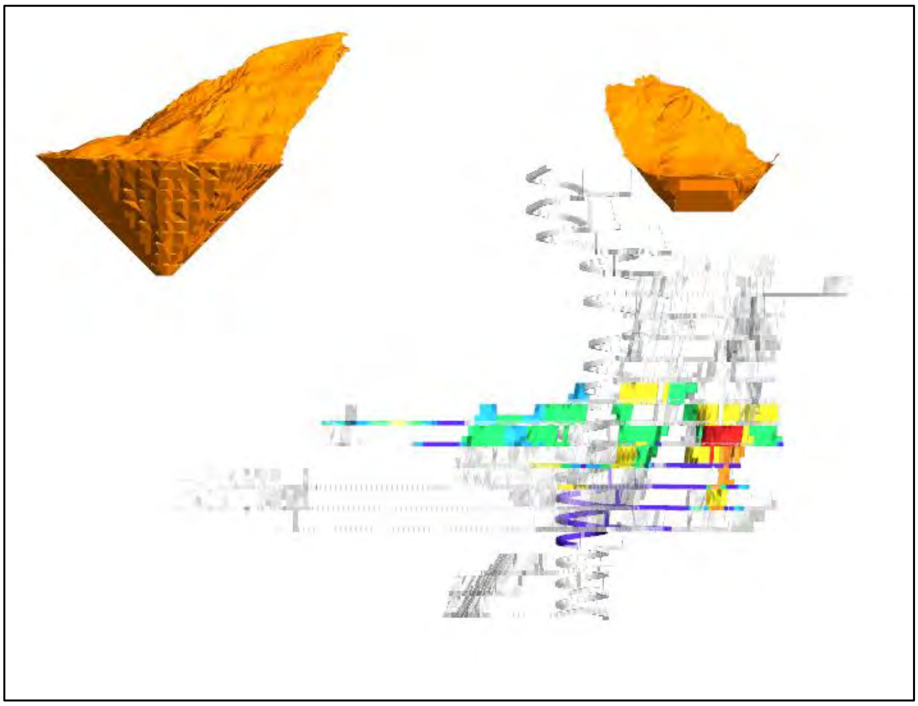


Figure 15-39: South Mine Schedule. Long-section View. Year 3 Mined Areas

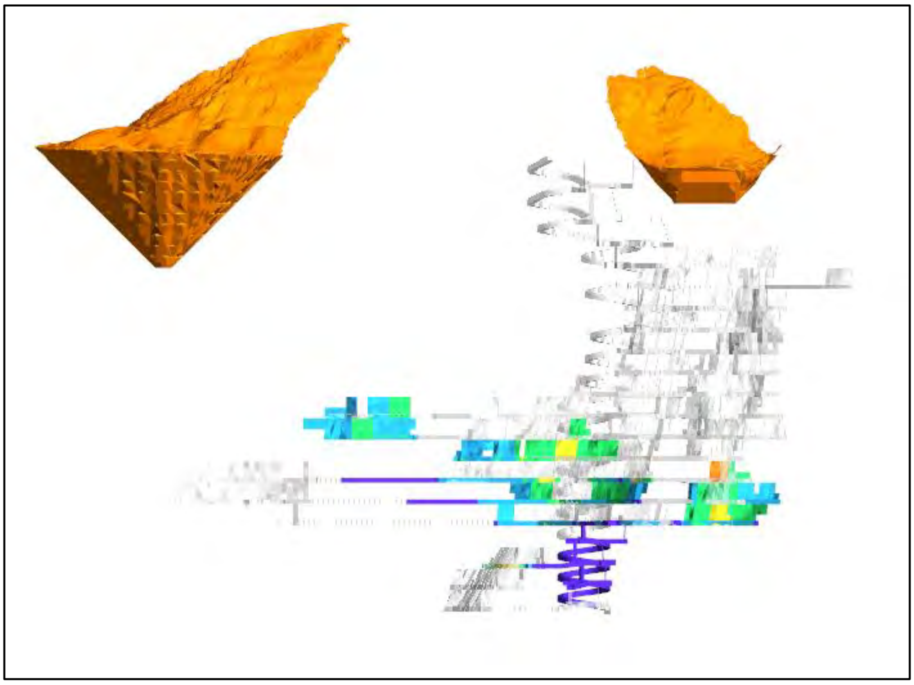


Figure 15-40: South Mine Schedule. Long-section View. Year 4 Mined Areas

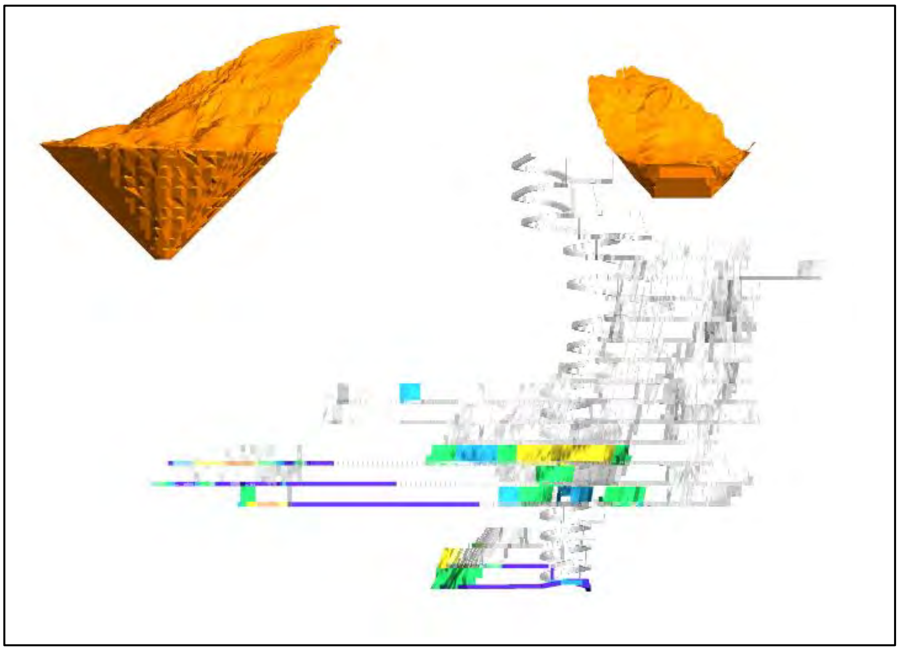


Figure 15-41: South Mine Schedule. Long-section View. Year 5 Mined Areas

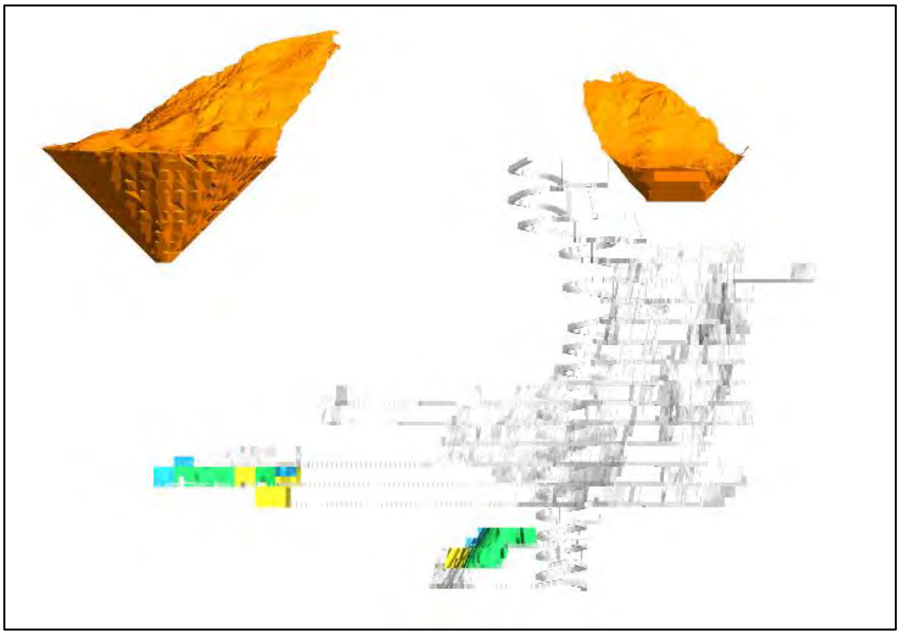


Figure 15-42: South Mine Schedule. Long-section View. Year 6 Mined Areas

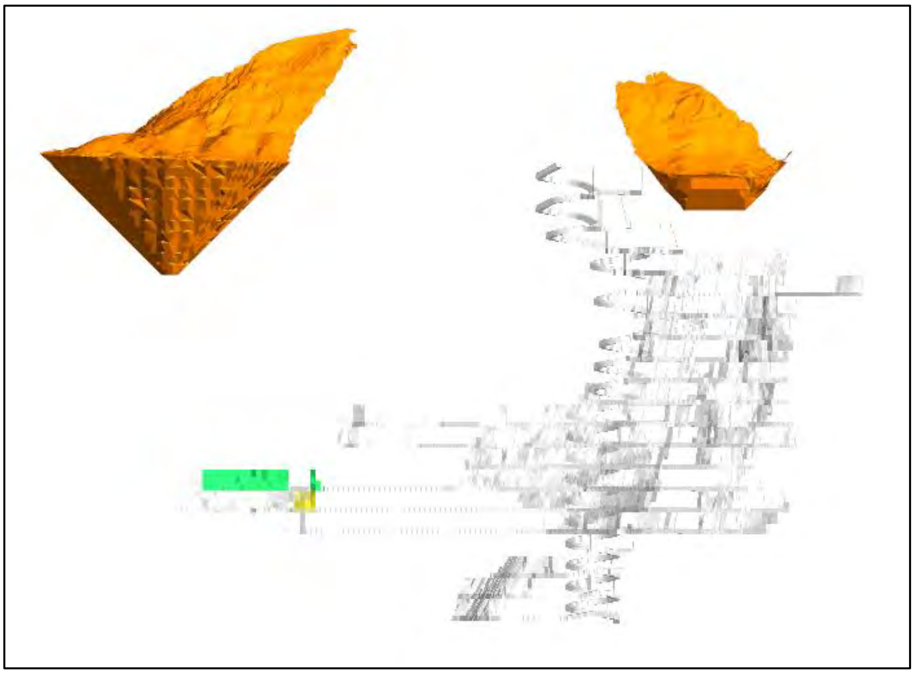


Figure 15-43: South Mine Schedule. Long-section View. Year 7 Mined Areas

15.5.5 Scheduled Physicals Output

A summary of the key mining physicals for the Hualilan Underground Design and Schedule are as shown in Table 15-15.

Table 15-15: Hualilan Key Underground Mining Physicals

Mining Physicals	Total	Unit
Mine Life	8	years
Total Mined Ore	5,797,588	t
Mined Ore Grade	4.39	AuEq g/t
Mined Au Ounces	819,178	AuEq Ounces
Mined Waste	2,054,978	t
Waste/Ore Ratio	0.35	t waste / t ore
Capital Development (Lateral)	20,689	m advance
Operating Development (Lateral)	24,555	m advance
Development Ore	606,709	t
Stope Ore	5,190,880	t
Stope / Development ore	8.56	t (Stope ore) / t dev ore
Stope Production Drill Metres	648,832	m
Stope Tonnes / Drill Metres	8.00	stope ore tonnes / prod drill m

The resultant mining physicals outputs from the schedule for the Hualilan underground are shown in Table 15-16.

Table 15-16: Hualilan Scheduled Underground Mine Physicals per Year

	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Ore Tonnes	25,690	669,974	995,524	820,990	971,987	1,018,474	1,005,118	289,831
Lateral Development (m)	6,353	9,966	10,440	9,260	10,538	5,021	2,925	0
Au Eq Grade (g/t)	4.27	4.59	4.24	4.27	4.42	4.42	4.58	4.02
AuEq Ounces	3,524	98,834	135,782	112,754	138,222	144,640	147,965	37,470

A production rate of ~1.0 Mtpa is achieved in year 2, reduces in Year 3, but then ramps back up to ~1.0 Mtpa in Years 4 to 6 (Figure 15-44). In Year 2, the main contributor to the ore production profile is the South mine until the North mine ramps up production and provides the majority of the ore feed of around 0.5 Mtpa for the next 5 years.

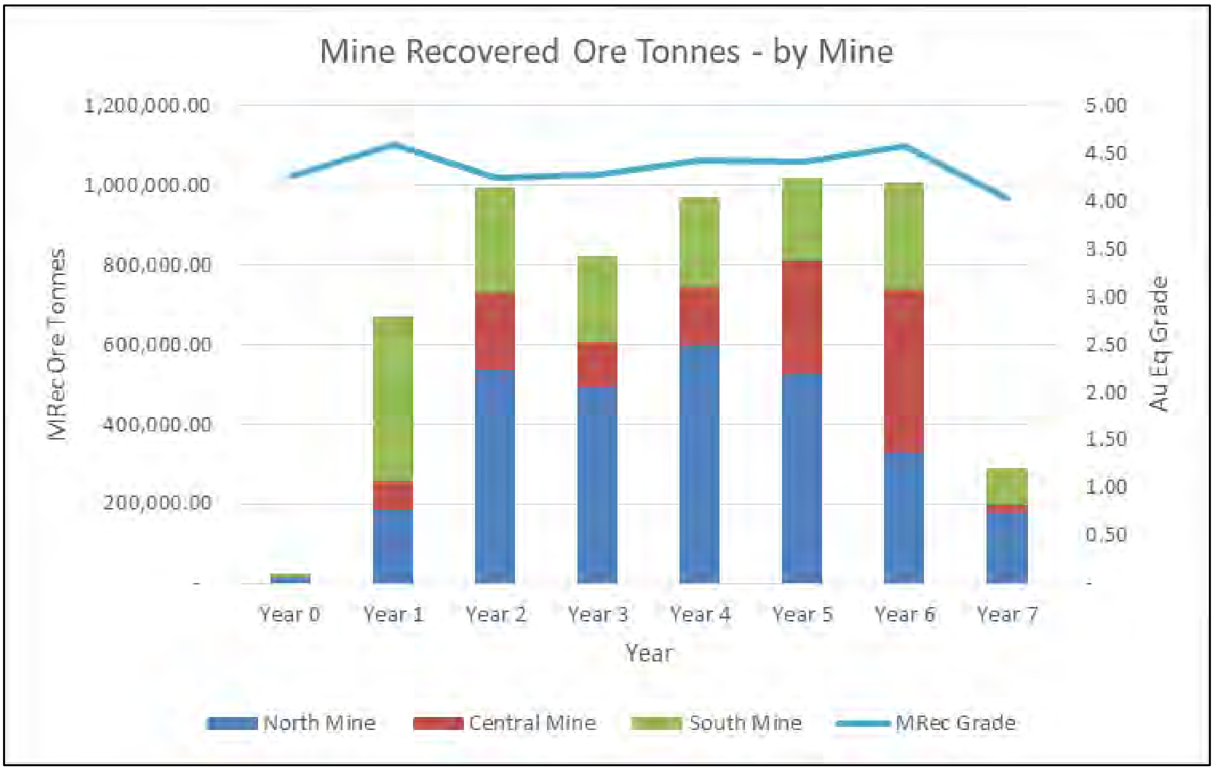


Figure 15-44: Hualilan Annual Scheduled Ore Tonnes

In order to achieve the desired ore tonnage and grade profile, the north and south mines start at the beginning of the schedule. The Central mine is delayed by one year. Figure 15-45 shows the scheduled annual lateral development chart by mine area.

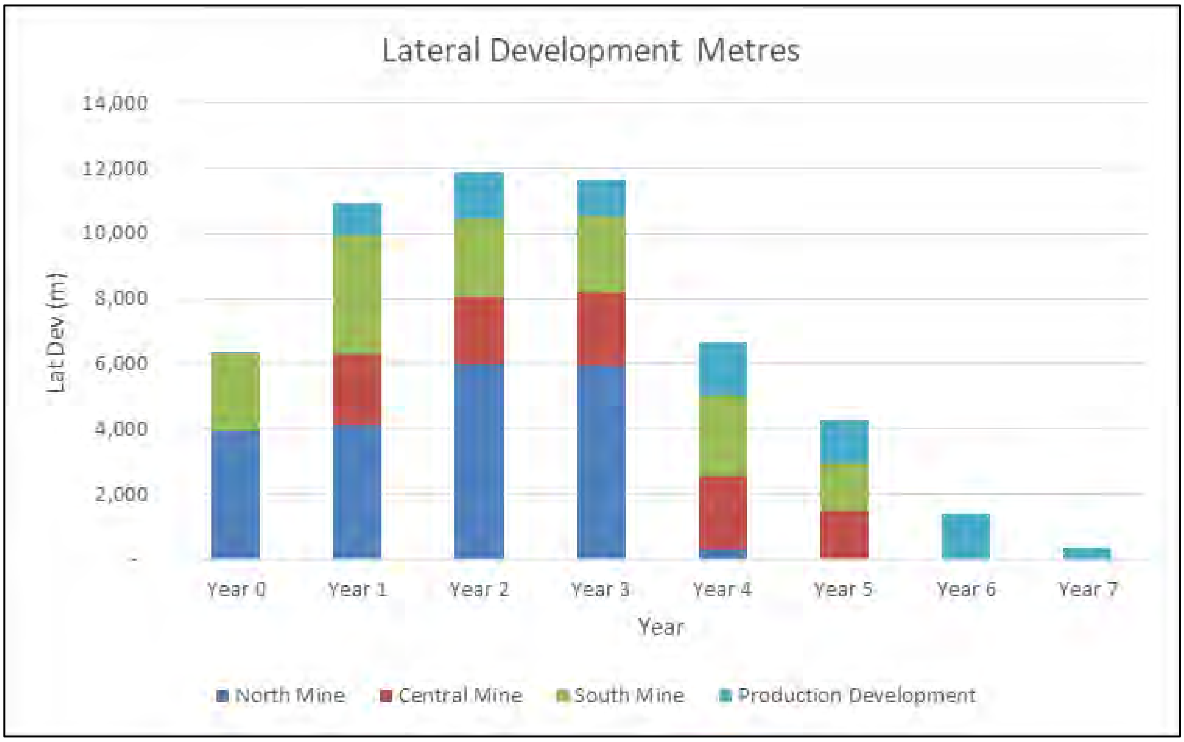


Figure 15-45: Hualilan Annual Scheduled Lateral Development

The schedule aims to ramp up to ~100 koz pa. The mined gold profile is shown in Figure 15-46.

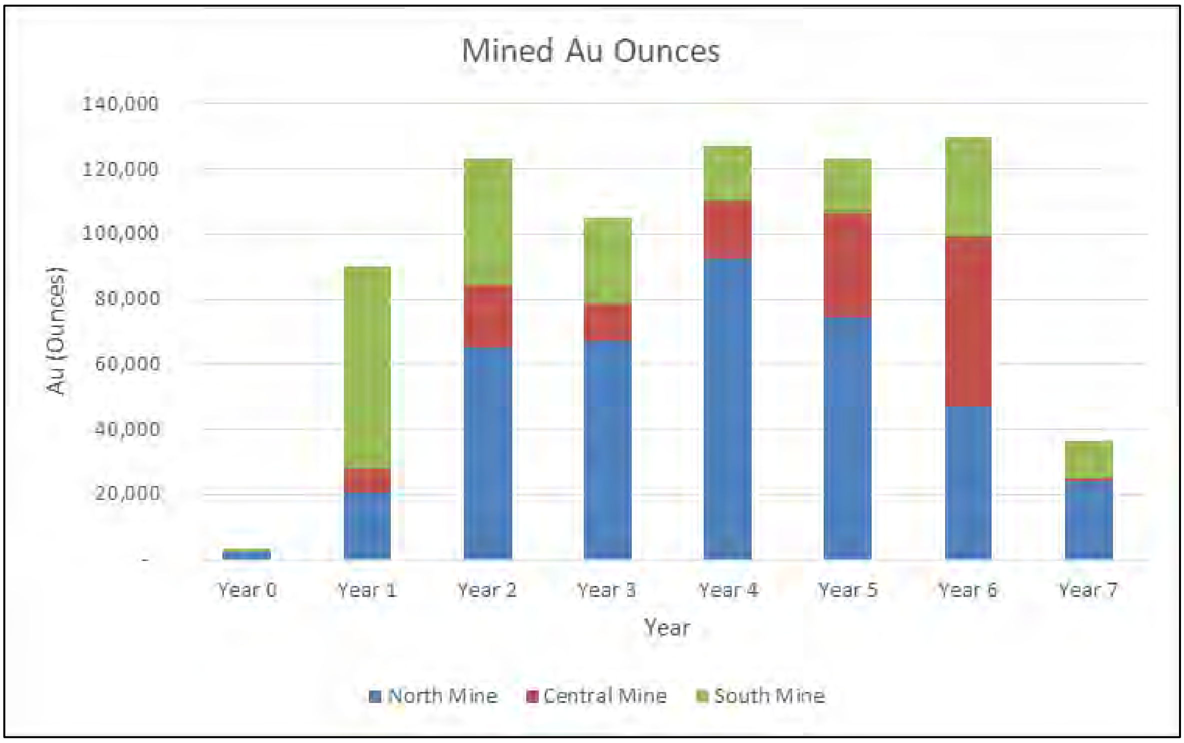


Figure 15-46: Mined Gold Ounces Profile

15.6 Combined Mine Schedule

The combined mine schedule was developed considering the material coming from the pits and from underground. The main features are detailed below.

- The mine plan was made from the optimisation pit shells, therefore the grades were penalised by a factor of 3% and the waste was increased by 20% to simulate pit design.
- Mill throughput of approximately 1 Mt per year.
- 8 years of life of mine, 1 year of pre-production (Year 0) and 7 years of production (Year 1-7).
- The pit will be mined at a constant rate of 200 kt per month from the beginning of Year 0 until the end of Year 3 (4 years mining in total). During the pre-production year, approximately 500 kt of ore will be stockpiled, while waste will be directed predominantly to the TSF embankment. Most of the cover material is not considered appropriate for TSF embankment construction and will be treated as topsoil and sent to dumps.
- The UG does not produce significant quantities of feed PMI until approximately the middle of Year 1, with most UG activity before that being development.
- The first year of plant feed will consume the high grade zinc PMI stockpiled from the OP during Y0, as well as some of the HG Au/LG Zn material, with the balance being direct from the OP and UG. Type A material (LG Au/LG Zn) will be fed from Y5 in minor amounts as feed top-up and fed in larger concentrations in Y7.

Figure 15-47 and Table 15-17 show the combined mine schedule. 18% of the total PMI comes from the pits (1.3 Mt) and 82% comes from the underground mine (5.8 Mt).

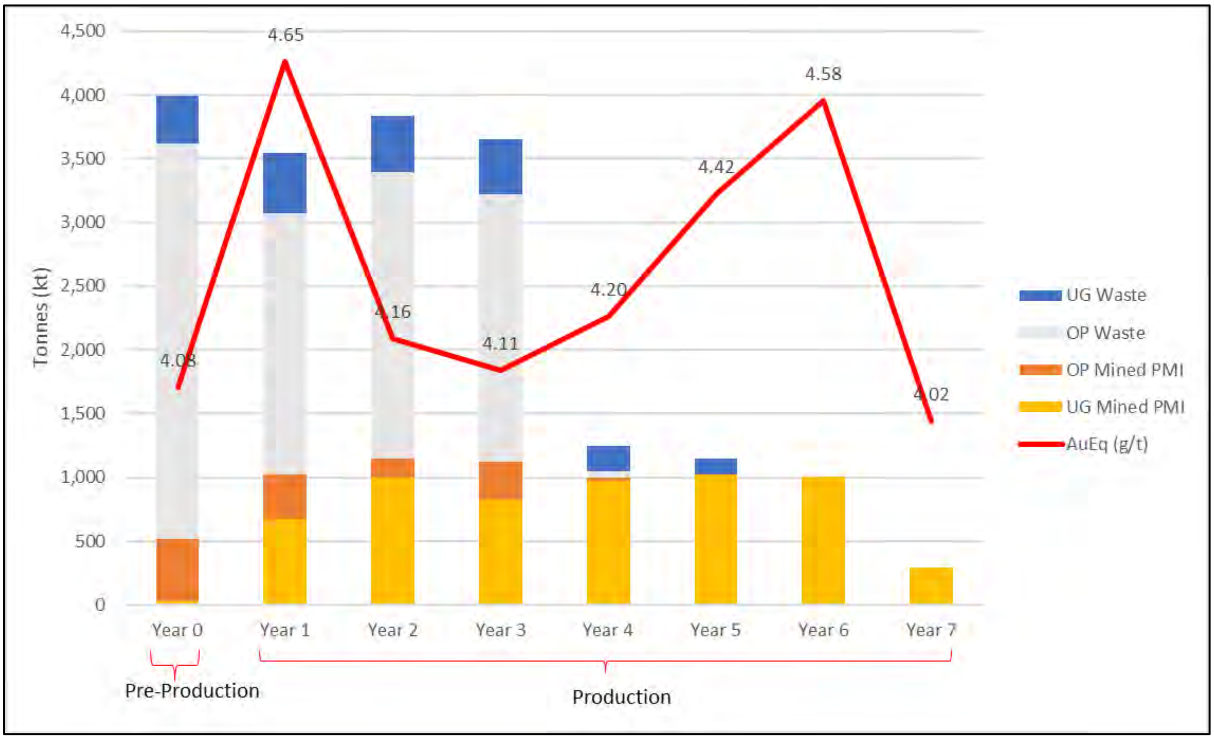


Figure 15-47: Combined Mine Schedule

Table 15-17: Mine Schedule

Material Source	Pre-Productio n	Production							Total PMI
Year	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	
Open Pit									
PMI (kt)	488	351	148	300	24				1,311
Au (g/t)	3.88	3.54	2.57	2.89	3.65				3.41
Ag (g/t)	22.87	19.40	24.85	23.68	20.59				22.31
Pb (%)	0.29	0.18	0.22	0.19	0.21				0.23
Zn (%)	1.30	2.78	1.05	0.69	0.84				1.52
AuEq (g/t)	4.42	4.39	3.10	3.27	4.03				3.99
Underground									
PMI (kt)	26	670	996	821	972	1018	1005	290	5,798
Au (g/t)	3.75	3.81	3.50	3.62	3.70	3.43	3.66	3.57	3.61
Ag (g/t)	9.44	11.87	14.73	10.68	9.97	13.01	12.78	9.27	12.09
Pb (%)	0.09	0.08	0.07	0.06	0.06	0.10	0.12	0.04	0.08
Zn (%)	0.84	1.34	1.19	1.11	1.28	1.75	1.59	0.72	1.35
AuEq (g/t)	4.27	4.59	4.24	4.27	4.42	4.42	4.58	4.02	4.39
Total Production									
PMI (kt)	514	1,020	1,144	1,121	996	1,018	1,005	290	7,108
Au (g/t)	3.88	3.72	3.38	3.42	3.70	3.43	3.66	3.57	3.57
Ag (g/t)	22.20	14.46	16.05	14.15	10.23	13.01	12.78	9.27	13.98
Pb (%)	0.28	0.11	0.09	0.09	0.06	0.10	0.12	0.04	0.11

Material Source	Pre-Production	Production							Total PMI
Year	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	
Zn (%)	1.27	1.83	1.17	1.00	1.27	1.75	1.59	0.72	1.38
AuEq (g/t)	4.41	4.52	4.09	4.00	4.41	4.42	4.58	4.02	4.32
Waste (kt)	3,482	2,527	2,692	2,531	248	128	0	0	11,607
Open Pit	3,106	2,048	2,250	2,099	50	0	0	0	9,552
Underground	376	479	442	432	199	128	0	0	2,055

The Au equivalent grade considers the contributions of Au, Ag and Zn depending on the subprocess. Equations:

1. For sub process Au <1.5: $Au\ Eq = \frac{37.17*Au\ grade+0.36*Ag\ grade}{37.17}$
2. For sub process Au>1.5 and Zn <1.5: $Au\ Eq = \frac{42.26*Au\ grade+0.42*Ag\ grade}{42.26}$
3. For sub process Au≥1.5 and Zn≥1.5: $Au\ Eq = \frac{42.26*Au\ grade+0.42*Ag\ grade+18.18*Zn\ grade}{42.26}$

The mine production is divided by sub processes as shown at Table 15-18. The mill production divided by sub process is shown in Table 15-18 and Figure 15-48. Of the mill process 72% of the PMI goes to sub processes that recover only Au and Ag (4.5 Mt), while 28% goes to sub process that recovers Au, Ag and Zn (1.8 Mt).

Table 15-18: Mine production by sub process

Sub Process	Pre-Production	Production							Total PMI
Year	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	
Au<1.5 & Zn<1.5									
PMI (kt)	149	91	87	140	21	5			493
Au (g/t)	0.81	0.84	0.86	0.69	0.93	1.13			0.80
Ag (g/t)	12.99	7.67	14.34	12.10	11.78	13.42			11.94
Pb (%)	0.13	0.08	0.09	0.11	0.10	0.08			0.11
Zn (%)	0.39	0.54	0.34	0.30	0.80	1.14			0.41
Au>1.5 & Zn<1.5									
PMI (kt)	224	473	764	713	644	519	606	255	4,198
Au (g/t)	4.23	3.13	3.22	3.83	3.98	3.72	3.55	3.46	3.61
Ag (g/t)	21.92	9.13	13.32	10.26	6.18	7.11	9.53	9.20	10.13
Pb (%)	0.31	0.09	0.09	0.07	0.04	0.03	0.03	0.03	0.07
Zn (%)	0.59	0.62	0.56	0.50	0.64	0.81	0.80	0.58	0.64
Au>1.5 & Zn>1.5 and Au<1.5 & Zn>1.5									

Sub Process	Pre-Production	Production							Total PMI
Year	Y0	Y1	Y2	Y3	Y4	Y5	Y6	Y7	
PMI (kt)	140	456	293	268	331	494	399	35	2,417
Au (g/t)	6.57	4.90	4.55	3.77	3.32	3.15	3.84	4.33	4.07
Ag (g/t)	32.44	21.33	23.68	25.60	17.99	19.19	17.73	9.73	21.07
Pb (%)	0.39	0.14	0.11	0.14	0.10	0.18	0.26	0.10	0.17
Zn (%)	3.30	3.35	3.00	2.68	2.53	2.75	2.79	1.70	2.88

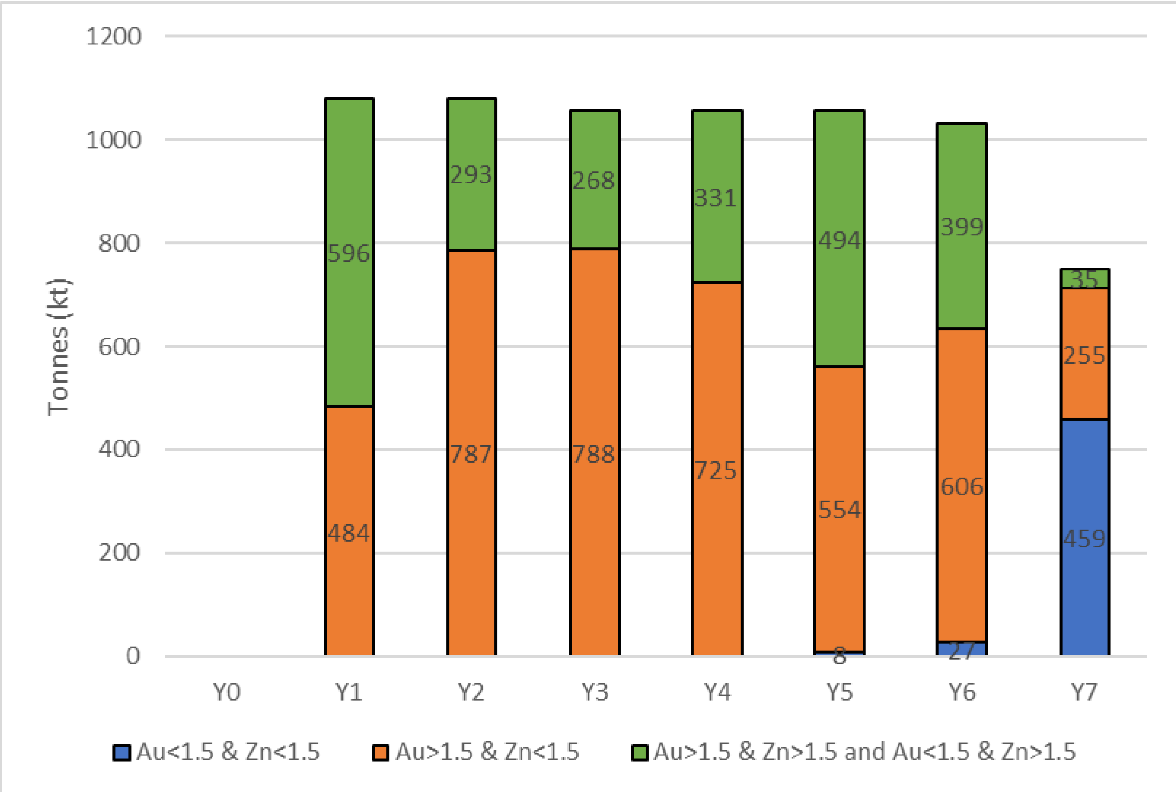


Figure 15-48: Mill production by sub process

PMI divided by resources category is shown in Figure 15-49. The block model has no measured resources, 81% of the Potential Mineral Inventory is indicated (5.7 Mt) and 19% is inferred material (1.4 Mt).

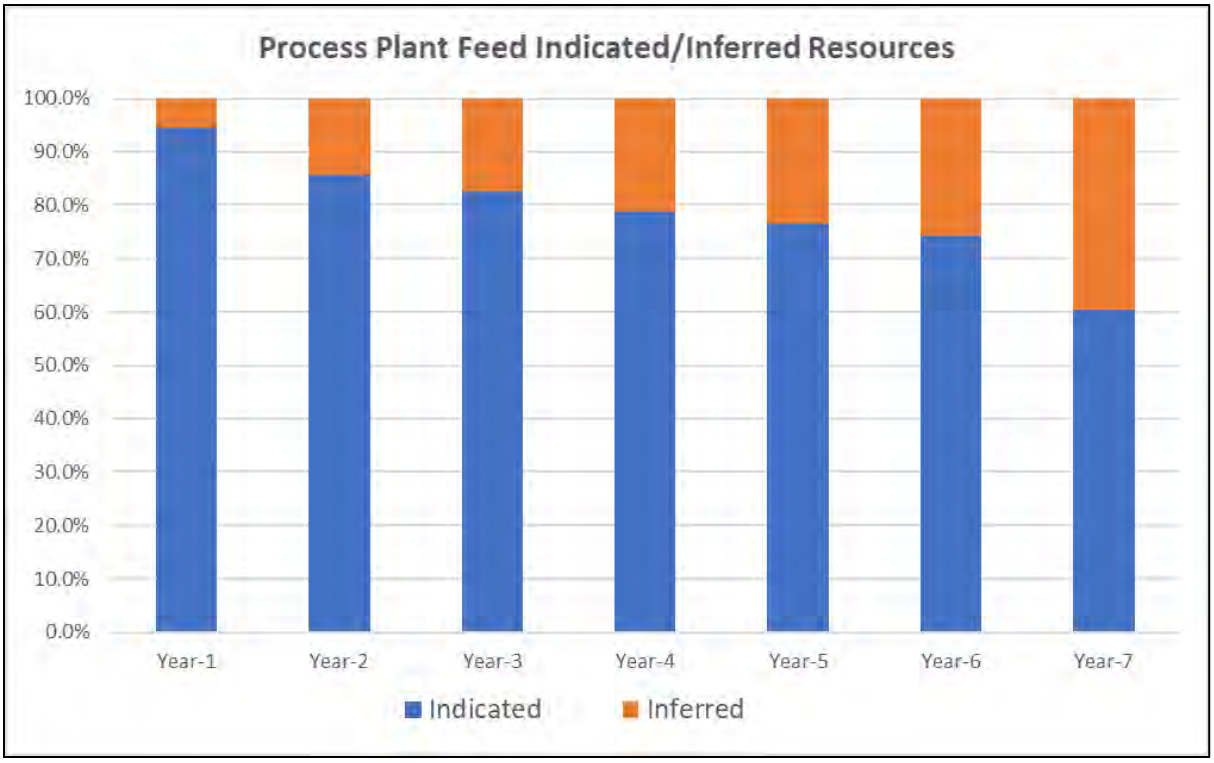


Figure 15-49: Mill throughput by category

15.7 Mine Equipment

15.7.1 Open pit

The maximum annual movement of the pit is 2.4 Mtpa and the open pit mine life is approximately five years, and it has been assumed that the equipment will be operated by a contractor. The fleet of equipment needed to fulfill the production of the pit was estimated based on similar operations and is shown in Table 15-19.

Table 15-19: Suggested open pit mining fleet

Equipment	Unit
Drill Epiroc D65	1
Excavator Liebherr R9150	1
Wheel Loader CAT 988	1
Trucks CAT773	3
Bulldozer D8T	1
Motor Grader 16M	1
Water Truck 40 m3	2
Lighting Towers	6
Service truck	1

15.7.2 Underground

Table 15-20 shows the equipment types assumed for the underground mine design and schedule. The equivalent sizes in different makes would also be appropriate for the chosen mining method and required drive sizes.

Table 15-20: Primary Mining Equipment for Hualilan Project

Primary Development Equipment	
Development drilling - Jumbo	Epiroc Boomer S2
Support - Bolter	Sandvik Ds422i
Support- Shotcrete sprayers	Normet Shotcrete Sprayer 6050WP
Loader -Scoop	Epiroc St14
Haulage - Truck	Epiroc Mt42
Raisebore & Slots drilling	Rhino 100
Primary Production Equipment	
Production Drill	Epiroc M4 S
Loader – Scoop	Epiroc St14
Haulage – Truck	Epiroc Mt42
Ancillary equipment	
Services truck	MacCleans Services Truck FL3
Fuel truck	Utimec LF1000
Water truck	Utimec LF1000
Grader	CAT Grader CAT 12F
Agitator truck	Normet Agitator truck LF600
Pick Up	Toyota HILUX LV

15.8 Waste Storage Area and Stockpile

The waste dump would be located to the northeast side, next to the North Pit. This waste storage facility will house material from the pit that is not considered competent for tailings construction such as cover, waste from the pit and underground which is not required the TSF embankment or other construction relate earthworks, so it will have a capacity of 1.9 Mm^3. The waste dump design parameters consider 10 m high benches and 37º batter angles.

A stockpile will be required for the storage of the high grade PMI during the pre-stripping and low grade coming from the pit. Its maximum annual storage capacity should be 550 kt and will be located on the east side of the pits adjacent to the primary crusher. The stockpile design parameters consider 10 m high benches and 37° batter angles.

A stability analysis for the waste dump and stockpile will be conducted during the PFS stage.

15.9 Tailings Storage Facility

The TSF would be located to the southeast of the pits, close to the process plant, within the lease boundaries and constrained by topography. The TSF was designed to store up to 2.9 Mm^3 of tailings (approximately 4.4 Mt), which is the balance of tailings which will not be used for paste backfill in the UG. Waste from the pit (oxides and fresh) will be used for the construction of the embankment. The design concept is based on downstream construction techniques with both upstream and downstream slopes of 3.0:1 (H:V). More information is described in Section 17.

16 PLANT DESIGN AND OPERATION

16.1 Process Introduction

The 1 Mtpa treatment plant has been designed to allow certain flotation stages and the Flotation Tails Leach (FTL) to be bypassed to allow the three different types of mineralised material to be processed using the same plant. For the Type C (Au ≥1.5 g/t, Zn ≥1.5%) the process flow sheet involves gravity gold recovery, followed by Cu/Pb flotation (and cleaning), followed by Zn flotation (and cleaning) with the tails subject to a Flotation Tailings Leach for remaining gold (FTL). For the Type B material (Au≥1.5 g/t Au, Zn <1.5%) the flow sheet involved gravity gold recovery with the Cu/Pb circuit operating as a simple bulk sulphide flotation (and cleaning) stage followed by FTL. For the low grade mineralised material Type A (Au <1.5 g/t Zn <1.5%) the same bulk flotation process route with the FTL bypassed due to the low Au grade in the flotation tails. Oxide material can also be treated by bypassing all flotation circuits to feed the gold leaching circuit directly from the milling circuit.

16.2 Bulk Flotation with FTL Circuit (Mineralised Material Type A and B)

The Process Flow Sheet is shown in Figure 16-1.

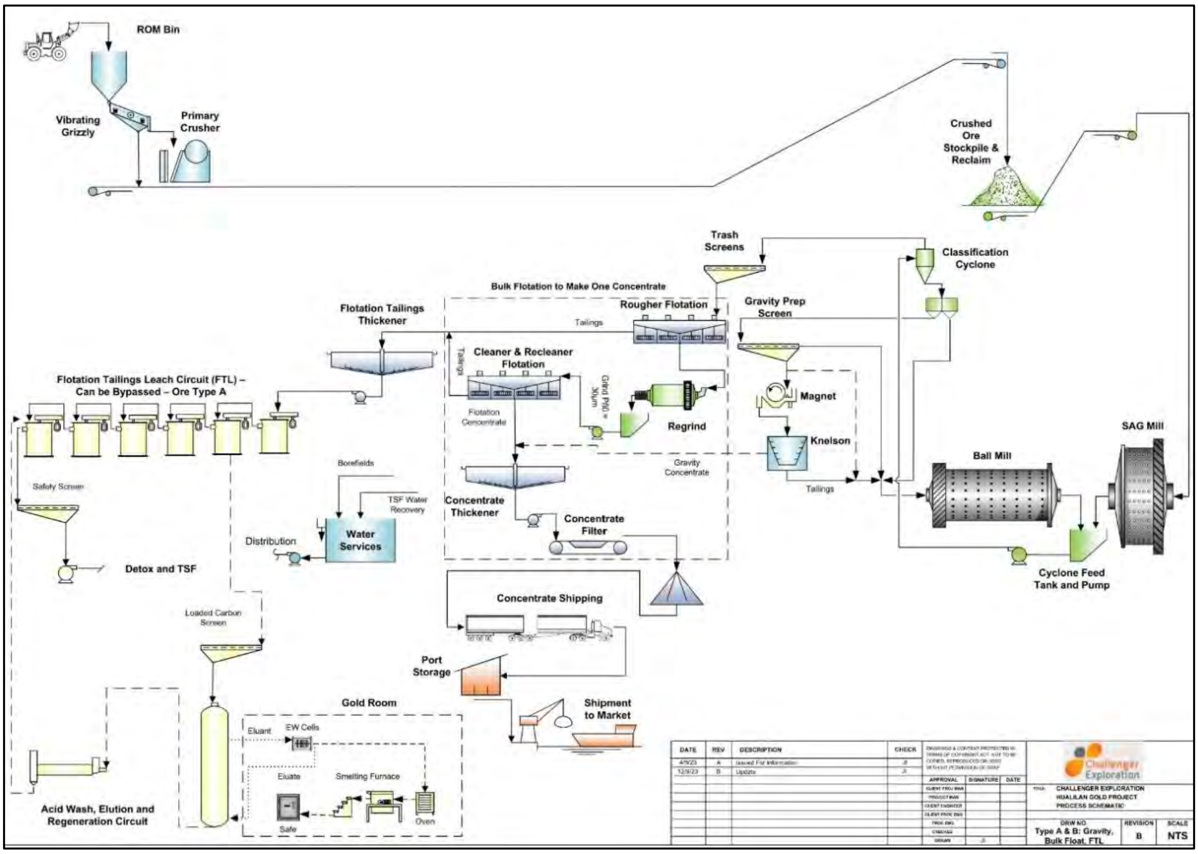


Figure 16-1: Schematic Gravity, Bulk Flotation and FTL Circuit

16.2.1 Process Description

The low grade Zn sulphide flotation plant, using bulk flotation with FTL, follows the process described in order in this section. The FTL circuit is bypassed if gold grade is too low; typically <1.5 g/t gold in feed (Type A Mineralised Material).

*16.2.1.1 **Primary Crushing***

- Ore Delivery: The mined Hualilan mineralised material will be delivered to the ROM Pad by mine haulage trucks and stockpiled.
- Crushing: Mineralised material is reclaimed and then crushed with a single Jaw crusher stage to a product size of P80 = 180mm. It is then stored in a crushed mineralised material stockpile with 16 hours of live residence.

*16.2.1.2 **Grinding and Primary Gravity Recovery***

- Grinding: Crushed mineralised material is reclaimed from the stockpile via an apron feeder, mixed with water, and fed into a SAG mill that operates with recycle pebble crusher and a ball mill, closed with classifying cyclones, and ground to a primary grind size of P80 = 75 microns to facilitate the separation of the valuable minerals from the waste.
- Gravity Recovery: The ground mineralised material is subjected to a gravity separation process, using centrifugal concentrators, to separate the heavy minerals, including gold, from the lighter minerals. The gravity concentrate is directed to the precious metal dewatering circuit.

*16.2.1.3 **Flotation***

- Conditioning: The ground mineralised material is mixed with water and chemicals such as collectors and frothers to create a slurry. The slurry is then agitated to ensure that the chemicals are evenly distributed throughout the mixture.
- Rougher Flotation: Air is injected into the slurry to create bubbles, which attach to the sulphide minerals and float them to the surface. The froth containing the sulphide minerals is then skimmed off and sent to a concentrate tank.
- Cleaning: The rougher concentrate is reground using a vertical regrind mill and a cleaning stage, where additional chemicals are added to further separate the valuable minerals from each other and from any remaining impurities. To further upgrade the sulphide minerals into a saleable concentrate. Cleaner flotation tailings, which include appreciable gold content, are directed to the precious metal dewatering circuit.

16.2.1.4 *Concentrate Dewatering*

- Concentrate Dewatering: The precious metal and sulphide concentrates are separately thickened and filtered to remove excess water and create final products with a high concentration of the respective metals.

16.2.1.5 *Flotation Tailings Leaching*

- Flotation Tailings Leach for Gold: The flotation tailings are thickened then sent to a flotation tailings leach plant using cyanide, to dissolve the gold into a solution. The leached gold is then adsorbed onto activated carbon.
- Elution and Electrowinning: The loaded carbon is then eluted, or stripped, to remove the adsorbed gold and other metals. The eluate is then sent to an electrowinning cell, where the gold is electrochemically deposited onto a cathode.
- Smelting and Refining: The gold deposit on the cathode is then smelted and refined to produce pure gold.

16.2.1.6 *Tailings Handling*

- Cyanide Detoxification: Leaching tailings are thickened then sent to the Inco cyanide detoxification circuit. The Inco cyanide detoxification process is a proven method of treating slurry containing cyanide and has been used in the mining industry for several decades. It is effective at reducing the toxicity of the tailings material.
- Tailings Disposal: The waste material from the detoxification process, known as tailings, is pumped to tailings storage facility to prevent contamination of the surrounding environment.

16.2.2 Main Processing Equipment- Bulk Flotation with FTL Circuit

The design criteria and equipment selection for each part of the bulk flotation with FTL circuit are summarised in Table 16-1 to Table 16-5.

Table 16-1: Summary Process Design Criteria and Major Equipment Selection for Crushing Circuit

Description	Units	Design
Production stage		LG
Feed rate	Nominal tpa	1,000,000
Ore Characteristics		
Resource category	Type	Inpit Resource
	Location	Stockpile/open pit
Bond ball Wi	kWh/t	20.2
Bond rod Wi	kWh/t	19.4
Abrasion index		0.27
DWi	kWh/m^3	7.20
A		-

Description	Units	Design
B		-
Axb		37.2
General Concentrator		
Annual treatment rate- design	t/a	1,000,000
Crushing Circuit		
ROM feed size - Maximum	mm	600
Product size P_{80} - Design	mm	160
Operating schedule	h/d	18
Operating time	%	80
Crusher annual run hours	hpa	5,256
Crusher Feed Rate - Nominal dry	dt/h	190
Crusher Circuit Type		Single Stage
Primary Crusher Equipment		Metso Jaw C106
Ore storage	Type	Open stockpile
Live capacity	t	2,000
	h	16

Table 16-2: Summary Process Design Criteria and Major Equipment Selection for Grinding and Gravity Circuit

Description	Units	Design
Concentrator		
Grinding circuit	Type	SAG Mill and Pebble Crusher with Ball Mill closed with Cyclones (SABC)
SAG Mill		SAG Mill 19' Diam x 10.2' Length with 1.4MW Drive
Ball Mill		Ball Mill 15.5' Diam x 26.1' Length with 2.9MW Drive
Pebble Crusher Equipment		Metso Cone HP200
Feed F80	mm	160
Mill Circuit Product P80	um	75
Circulating Load - Design	%	275
Operating schedule	d/a	365
Daily treatment rate	t/d	2,740
Operating schedule	h/d	24
Operating time	%	92
Concentrator annual run hours	h/a	8,059
Concentrator Feed Rate - Nominal dry	dt/h	124
Primary Gravity circuit	location	Split of Cyclone Underflow
	Type	centrifugal
	Model	2 x Knelson
Gravity concentrate treatment	Type	Export for Sale
	location	Combined with Clnr Concentrate

Table 16-3: Summary Process Design Criteria and Major Equipment Selection Bulk Flotation Circuit

Description	Units	Design
Flotation	Feed	Mill cyclone overflow
Flotation	Cell	Mech

Description	Units	Design
Circuit Selection Rougher	No.	4 x TC50
Circuit Selection Cleaner	No.	3 x OK8
Circuit Selection Recleaner	No.	3 x OK8
Rougher Flotation	Conc	Flotation con
Cleaner Flotation	Regrind P80µm	20
Cleaner Flotation	No stage	2
Regrind Mill	Type	HIG Mill
	kW	500
No. of Regrind Mills	no.	1
Regrind Mill Specific Energy	kWh/t	33.4
Regrind Classification	Type	Pre-cyclone & Internal
Mass Pull from Rougher	%	7.444
Mass Pull to Final Cleaner Concentrate	%	2.4965
Rougher Conditioning Time	min	11
Rougher Residence	min	32
Cleaner 1 Residence	min	30
Cleaner 2 Residence	min	30
Concentrate thickening	Type	High rate
Concentrate thickener flux rate	$t/m^2/h$	0.20
Concentrate thickener diameter	m	5
Concentrate thickener underflow density	%	65
Copper Concentrate Storage time	h	24.0
Copper Conc Storage Tank Vol	m^3	70
Concentrate filtration	Type	Larox PF
Filter solids loading	$kg/m^2/h$	250
Filtration Area required	m^2	14.2
Filter Model	Model	TBA
No. of Filters	no.	1
Concentrate loadout	Type	Bulk/ Bulk bag
Rougher Flotation tailing	Destination	Tailings Thickener

Table 16-4: Summary Process Design Criteria and Major Equipment Selection for Flotation Tailings Leach Circuit (FTL)

Description	Units	Design
CIL (FTL)		
Rougher Tailings	Type	Flotation Tailings
Rougher Tailings Throughput	t/h	114.6
Au Grade - Rougher Tailings	Au g/t	0.11
Preleach thickening	Type	High rate
Preleach thickener flux rate	$t/m^2/h$	0.50
Preleach thickener underflow density (Post Dilution)	%	45
Preleach thickener diameter	m	19
Feed rate	t/a	923,960

Description	Units	Design
Trash Screen	Type	Linear, 600x700µm, 10m²
Total Leach/CIL Retention time	h	25.0
Leach dissolution	Au %	64
Leach tank	No	1
Tank Vol	m^3/tank	750
Tank Size	mD x mH	9.3mD x 11.6m H
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge
CIL Tanks	No	6
Tank Vol	m^3/tank	750
Tank Size	mD x mH	9.3mD x 11.6m H
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge
Intertank screens	Type	Mech Swept
	Model	Kemix MPS (P) 500
Carbon concentration	g/L	12
Carbon loading	Au g/t	500
Carbon transfer	Type	Pump
Carbon Safety Screen	Type	Linear, 800x800µm, 10m²
Elution circuit		
Loaded carbon screen	Type	Linear, 800x800µm, 0.5m²
Elution	Type	Zadra
Operating Days	d/week	5
Column Size	t/strip	0.50
Acid Wash	Type	Separate column
Elution temp	°C	120
Electrowinning	Type of Cathode	Steel Wool
Smelting	Type	Smelt calcine
Carbon regeneration	Kiln Type	Horizontal
Carbon regeneration Kiln Size	kg/h	30

Table 16-5: Summary Process Design Criteria and Major Equipment Selection for Tailings Circuit

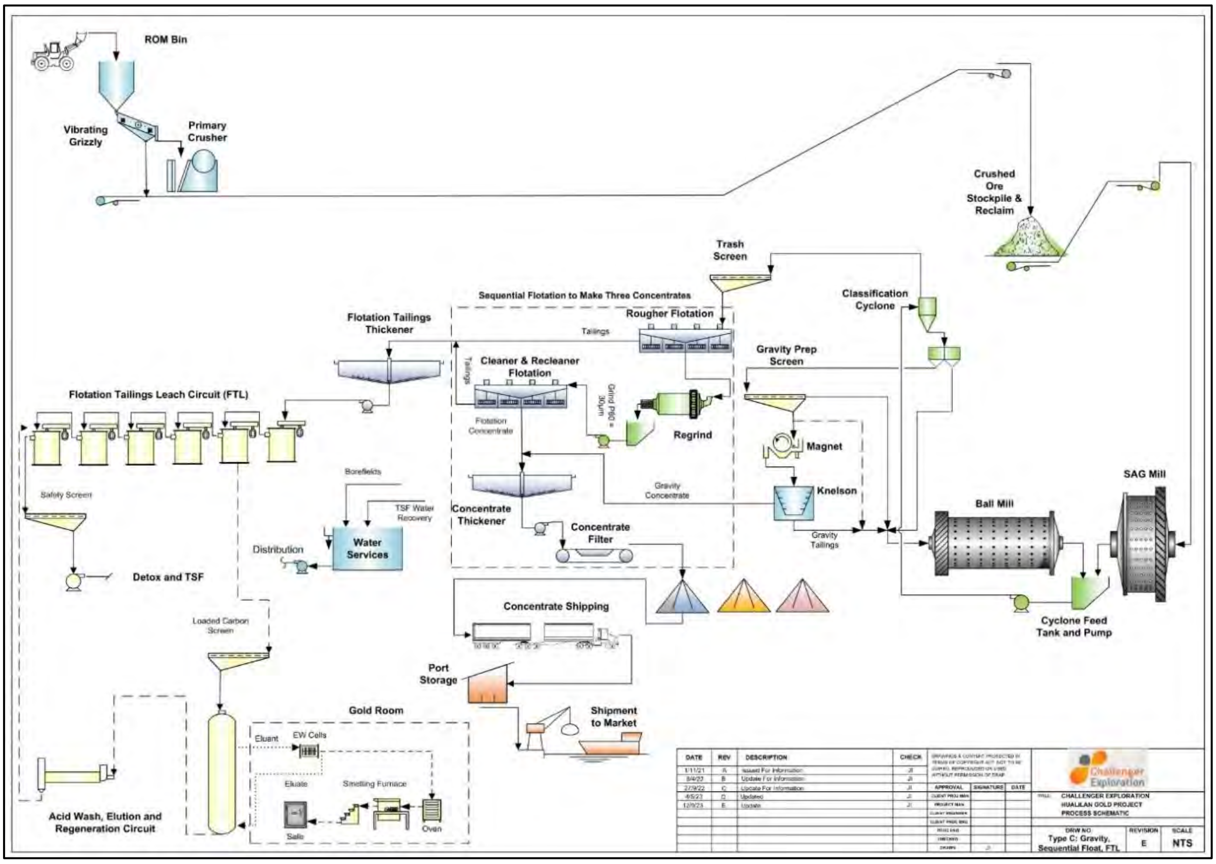
Description	Units	Design
Detox & Tailings		
Tailings Thickening	Type	High Rate
Tails thickener flux rate	$t/m^2/h$	0.50
Tails thickener underflow density	%	55
Tails thickener diameter	m	19
Cyanide detox	Type	INCO
Detox tank	No	1
Tank Vol	m^3/tank	400
Tank Size	mD x mH	9.0 x 11.3

Description	Units	Design
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge
WADCN	level ppm	10
Tailings Disposal	Type	Sub ariel multispigot

16.3 Sequential Flotation with FTL Circuit (Ore Type C)

The Process Flow Sheet for the gravity, sequential flotation and FTL circuit is shown in Figure 16-2.

Figure 16-2: Schematic Gravity, Sequential Flotation and FTL Circuit



16.3.1 Process Description

The high grade zinc sulphide flotation plant using sequential flotation with flotation tailings leach follows the process outlined below.

*16.3.1.1 **Primary Crushing***

- Ore Delivery: The mined Hualilan mineralised material will be delivered to the ROM Pad by mine haulage trucks and stockpiled.

- Crushing: Mineralised material is reclaimed and then crushed with a single Jaw crusher stage to a product size of P80 = 180mm. It is then stored in a crushed mineralised material stockpile with 16 hours of live residence.

16.3.1.2 *Grinding and Primary Gravity Recovery*

- Grinding: Crushed mineralised material is reclaimed from the stockpile via an apron feeder, mixed with water, and fed into a SAG mill that operates with recycle pebble crusher and a ball mill closed with classifying cyclones and ground to a primary grind size of P80 = 50 microns to facilitate the separation of the valuable minerals from the waste. The pebble crusher is unnecessary for this mineralised material and will be bypassed.
- Gravity Recovery: The ground mineralised material is subjected to a gravity separation process, using centrifugal concentrators, to separate the heavy minerals, including gold, from the lighter minerals. The gravity concentrate is directed to the precious metal dewatering circuit.

16.3.1.3 *Flotation*

- Conditioning: The ground mineralised material as a slurry is mixed with flotation chemicals including collectors, frothers and depressants for the zinc minerals and agitated to ensure that the chemicals are evenly distributed throughout the mixture.
- Copper plus Lead Flotation: Air is injected into the slurry to create bubbles, which attach to the copper and lead sulphide minerals and float them to the surface. The froth containing the copper minerals is then skimmed off and sent to a concentrate tank. The circuit includes rougher flotation stage and two stages of cleaner flotation to further upgrade the copper and lead minerals into a saleable concentrate. Cleaner flotation tailings which include appreciable gold content are directed to the precious metal dewatering circuit.
- Zinc Flotation: The tailings from the copper flotation process are then subjected to a zinc flotation process which includes activation of the depressed zinc minerals. Air is again injected into the slurry to create bubbles, which attach to the zinc sulphide minerals and float them to the surface. The froth containing the zinc minerals is then skimmed off and sent to a concentrate tank. The circuit includes rougher flotation stage, regrinding of rougher concentrate using a vertical regrind mill and four stages of cleaner flotation to further upgrade the zinc minerals into a saleable concentrate. Cleaner flotation tailings which include appreciable gold content are directed to the precious metal dewatering circuit.

16.3.1.4 *Concentrate Dewatering*

- Concentrate Dewatering: The precious metal, copper plus lead and zinc concentrates are separately thickened and filtered to remove excess water and create final products with a high concentration of the respective metals.

16.3.1.5 *Flotation Tailings Leaching*

- Flotation Tailings Leach for Gold: The flotation tailings are thickened then sent to a flotation tailings leach plant using cyanide, to dissolve the gold into a solution. The leached gold is then adsorbed onto activated carbon.
- Elution and Electrowinning: The loaded carbon is then eluted, or stripped, to remove the adsorbed gold and other metals. The eluate is then sent to an electrowinning cell, where the gold is electrochemically deposited onto a cathode.
- Smelting and Refining: The gold deposit on the cathode is then smelted and refined to produce pure gold.

16.3.1.6 *Tailings Handling*

- Cyanide Detoxification: Leaching tailings are thickened then sent to the Inco cyanide detoxification circuit. The Inco cyanide detoxification process is a proven method of treating slurry containing cyanide and has been used in the mining industry for several decades. It is effective at reducing the toxicity of the tailings material.
- Tailings Disposal: The waste material from the detoxification process, known as tailings, is pumped to tailings storage facility to prevent contamination of the surrounding environment.

16.3.2 Main Processing Equipment- Sequential Flotation with FTL Circuit

The design criteria and equipment selection for each part of the bulk flotation with FTL circuit are summarised in Table 16-6 to Table 16-11.

Table 16-6: Summary Process Design Criteria and Major Equipment Selection for Crushing Circuit

Description	Units	Design
Production stage		HG
Feed rate	Nominal tpa	1,000,000
Ore Characteristics		
Resource category	Type	Inpit Resource
	Location	Stockpile/open pit
Bond ball Wi	kWh/t	15.3
Bond rod Wi	kWh/t	16.0
Abrasion index		0.17
DWi	kWh/m^3	5.55
A		-

Description	Units	Design
B		-
Axb		56.8
General Concentrator		
Annual treatment rate- design	t/a	1,000,000
Crushing Circuit		
ROM feed size - Maximum	mm	600
Product size P_{80} - Design	mm	160
Operating schedule	h/d	18
Operating time	%	80
Crusher annual run hours	hpa	5,256
Crusher Feed Rate - Nominal dry	dt/h	190
Crusher Circuit Type		Single Stage
Primary Crusher Equipment		Metso Jaw C106
Ore storage	Type	Open stockpile
Live capacity	t	2,000
	h	16

Table 16-7: Summary Process Design Criteria and Major Equipment Selection for Grinding and Gravity Circuit

Description	Units	Design
Concentrator		
Grinding circuit	Type	SAG Mill with Ball Mill closed with Cyclones (SAB)
SAG Mill		SAG Mill 19' Diam x 10.2' Length with 1.4MW Drive
Ball Mill		Ball Mill 15.5' Diam x 26.1' Length with 2.9MW Drive
Pebble Crusher Equipment		Bypassed
Feed F80	mm	160
Mill Circuit Product P80	µm	50
Circulating Load - Design	%	275
Operating schedule	d/a	365
Daily treatment rate	t/d	2,740
Operating schedule	h/d	24
Operating time	%	92
Concentrator annual run hours	h/a	8,059
Concentrator Feed Rate - Nominal dry	dt/h	124
Primary Gravity circuit	location	Split of Cyclone Underflow
	Type	centrifugal
	Model	2 x Knelson
Gravity concentrate treatment	Type	Export for Sale
	location	Zn Clnr Tailings and Cu/Pb Clnr Tailings
Concentrate Dewatering	Conc	Precious Metal Concentrate (PM Conc)
Concentrate thickening	Type	High rate
Concentrate thickener flux rate	$t/m^2/h$	TBA
Concentrate thickener diameter	m	6

Description	Units	Design
Concentrate thickener underflow density	%	65
Copper Concentrate Storage time	h	24.0
Copper Conc Storage Tank Vol	m^3	89
Concentrate filtration	Type	Larox PF
Filter solids loading	$kg/m^2/h$	250
Filtration Area required	m^2	17.9
Filter Model	Model	TBA
No. of Filters	no.	1
Concentrate loadout	Type	bulk/ bulk bag

Table 16-8: Summary Process Design Criteria and Major Equipment Selection for Copper and Lead Flotation Circuit

Description	Units	Design
Flotation - Cu/Pb	Feed	Mill cyclone overflow
Flotation	Cell	Mech
Circuit Selection Rougher	No.	4 x OK16
Circuit Selection Cleaner	No.	2 x OK3
Circuit Selection Recleaner	No.	2 x OK3
Rougher Flotation	Conc	Cu/Pb con
Cleaner Flotation	Regrind P80µm	N/A
Cleaner Flotation	No stage	2
Regrind Mill	Type	N/A
Mass Pull from Cu/Pb Rougher	%	2.0202
Mass Pull to Final Cu/Pb Concentrate	%	0.58548
Cu/Pb Rougher Conditioning Time	min	6
Cu/Pb Rougher Residence	min	8
Cu/Pb Cleaner 1 Residence	min	20
Cu/Pb Cleaner 2 Residence	min	20
Concentrate thickening	Type	High rate
Concentrate thickener flux rate	$t/m^2/h$	0.20
Concentrate thickener diameter	m	3
Concentrate thickener underflow density	%	65
Copper Concentrate Storage time	h	24.0
Copper Conc Storage Tank Vol	m^3	17
Concentrate filtration	Type	Pressure Filter
Filter solids loading	$kg/m^2/h$	250
Filtration Area required	m^2	3.3
Filter Model	Model	TBA
No. of Filters	no.	1
Concentrate loadout	Type	bulk/ bulk bag
Rougher Flotation tailing	Destination	Zinc Roughers

Table 16-9: Summary Process Design Criteria and Major Equipment Selection for Zinc Flotation Circuit

Description	Units	Design
Flotation - Zn	Feed	Cu/Pb Tailings
Flotation	Cell	mech

Description	Units	Design
Circuit Selection Rougher	No.	4 x OK38
Circuit Selection Cleaner 1	No.	4 x OK8
Circuit Selection Cleaner 2	No.	4 x OK8
Circuit Selection Cleaner 3	No.	4 x OK8
Circuit Selection Cleaner 4	No.	4 x OK3
Rougher Flotation	Conc	Zn con
Cleaner Flotation	Regrind P80µm	20
Cleaner Flotation	No stage	4
Regrind Mill	Type	HIG Mill
	kW	600
No. of Regrind Mills	no.	1
Regrind Mill Specific Energy	kWh/t	25.4
Regrind Classification	Type	Pre-cyclone
Flotation final concentrates	Type	Zn con
Mass Pull from Zinc Rougher	%	13.385
Mass Pull to Final Zinc Concentrate	%	6.9872
Zinc Rougher Conditioning Time	min	21
Zinc Rougher Residence	min	25
Zinc Cleaner 1 Residence	min	23
Zinc Cleaner 2 Residence	min	15
Zinc Cleaner 3 Residence	min	15
Zinc Cleaner 4 Residence	min	8
Concentrate thickening	Type	High rate
Concentrate thickener flux rate	$t/m^2/h$	0.20
Concentrate thickener diameter	m	8
Concentrate thickener underflow density	%	65
Copper Concentrate Storage time	h	24.0
Copper Conc Storage Tank Vol	m^3	194
Concentrate filtration	Type	Larox PF
Filter solids loading	$kg/m^2/h$	250
Filtration Area required	m^2	39.2
Filter Model	Model	TBA
No. of Filters	no.	1
Concentrate loadout	Type	bulk/ bulk bag
Rougher Flotation tailing	Destination	CIL

Table 16-10: Summary Process Design Criteria and Major Equipment Selection for Flotation Tailings Leach Circuit

Description	Units	Design
CIL		
Additional Bleed from LG Cleaner Tailings	Type	Cleaner Total Tailings from LG Plant
Additional Throughput - LG Cleaner Tailings	t/h	12.3
Au Grade - LG Cleaner Tailings	Au g/t	0.63
Rougher Tailings	Type	Rougher Tailings Plant
Rougher Tailings Throughput	t/h	109.2

Description	Units	Design
Au Grade - Rougher Tailings	Au g/t	0.45
Preleach thickening	Type	High rate
Preleach thickener flux rate	$t/m^2/h$	0.50
Preleach thickener underflow density (Post Dilution)	%	45
Preleach thickener diameter	m	20
Feed rate	t/a	979,018
Trash Screen	Type	Linear, 600x700µm, $5m^2$
Total Leach/CIL Retention time	h	24.4
Leach dissolution	Au %	67
Leach tank	No	1
Tank Vol	m^3/tank	775
Tank Size	mD x mH	9.4mD x 11.8m H
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge
CIL Tanks	No	6
Tank Vol	m^3/tank	775
Tank Size	mD x mH	9.4mD x 11.8m H
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge
Intertank screens	Type	Mech Swept
	Model	Kemix MPS (P) 500
Carbon concentration	g/L	12
Carbon loading	Au g/t	1,000
Carbon transfer	Type	Pump
Carbon Safety Screen	Type	Linear, 800x800µm, $5m^2$
Elution circuit		
Loaded carbon screen	Type	Linear, 800x800µm, $0.5m^2$
Elution	Type	Zadra
Operating Days	d/week	6
Column Size	t/strip	1.00
Acid Wash	Type	Separate column
Elution temp	°C	120
Electrowinning	Type of Cathode	Steel Wool
Smelting	Type	Smelt calcine
Carbon regeneration	Kiln Type	Horizontal
Carbon regeneration Kiln Size	kg/h	60

Table 16-11: Summary Process Design Criteria and Major Equipment Selection for Tailings Circuit

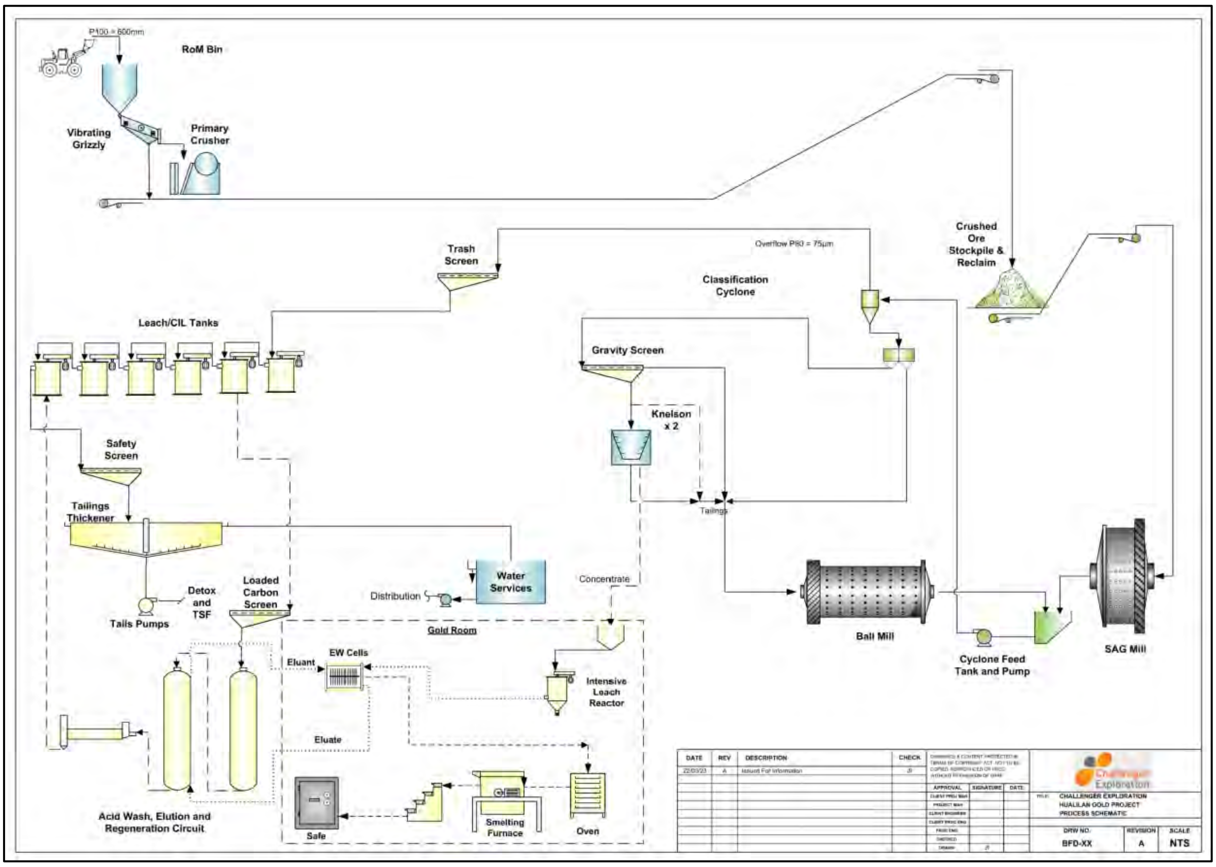
Description	Units	Design
Detox & Tailings		
Tailings Thickening	Type	High Rate
Tails thickener flux rate	$t/m^2/h$	0.50
Tails thickener underflow density	%	55
Tails thickener diameter	m	20
Cyanide detox	Type	INCO

Description	Units	Design
Detox tank	No	1
Tank Vol	m^3/tank	400
Tank Size	mD x mH	6.8 x 8.7
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge
WADCN	level ppm	10
Tailings Disposal	Type	Sub ariel multispigot

16.4 CIL Circuit

The Process Flow Sheet for the CIL circuit is shown in Figure 16-3.

Figure 16-3: Schematic Gravity and CIL Circuit



16.4.1 Process Description

The gravity and CIL plant for treating of oxide mineralised materials follows the process outlined in the following sub-sections.

16.4.1.1 Primary Crushing

- Ore Delivery: The mined Hualilan mineralised material will be delivered to the ROM Pad by mine haulage trucks and stockpiled.
- Crushing: Mineralised material is reclaimed and then crushed with a single Jaw crusher stage to a product size of P80 = 180mm. It is then stored in a crushed mineralised material stockpile with 16 hours of live residence.

16.4.1.2 Grinding and Primary Gravity Recovery

- Grinding: Crushed mineralised material is reclaimed from the stockpile via an apron feeder, mixed with water, and fed into a SAG mill that operates with recycle pebble crusher and a ball mill closed with classifying cyclones and ground to a primary grind size of P80 = 75 microns to facilitate the separation of the valuable minerals from the waste.
- Gravity Recovery: The ground mineralised material is subjected to a gravity separation process, using centrifugal concentrators, to separate the heavy minerals, including gold, from the lighter minerals. The gravity concentrate is leached for gold using an intensive leaching.

16.4.1.3 CIL Circuit

- CIL for Gold Recovery: The ground slurry is then sent to a CIL leach plant using cyanide, to dissolve the gold into a solution. The leached gold is then adsorbed onto activated carbon.
- Elution and Electrowinning: The loaded carbon is then eluted, or stripped, to remove the adsorbed gold and other metals. The elution eluate and the intensive leach pregnant solution is then sent to an electrowinning cell, where the gold is electrochemically deposited onto a cathode.
- Smelting and Refining: The gold deposit on the cathode is then smelted and refined to produce pure gold.

16.4.1.4 Tailings Handling

- Cyanide Detoxification: Leaching tailings are thickened then sent to the Inco cyanide detoxification circuit. The Inco cyanide detoxification process is a proven method of treating slurry containing cyanide and has been used in the mining industry for several decades. It is effective at reducing the toxicity of the tailing’s material.
- Tailings Disposal: The waste material from the detoxification process, known as tailings, is pumped to tailings storage facility to prevent contamination of the surrounding.

16.4.2 Main Processing Equipment - CIL Circuit

The design criteria and equipment selection for each part of the bulk flotation with FTL circuit are summarised in Table 16-12 to Table 16-15.

Table 16-12: Summary Process Design Criteria and Major Equipment Selection for Crushing Circuit

Description	Units	Design
Production stage		LG
Feed rate	Nominal tpa	1,000,000
Ore Characteristics		
Resource category	Type	Inpit Resource
	Location	Stockpile/open pit
Grade - Design		
Au	g/t	0.69
Ag	g/t	2.95
Cu	%	0.03
Pb	%	0.05
Zn	%	0.15
Bond ball Wi	kWh/t	20.2
Bond rod Wi	kWh/t	19.4
Abrasion index		0.27
DWi	kWh/m^3	7.20
A		-
b		-
Axb		37.2
General Concentrator		
Annual treatment rate- design	t/a	1,000,000
Crushing Circuit		
ROM feed size - Maximum	mm	600
Product size P_{80} - Design	mm	160
Operating schedule	h/d	18
Operating time	%	80
Crusher annual run hours	hpa	5,256
Crusher Feed Rate - Nominal dry	dt/h	190
Crusher Circuit Type		Single Stage
Primary Crusher Equipment		Metso Jaw C106
Ore storage	Type	Open stockpile
Live capacity	t	2,000
	h	16

Table 16-13: Summary Process Design Criteria and Major Equipment Selection for Grinding and Gravity Circuit

Description	Units	Design
Concentrator		
Grinding circuit	Type	SAG Mill and Pebble Crusher with Ball Mill closed with Cyclones (SABC)
SAG Mill		SAG Mill 19' Diam x 10.2' Length with 1.4MW Drive
Ball Mill		Ball Mill 15.5' Diam x 26.1' Length with 2.9MW Drive
Pebble Crusher Equipment		Metso Cone HP200
Feed F80	mm	160
Mill Circuit Product P80	um	75
Circulating Load - Design	%	275
Operating schedule	d/a	365
Daily treatment rate	t/d	2,740
Operating schedule	h/d	24
Operating time	%	92
Concentrator annual run hours	h/a	8,059
Concentrator Feed Rate - Nominal dry	dt/h	124
Primary Gravity circuit	location	Split of Cyclone Underflow
	Type	centrifugal
	Model	2 x Knelson
Gravity concentrate treatment	Type	Intensive Leach for Dore - Acacia Reactor
	location	Gold Room

Table 16-14: Summary Process Design Criteria and Major Equipment Selection for CIL Circuit

Description	Units	Design
CIL		
Rougher Tailings	Type	Flotation Tailings
Rougher Tailings Throughput	t/h	124.1
Au Grade - Gravity Tailings	Au g/t	0.26
Feed rate	t/a	1,000,000
Trash Screen	Type	Linear, 600x700µm, 5m^2
Total Leach/CIL Retention time	h	23.1
Leach dissolution	Au %	84
Leach tank	No	1
Tank Vol	m^3/tank	750
Tank Size	mD x mH	9.3mD x 11.6m H
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge

Description	Units	Design
CIL Tanks	No	6
Tank Vol	m^3/tank	750
Tank Size	mD x mH	9.3mD x 11.6m H
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge
Intertank screens	Type	Mech Swept
	Model	Kemix MPS (P) 500
Carbon concentration	g/L	12
Carbon loading	Au g/t	500
Carbon transfer	Type	Pump
Carbon Safety Screen	Type	Linear, 800x800µm, 5m^2
Elution circuit		
Loaded carbon screen	Type	Linear, 800x800µm, 0.5m^2
Elution	Type	Zadra
Operating Days	d/week	6
Column Size	t/strip	1.50
Acid Wash	Type	Separate column
Elution temp	°C	120
Electrowinning	Type of Cathode	Steel Wool
Smelting	Type	Smelt calcine
Carbon regeneration	Kiln Type	Horizontal
Carbon regeneration Kiln Size	kg/h	85

Table 16-15: Summary Process Design Criteria and Major Equipment Selection for Tailings Circuit

Description	Units	Design
Detox & Tailings		
Tailings Thickening	Type	High Rate
Tails thickener flux rate	t/m^2/h	0.50
Tails thickener underflow density	%	55
Tails thickener diameter	m	19
Cyanide detox	Type	INCO
Detox tank	No	1
Tank Vol	m^3/tank	400
Tank Size	mD x mH	9.0 x 11.3
Agitation	Type	Mech Open Impeller
Oxygen/Air Addition Method	Type	Sparge
WADCN	level ppm	10
Tailings Disposal	Type	Sub ariel multispigot

16.5 Process Reagents and Consumables

Process consumables for the plant for the three types of mineralised material are listed in Table 16-16 to Table 16-18. Table 16-16 lists the reagent and consumable consumption for the low grade Ore Type A (Au <1.5 g/t Zn <1.5%) material. Table 16-17 lists reagent and consumable consumption for the Type B material (Au≥1.5 g/t Au, Zn <1.5%). Table 16-18 lists the reagent and consumable consumption for the Type C (Au ≥1.5 g/t, Zn ≥1.5%) material where the process flow sheet involves gravity gold recovery, followed by Cu/Pb flotation (and cleaning), followed by Zn flotation (and cleaning) with the tails subject to FTL.

Table 16-16: Type A material reagent and consumable consumption

HUALILAN LG PROJECT - REAGENTS AND CONSUMABLES SUMMARY (Type A Material)								
		Treatment Regime at	**1,000,000**	t/a				
		Grinding Circuit Availability	**92**	%				
		Flotation Reagent Usage Proportion	**70**	%				
		Flotation Tailings Leach (FTL) Rate	**923,960**	t/a				
	Usage Rates (at 100% Concentration)							
	Usage Rates (at 100% Concentration or Equivalent)							
	Usage at Design Circuit Availability						Continuous Usage	
Reagent Description	Consumption		Annual	Daily	Hourly	Conc.	Daily	Hourly
	Rate	Unit	(t/annum)	(t/day)	(kg/h)	(%w/w)	(t/day)	(kg/h)
Grinding Media								
SAG Mill - 120 mm Balls	0.62	kg/t Ore	617	1.69	70	N/A	1.84	76
Ball Mill - 60 mm Balls	0.41	kg/t Ore	411	1.13	47	N/A	1.22	51
econdary Mill - MT1 Ceramic	0.45	kg/t Regrind	33	0.09	4	N/A	0.1	4
Flotation Collector 1 - PAX								
Rougher Flotation	63	g/t solids	63	0.17	7	100	0.188	7.8
Cleaner Flotation	0	g/t solids	0	0	0	100	0	0
Total Collector 1			63	0.173	7.2		0.188	7.8
Flotation Collector 2 - SIPX								
Rougher Flotation	28	g/t solids	28	0.08	3	100	0.083	3.5
Cleaner Flotation	84	g/t solids	84	0.23	10	100	0.25	10.4
Total Collector 2			112	0.307	12.8		0.334	13.9
Flotation Collector 3 - R208								
Rougher Flotation	35	g/t solids	35	0.1	4	100	0.104	4.3
Cleaner Flotation	42	g/t solids	42	0.12	5	100	0.125	5.2
Total Collector 3			77	0.211	8.8		0.229	9.6
Flotation Frother - MIBC								
Rougher Flotation	28	g/t solids	28	0.08	3	100	0.083	3.5
Cleaner Flotation	21	g/t solids	21	0.06	2	100	0.063	2.6
Total Frother			49	0.134	5.6		0.146	6.1
Flotation Activator - CuSO4								
Rougher Flotation	35	g/t solids	35	0.1	4	100	0.104	4.3
Cleaner Flotation	0	g/t solids	0	0	0	100	0	0
Detox	20	g/t solids stream	18.5	0.05	2.11	100	0.055	2.3
Total CuSO4			35	0.096	4		0.104	4.3
Flocculant								
Concentrate Thickener	50	g/t solids stream	1.2	0	0.14	100	0.004	0.2
Concentrate Filter	0	g/t solids stream	0	0	0	100	0	0
Preleach Thickener	20	g/t solids stream	18.5	0.05	2.11	100	0.055	2.3
Tailings Thickener	20	g/t solids stream	18.5	0.05	2.11	100	0.055	2.3
Total Flocculant			38.2	0.1	4.36	100	0.114	4.7

Table 16-17: Type B material reagent and consumable consumption

Treatment Regime at	1,000,000	t/a
Grinding Circuit Availability	92	%
Flotation Reagent Usage Proportion	70	%
Flotation Tailings Leach (FTL) Rate	923,960	t/a

	Usage Rates (at 100% Concentration)							
	Usage Rates (at 100% Concentration or Equivalent)							
	Usage at Design Circuit Availability						Continuous Usage	
Reagent Description	Consumption		Annual	Daily	Hourly	Conc.	Daily	Hourly
	Rate	Unit	(t/annum)	(t/day)	(kg/h)	(%w/w)	(t/day)	(kg/h)
Grinding Media								
SAG Mill - 120 mm Balls	0.62	kg/t Ore	617	1.69	70	N/A	1.84	76
Ball Mill - 60 mm Balls	0.41	kg/t Ore	411	1.13	47	N/A	1.22	51
Secondary Mill - MT1 Ceramic	0.45	kg/t Regrind	33	0.09	4	N/A	0.1	4
Flotation Collector 1 - PAX								
Rougher Flotation	63	g/t solids	63	0.17	7	100	0.188	7.8
Cleaner Flotation	0	g/t solids	0	0	0	100	0	0
Total Collector 1			63	0.173	7.2		0.188	7.8
Flotation Collector 2 - SIPX								
Rougher Flotation	28	g/t solids	28	0.08	3	100	0.083	3.5
Cleaner Flotation	84	g/t solids	84	0.23	10	100	0.25	10.4
Total Collector 2			112	0.307	12.8		0.334	13.9
Flotation Collector 3 - R208								
Rougher Flotation	35	g/t solids	35	0.1	4	100	0.104	4.3
Cleaner Flotation	42	g/t solids	42	0.12	5	100	0.125	5.2
Total Collector 3			77	0.211	8.8		0.229	9.6
Flotation Frother - MIBC								
Rougher Flotation	28	g/t solids	28	0.08	3	100	0.083	3.5
Cleaner Flotation	21	g/t solids	21	0.06	2	100	0.063	2.6
Total Frother			49	0.134	5.6		0.146	6.1
Flotation Activator - CuSO4								
Rougher Flotation	35	g/t solids	35	0.1	4	100	0.104	4.3
Cleaner Flotation	0	g/t solids	0	0	0	100	0	0
Detox	20	g/t solids stream	18.5	0.05	2.11	100	0.055	2.3
Total CuSO4			35	0.096	4		0.104	4.3
Flocculant								
Concentrate Thickener	50	g/t solids stream	1.2	0	0.14	100	0.004	0.2
Concentrate Filter	0	g/t solids stream	0	0	0	100	0	0
Preleach Thickener	20	g/t solids stream	18.5	0.05	2.11	100	0.055	2.3
Tailings Thickener	20	g/t solids stream	18.5	0.05	2.11	100	0.055	2.3
Total Flocculant			38.2	0.1	4.36	100	0.114	4.7
Lime - CIL & Detox								
CIL Leach	1,000	g/t solids stream	924	2.53	105.47	100	2.752	114.6
Detox	600	g/t solids stream	554.4	1.52	63.28	100	1.651	68.8
Total Lime			1478.3	4.1	168.8		4.402	183.4
Sodium Metabisulphite - Detox								
Detox	500	g/t solids stream	462	1.27	52.74	100	1.376	57.3
Total SMBS			462	1.3	52.7		1.376	57.3
Oxygen - CIL (Air injection)								
CIL Leach	0	g/t solids stream	0	0	0	100	0	0
Total O2			0	0	0		0	0
Activated Carbon - CIL								
CIL Leach	90	g/t solids stream	83.2	0.23	9.49	100	0.248	10.3
Total Carbon			83.2	0.2	9.5		0.248	10.3
LPG - CIL (Elution)								
CIL Leach	53	g/t solids stream	49	0.13	5.59	100	0.146	6.1
Total LPG			49	0.1	5.6		0.146	6.1
Sodium Hydroxide - CIL (Elution)								
CIL Leach	53	g/t solids stream	49	0.13	5.59	100	0.146	6.1
Total NaOH			49	0.1	5.6		0.146	6.1
Hydrochloric Acid - CIL (Elution)								
CIL Leach	53	g/t solids stream	49	0.13	5.59	100	0.146	6.1
Total HCl			49	0.1	5.6		0.146	6.1
Cyanide								
CIL Leach & Elution	500	g/t solids stream	462	1.27	52.74	100	1.376	57.3
Total NaCN			462	1.3	52.7		1.376	57.3

Table 16-18: Type C material reagent and consumable consumption

Stage 1 Treatment Regime at	1,000,000	t/a
Grinding Circuit Availability	92	%
Flotation Reagent Portion of Usage	70	%
Flotation Tailings Leach (FTL) Rate	879,890	t/a

	Usage Rates (at 100% Concentration)							
	Usage Rates (at 100% Concentration or Equivalent)							
	Usage at Design Circuit Availability						Continuous Usage	
Reagent Description	Consumption		Annual	Daily	Hourly	Conc.	Daily	Hourly
	Rate	Unit	(t/annum)	(t/day)	(kg/h)	(%w/w)	(t/day)	(kg/h)
Grinding Media								
SAG Mill - 120 mm Balls	0.54	kg/t Ore	537	1.47	61	N/A	1.6	67
Ball Mill - 60 mm Balls	0.36	kg/t Ore	358	0.98	41	N/A	1.07	44
Secondary Mill - MT1 Ceramic	0.6	kg/t Regrind	80	0.22	9	N/A	0.24	10
Flotation Collector 1 - SIPX								
Rougher Flotation	60	g/t solids	60	0.16	7	100	0.177	7.4
Cleaner Flotation	25	g/t solids	25	0.07	3	100	0.073	3
Total Collector 1			84	0.23	9.6		0.25	10.4
Flotation Collector 2 - R208								
Rougher Flotation	4	g/t solids	4	0.01	0	100	0.01	0.4
Cleaner Flotation	0	g/t solids	0	0	0	100	0	0
Total Collector 2			3.5	0.01	0.4		0.01	0.4
Flotation Frother - MIBC								
Rougher Flotation	46	g/t solids	46	0.12	5	100	0.135	5.6
Cleaner Flotation	35	g/t solids	35	0.1	4	100	0.104	4.3
Total Frother			80.5	0.221	9.2		0.24	10
Flotation Depressant 1 & CIL - NaCN								
Rougher Flotation	105	g/t solids	105	0.29	12	100	0.313	13
Cleaner Flotation	0	g/t solids	0	0	0	100	0	0
CIL Leach & Elution	500	g/t solids stream	439.9	1.21	50.22	100	1.31	54.6
Total NaCN			544.9	0.288	12		0.313	13
Flotation Depressant 2 - ZnSO4-								
Rougher Flotation	490	g/t solids	490	1.34	56	100	1.459	60.8
Cleaner Flotation	0	g/t solids	0	0	0	100	0	0
Total ZnSO4-			490	1.342	55.9		1.459	60.8
Flotation pH Modifier - Soda Ash								
Rougher Flotation	0	g/t solids	0	0	0	100	0	0
Cleaner Flotation	2,185	g/t solids	2,185	5.99	249	100	6.506	271.1
Total ZnSO4-			2184.7	5.985	249.4		6.506	271.1
Flotation Activator & Detox- CuSO4								
Rougher Flotation	700	g/t solids	700	1.92	80	100	2.085	86.9
Cleaner Flotation	0	g/t solids	0	0	0	100	0	0
Detox	20	g/t solids stream	17.6	0.05	2.01	100	0.052	2.2
Total CuSO4			717.6	1.918	79.9		2.085	86.9
Flocculant								
Concentrate Thickener Cu/Pb	50	g/t solids stream	0.3	0	0.03	100	0.001	0
Concentrate Thickener Zn	50	g/t solids stream	3.5	0.01	0.4	100	0.01	0.4
Concentrate Thickener PM	50	g/t solids stream	2.3	0.01	0.26	100	0.007	0.3
Concentrate Filter Cu/Pb	0	g/t solids stream	0	0	0	100	0	0
Concentrate Filter Zn	0	g/t solids stream	0	0	0	100	0	0
Preleach Thickener	20	g/t solids stream	17.6	0.05	2.01	100	0.052	2.2
Tailings Thickener	20	g/t solids stream	17.6	0.05	2.01	100	0.052	2.2
Total Flocculant			41.3	0.11	4.71	100	0.123	5.1
Lime - Depressant, CIL & Detox								
Rougher Flotation	1,218	g/t solids	1,218	3.34	139	100	3.627	151.1
Cleaner Flotation	157	g/t solids	157	0.43	18	100	0.467	19.5
CIL Leach	1,000	g/t solids stream	879.9	2.41	100.44	100	2.62	109.2
Detox	600	g/t solids stream	527.9	1.45	60.27	100	1.572	65.5
Total Lime			2782.6	7.6	317.7		8.287	345.3
Sodium Metabisulphite - Detox								
Detox	500	g/t solids stream	439.9	1.21	50.22	100	1.31	54.6
Total SMBS			439.9	1.2	50.2		1.31	54.6
Oxygen - CIL								
CIL Leach	0	g/t solids stream	0	0	0	100	0	0
Total O2			0	0	0		0	0
Activated Carbon - CIL								
CIL Leach	90	g/t solids stream	79.2	0.22	9.04	100	0.236	9.8
Total Carbon			79.2	0.2	9		0.236	9.8
LPG - CIL (Elution)								
CIL Leach	100	g/t solids stream	88	0.24	10.04	100	0.262	10.9
Total LPG			88	0.2	10		0.262	10.9
Sodium Hydroxide - CIL (Elution)								
CIL Leach	100	g/t solids stream	88	0.24	10.04	100	0.262	10.9
Total NaOH			88	0.2	10		0.262	10.9
Hydrochloric Acid - CIL (Elution)								
CIL Leach	100	g/t solids stream	88	0.24	10.04	100	0.262	10.9
Total HCl			88	0.2	10		0.262	10.9

16.6 Water Consumption

The water balance for the plant processing the Type C material is shown in Figure 16-4. In steady state operation the process plant will consume 0.77 cubic metres of water per ton of material processed.

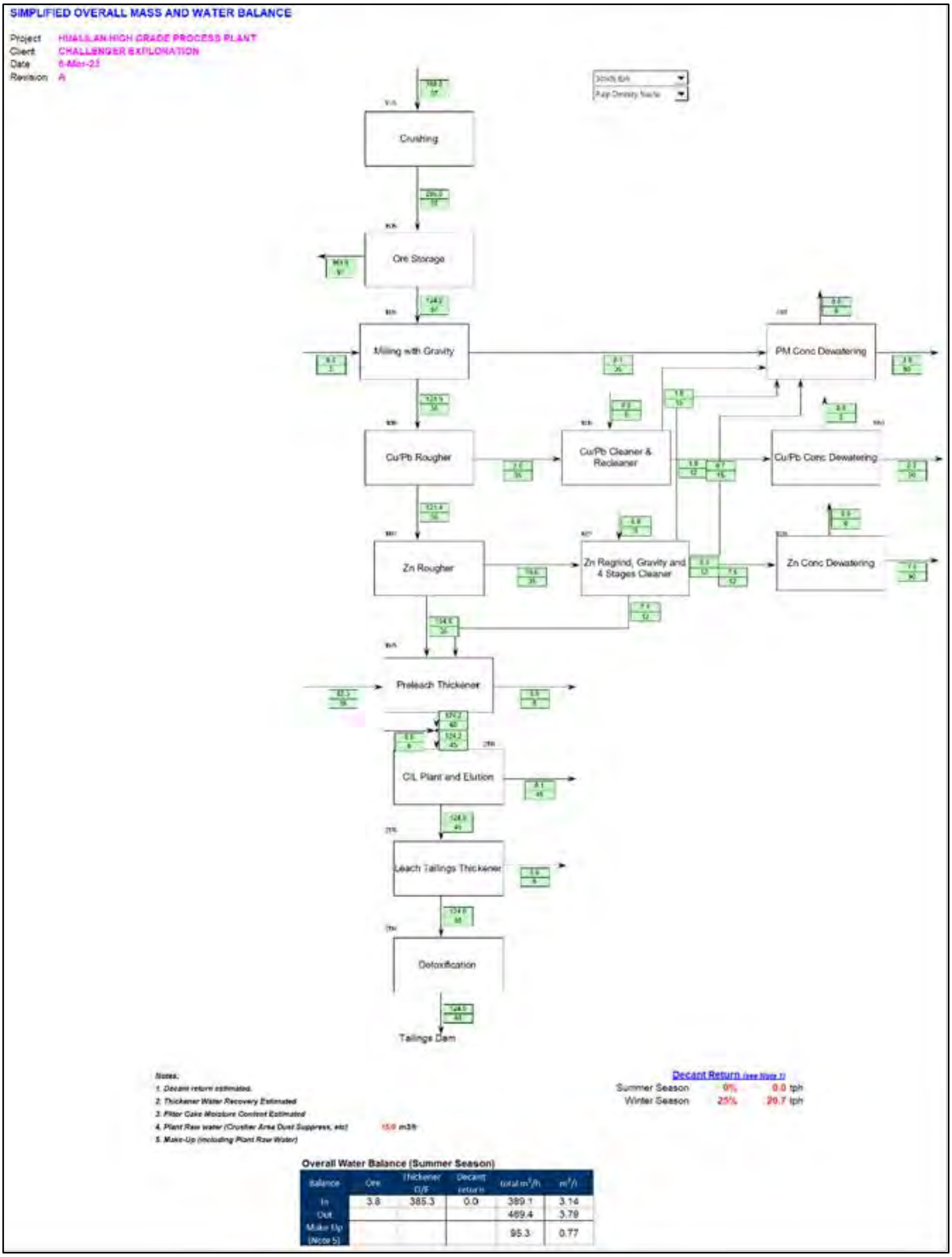


Figure 16-4: Water balance for the plant processing the Type C material.

16.7 Power Consumption

The power consumption for the comminution circuit has been calculated using vendor specification for the components of the comminution circuit and expected availabilities. Power usage of 15 million kWh per annum in the comminution circuit was estimated based on vendor specification of the components of the circuit and expected availabilities. An allowance of an additional 15 million kWh per annum was made for power usage in the remainder of the process plant based on benchmarking of similar operations in South America.

17 TAILINGS REQUIREMENT AND MANAGEMENT

17.1 Tailings Storage Facility

17.1.1 Standards and Guidelines

The following standards and design guidelines were used in the preparation of the Tailings Storage Facility (TSF) concept for the Scoping Study for the Hualilan Project:

- Global Industry Standard for Tailings Management (GISTM) August 2020.
- ANCOLD "Guidelines on Tailings Dams - Planning, Design, Construction, Operation and Closure", Revision 1 dated July 2019.
- ICOLD Bulletin 153 "Sustainable Design and Post-Closure Performance of Tailings Dams", dated 2013.

17.1.2 Hazard Category Assessment

A preliminary hazard category assessment was also undertaken for the TSF using the criteria provided in Table 1 Consequence Classification Matrix of GISTM (Global Tailings Review, 2020) and Tables 1 and 2 of (ANCOLD, Revision 1, July 2019) Guidelines. In accordance with the GISTM, the potential population at risk (PPAR) is estimated to be between 1 and 10, with the actual potential loss of life unspecified at the moment. Preliminary evaluation of the GISTM Dam Failure Consequence Classification (DFCC) indicates the TSF would have a classification of Significant.

A preliminary hazard category assessment was undertaken for the TSF using the criteria provided in Tables 1 and 2 of the ANCOLD Guidelines2, presented as Table 17-1 and Table 17-2, respectively, in this document. In accordance with ANCOLD Guidelines, the TSF has a ***preliminary*** Consequence Category Assessment (CCA) of Significant.

The severity of damage and loss was assessed as Medium, based on health, social and environmental impacts associated with a potential failure of the facility. The population at risk (PAR) is estimated to be >1 and ≤10.

A detailed analysis should be executed in the next stage of project studies to determine the actual DFCC (GISTM) and CCA (ANCOLD).

Table 17-1: Severity Level impacts (summary from ANCOLD Consequence Guidelines 2019)

Damage Type	Minor	Medium	Major	Catastrophic
Infrastructure (dam, houses, commerce, farms, community)	<$10M	$10M-$100M	$100M-$1B	≥$1B

Damage Type	Minor	Medium	Major	Catastrophic
Business importance	Some restrictions	Significant impacts	Severe to crippling	Business dissolution, bankruptcy
Public health	<100 people affected	100-1000 people affected	<1000 people affected for more than one month	≥10,000 people affected for over one year
Social dislocation	<100 person or <20 business months	100-1000 person months or 20-2000 business months	≥1000 person months or ≥200 business months	≥10,000 person months or numerous business failures
Impact Area	<1 km2	<5km^2	<20km^2	≥ 20km^2
Impact Duration	<1 (wet) year	<5 years	<20 years	≥ 20 years
Impact on natural environment	Damage limited to items of low conservation value (e.g. degraded or cleared land, ephemeral streams, non-endangered flora and fauna). Remediation possible.	Significant effects on rural land and local flora & fauna. Limited effects on: A. Item(s) of local & state natural heritage. B. Native flora and fauna within forestry, aquatic and conservation reserves, or recognised habitat corridors, wetlands or fish breeding areas.	Extensive rural effects. Significant effects on river system and areas A & B. Limited effects on: C. Item(s) of National or World natural heritage. D. Native flora and fauna within national parks, recognised wilderness areas, RAMSAR wetlands and nationally protected aquatic reserves. Remediation difficult	Extensively affects areas A & B. Significantly affects areas C & D. Remediation involves significantly altered ecosystems.

Table 17-2: Recommended Consequence Category (from ANCOLD 2019)

Population At Risk (Par)	Severity Of Damage or Loss			
	Minor	Medium	Major	Catastrophic
<1	Very Low	Low	Significant	High C
>1 to 10	Significant	Significant	High C	High B
>10 to 100	High C	High C	High B	High A
>100 to 1000		High B	High A	Extreme
>1000			Extreme	Extreme

17.1.3 Types of Tailings Facility

A desktop options study was completed to assess the advantages and disadvantages of all possible types of tailings storage, with the objective of assessing which style of construction minimised the overall risk and viability of the project.

This desktop study produced only two possible TSF options which can reasonably be considered as viable given:

i) The site characteristics of high seismicity and challenging topography, with the natural ground having a fall of approximately 1 in 26 to the south and southeast.
ii) The geochemistry of the tailings, which is likely to be adverse, potentially acid-forming (PAF) and slightly enriched with some heavy metals, such that the TSF will be required to be HDPE-lined.
iii) The available materials for construction of containment embankments would comprise mine waste, supplemented with borrow from in-situ materials, as required.
iv) Constructability of the TSF.

These options are:

i) An integrated waste landform tailings storage facility (IWLTSF), which utilises the topography and available mine waste to create a robust structure which totally surrounds and encapsulates the tailings within the HDPE-lined facility.
ii) A side-hill storage which utilises the natural topography and is constructed from a combination of mine waste and/or borrow from in-situ materials which totally surrounds and encapsulates the tailings within the HDPE-lined facility. The side-hill storage has robust embankments on the downstream slope and sides.

The option selected for the design is the side-hill storage, which is easier to construct given the need for HDPE liner placement.

17.1.4 Location for the Tailings Storage Facility

The site for the TSF is within the lease boundaries and is constrained by topography, project infrastructure layout and the need for the facility to be within close proximity to the process plant. The topographic nature of the selected site is such that the above-ground storage will form an enclosed side-hill storage facility.

17.1.5 Sterilisation Drilling Requirement

Sterilisation drilling is yet to be executed over the selected TSF site.

17.1.6 Site Characteristics

17.1.6.1 Climate

The project is located in the San Juan region of Argentina, where the summers are hot, the winters are cold, and it is dry, with average annual precipitation in the range of 106 to 128 mm and average annual evaporation in the range of 1,540 to 1,932 mm per year. The annual temperature range is from 2.9 to 42.5 °C.

17.1.6.2 Geology and Soils

The surficial geology of the site comprises colluvial materials, derived from the weathering and erosion of the hills to the west of the TSF site. There is a colluvial/alluvial deposit in the drainage line to the north of the TSF location.

The colluvial materials typically comprise a mix of gravel, sand and with some silt and clay. The granular materials closer to the ridgeline which defines the western edge of the site, will likely have predominantly coarse granular materials (cobbles and gravel) with the finer fraction further to the east away from the ridge.

17.1.7 Geotechnical Assessment

There have been no specific geotechnical site investigations undertaken for the proposed TSF at this stage. Similarly, there has been no specific geotechnical testing of the tailings at this stage.

17.2 Tailings Storage Facility Design Operation and Management

17.2.1 Design Parameters

The assumed project design parameters are as follows:

- Total tailings production – 7.07 Mtpa
- Total tailings production discharged to the TSF – 4.2 Mt
- Total tailings production to paste backfill – 2.87 Mt
- TSF Life – 7 years
- Project Life – 8 years
- Slurry density – 50% solids

- Deposited in-situ of the tailings – 1.5 t/m^3 (assumed based on the TSF being well-managed, with very high-water recovery)
- Supernatant water recovery target (min) – 70%
- Operating hours - 7,900 hrs/yr

17.2.2 Design Concept

The perimeter embankment of the TSF will comprise a starter embankment with a compacted clayey material (oxide mine waste or screened in-situ materials) forming the upstream zone to accept the HDPE liner, which will be placed against the adjacent waste rock storage. The clay materials will be sourced from the areas which may be covered by the waste rock storage, and other areas to be developed as part of the project.

Materials sourced from mine waste from the open pit, from the development of the underground mine and external borrow areas, will be utilised for construction of the perimeter embankments. Geotechnical investigation work is yet to be executed to source, assess and evaluate the available materials and borrow pit locations.

The TSF is double HDPE-lined with an underdrainage connected to an external sump and a leak detection system between the liners on the floor and Stage 1 upstream embankment. A single HDPE liner is to be placed on the Stage 2 upstream embankment.

A rock filter decant is be constructed within the facility to enable supernatant water to be recovered and returned to the process plant for re-use if the water is suitable. A decant accessway will connect the decant to the perimeter embankment. Essentially, the TSF is designed to dry the tailings and allow supernatant water to be recovered for reuse in the process plant.

Tailings produced will be stored in a single cell TSF, with Stage 1 having a crest elevation of RL 1,750 m providing storage capacity for 2 years and Stage 2 having a crest elevation of RL 1,757 m providing storage capacity for 9 years. Tailings storage requirements are provided in the production schedule (Table 17-4).

Embankment raising from Stage 1 to Stage 2 is by downstream construction. Drawing 01 is the general arrangement plan for the Final Stage and Drawing 02 shows the Stage 1 Plan with typical embankment sections and details shown on Drawings 03 to 07. Table 17-3 presents the details (Drawing No. and Title) for each of the drawings and the drawings themselves are in Appendix 8 – TSF Drawings.

Table 17-3: Drawing details

Drawing No.	Title
01	TSF - Final Stage Plan
02	TSF - Stage 1 Plan
03	Sections and Details - Embankments and Underdrainage
04	Sections and Details - Underdrainage
05	Sections and Details - Underdrainage and Leak Detection
06	Sections and Details – Pipework and Underdrainage Sump
07	Sections and Details – Instrumentation

17.2.3 TSF Construction and Materials

TSF construction comprises a downstream zone which will be constructed using mine waste sourced from the proposed mixed open pit and underground operation and in-situ materials, if suitable, as required. Current indications are that adequate volumes of waste are available for the current LOM, refer to Table 17-4. It should be noted that minor cut-to-fill has been used in the design to facilitate the placement of the clay liner in the floor of the TSF Stage 1 footprint.

Table 17-4: Mineralised Material and Waste Schedule for Mixed Option of Small Open Pit and Underground

Source	Unit	Parameter Value	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Total Ore Tonnes	t		0	1,060,000	1,060,000	1,056,000	1,056,000	1,056,000	1,032,000	748,300		
Tails Tonnes in Paste fill	t			58,125	496,169	494,297	494,297	494,297	483,063	350,267		
Tails Tonnes to TSF	t			1,001,875	563,831	561,703	561,703	561,703	548,937	398,033		
Total Tailings Volume	m^3 at 1.5 t/m^3	1.5	0	667,917	375,887	374,469	374,469	374,469	365,958	265,355		
Cumulative Tailings Volume	m^3		0	667,917	1,043,804	1,418,273	1,792,741	2,167,210	2,533,168	2,798,523		
Total Open Pit Waste Tonnes	t		3,070,232	2,047,843	2,249,965	2,098,777	49,562	0	0	0		
Total UG Waste Tonnes	t		375,622	478,762	442,030	431,980	198,809	127,774				
Mine Waste Volume	m^3 at 2.1 t/m^3	2.1	1,640,883	1,203,145	1,281,902	1,205,123	118,272	60,845				
Mine Waste Volume (Oxide)	m^3 at 1.8 t/m^3	1.8										
ROM Pad (300 x 300 x 5)	t	810,000	450,000									
Roads	t	699,300	333,000	120,000								
Diversion Drains	t	64,108	35,616									
Mine Waste Volume available for TSF Construction	m^3		822,267	1,083,145	1,281,902							
Required Construction Volumes – Oxide Mine Waste	m^3		112,600	85,000	85,000							
Required Construction Volumes – Mine Waste (Transition/Fresh)	m^3		686,300	713,900	713,900							

Source	Unit	Parameter Value	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Mine Waste to TSF Construction per year	m^3		798,900	798,900	798,900							
Mine Waste Volume Available after TSF Requirements are met	m^3		23,367	284,245	483,002							
Cumulative Mine Waste Volume Available after TSF Requirements are met	m^3		23,367	307,612	790,615	1,995,738	2,114,009	2,174,854				

The maximum particle size for the downstream zone is 500 mm. The construction methodology for downstream embankment shall be as follows:

i. Spread a loose lift of Rock Fill Material with a maximum thickness not exceeding 750 mm.
ii. The Rock Fill Material shall be compacted using smooth drum vibrating rollers of not less than 11 tonnes front module mass in layers of loose lift thickness not exceeding 750 mm. Where the front module mass is less than 11 tonnes, the loose lift thickness of the Rock Fill shall be reduced.

The mine waste in the downstream zone will be supplemented with either moisture-conditioned oxide mine waste, or finer materials which are screened-moisture-conditioned and compacted to form a relatively smooth surface, free of projections on the upstream face of the embankment, suitable for placement of the HDPE liner. These materials are expected to be sourced locally and screening is likely to remove oversized materials to achieve the following requirements for placement of the HDPE liner, without a geotextile underlay.

- The materials must have 100% by weight passing 19 mm.
- The materials must have a minimum of 30% by weight passing 0.075 mm.
- The material must be free of organic or any other deleterious inclusions.
- The material must have a liquid limit of not less than 20%.
- The material must have a plasticity index of more than 8%.

After screening, the materials would be transported to the site of the TSF and placed in such a manner as to minimise segregation. The materials are to be moisture-conditioned and cured, to ensure the moisture is thoroughly mixed and evenly spread through all materials proposed for embankment construction, to within the range of ±2% of the optimum moisture content, as determined from laboratory testing in accordance with the latest version of ASTM D698.

All fill material destined for the upstream perimeter embankment is to be placed in homogeneous horizontal layers not exceeding 300 mm loose lift thickness. Each layer shall be compacted to achieve a minimum density ratio of not less than 96% of the maximum dry density, Standard Proctor (Standard Compaction) as determined from laboratory test ASTM D698. Placement should be continuous. If a break in fill placement allows the exposed surface to dry, it should be lightly tyned, watered and compacted prior to fill placement recommencing. The upstream batter upon which the HDPE liner will be founded shall be smooth and free of projections (e.g. cobbles, roots etc) that could damage the HDPE liner. Field density testing will be executed to ensure compliance with the specification and QA/QC procedures.

With the tailings likely being classified as PAF and the process water likely having low salinity, there is minimal potential for impacting the natural groundwater from the storage of tailings, given the inclusion of a double HDPE liner, with an underdrainage system to aid with water recover from the base of the deposited tailings and surface decant to collect supernatant water for reuse in processing operations.

With mitigation measures described above (double HDPE liner, underdrainage system and decant water system geared for high rates of water return from the supernatant pond through an appropriately-sized water recovery system), potential for seepage from the facility should be very low.

*17.2.3.1 **Embankment Geometry***

The proposed TSF design concept is based on downstream construction techniques with both upstream and downstream slopes of 3.0:1 (H:V). The perimeter embankments are to comprise an engineered, compacted 'select' soil embankment with a cut-off trench on the upstream face, with a minimum crest width of 8 m. The downstream section of the embankment will have a minimum crest width of 15 m and comprise traffic compacted mine waste.

*17.2.3.2 **Earthworks***

The upstream embankment of the TSF will comprise a starter embankment constructed using compacted, 'select' low-permeability material, either oxide mine waste or borrow materials, sourced from the borrow areas, including areas which might be covered by the waste rock dumps, and other areas to be developed as part of the project and processed to meet the specified construction requirements. The downstream section of the embankment will comprise traffic compacted mine waste. The materials for the embankment construction and the depth of the cut-off, will need to be confirmed in future geotechnical investigations.

Construction of the TSF is to be undertaken in accordance with the Construction Specification for the facility, developed as part of the following phases of design works to cover both earthworks construction and liner installation. Typical construction requirements for the TSF implementation include the following:

i) Inspection of the decant base.
ii) Inspection of pipework beneath the perimeter embankment comprising decant and underdrainage pipework.
iii) Inspection of the seepage cut off and the alignment of the embankment.
iv) Inspection of the compacted base of the TSF prior to liner construction.
v) Inspection of liners and Flownet during construction.
vi) Inspection of underdrainage, including pipework and outfall sump.
vii) Compliance testing of seepage cut off and perimeter embankment to ensure:

viii) Moisture curing of materials as required during the upstream embankment construction with content at the time of placement and compaction within ±2% of the optimum moisture content. Compaction is to achieve a density ratio greater than 95% of standard maximum dry density.

ix) Materials used in the upstream embankment construction shall comprise clayey material having a fines content (material finer than 75 microns) in excess of 30% for the compacted clay liner.

x) Mine waste materials used in the downstream embankment shall be spread with a maximum loose thickness not exceeding 750 mm. This material shall be compacted using smooth drum vibrating rollers of not less than 11 tonnes front module mass, in layers of loose lift thickness not exceeding 750 mm.

In accordance with (ANCOLD, Revision 1, July 2019), construction supervision for the TSF is to be undertaken on a full-time basis by an engineer who is suitably qualified in the design and construction of TSFs. ANCOLD (2019) recommends that the designer provides or coordinates the construction supervision and construction reporting.

Laboratory and field testing is to be implemented to ensure the construction materials, including the liner, are selected, placed, and in the case of the earthworks materials, compacted in accordance with the intent of the design. Test types and frequencies are to be addressed in the construction specifications. In addition to the specifications, a QA/QC Manual has to be developed for use in the construction. The main objectives for the implementation of the QA/QC Manual are to:

- Ensure that construction work adheres strictly to all requirements as defined in the technical specifications, supporting construction design documents, codes and standards, and regulatory requirements.
- Maintain quality and provide documentation to ensure that tasks performed will comply with project requirements.
- Prevent construction deficiencies through pre-construction quality coordination.
- Ensure non-conformances are reported and resolved through approved dispositions and follow-up actions to confirm acceptable results during the construction process.
- Provide auditable records of all tests, inspections, noncompliance and corrections, and any other pertinent data as required.
- Ensure that construction subcontractors have appropriate quality systems that will ensure work activities will meet contractual, technical and quality requirements.
- Comply with project requirements and specifications, through the use of project approved Quality plans, construction control procedures and Inspection and Test Plans.

- Ensure that the extent of inspections, verifications and assessments is commensurate with the risk, safety, cost, schedule and quality issues that may adversely impact the quality of the facility.
- Address and coordinate resolution of quality issues identified during reviews, surveillance, audits and inspections.

17.2.3.3 ***Liner Details***

The design concept incorporates a double HDPE-lined floor of the TSF and upstream embankments to the Stage 1 crest elevation. A single HDPE liner is to be placed on the upstream embankments to the Stage 2 crest elevation.

The basal liner 1.5 mm comprises a smooth HDPE liner, which is to be placed over a layer of compacted oxide mine waste or compacted in-situ low-permeability subgrade, free of gravel, cobbles, boulders, with minimum compacted thickness of 300 mm.

There have been no geotechnical investigations executed to investigate and confirm the geotechnical suitability and use of the oxide mine waste and in-situ materials as sources of suitable, low-permeability material, at this stage.

The HDPE liner requires stringent surface preparation of the underlying subgrade or deployment of a geotextile underlay where suitable material is not available. Extreme care is required during handling, placement and installation of the HDPE to avoid damage. The selected materials for the subgrade in the base of the TSF is also to be installed on the upstream embankment slopes of Stage 1 and 2.

The leak detection system comprising a Flownet, is to be located between the first HDPE liner placed over the compacted base and the upper HDPE liner up to the Stage 1 crest elevation.

17.2.3.4 ***Water Recovery***

Water recovery systems comprise an underdrainage over the upper HDPE liner and a pontoon-mounted pumped decant located within the rock filter decant.

The underdrainage is designed to recover water from the base of the deposited tailings stack and assist with consolidation of the tailings.

The presence of the underdrainage, coupled with the practically impermeable HDPE liner, underlying the underdrainage, will effectively limit vertical seepage from the TSF. Seepage losses from the TSF are expected to be in accordance with the action leakage rate (ALR) developed in accordance with the liner performance method. The actual requirements can be assessed during the next study phase once the geotechnical investigation works are completed.

17.2.4 Operation Plan

17.2.4.1 Tailings Deposition

Tailings will be deposited using sub-aerial deposition techniques from multi-spigot locations on the perimeter deposition embankments. Tailings spigotting or deposition is to be executed in thin layers of not more than 300 mm, to ensure a uniform tailings beach with a fall of 1% towards the decant is developed. The decant facility will be located on the eastern side of the facility. The spigotting sequence is to be formulated such that the supernatant water pond is always maintained around a decant structure.

Tailings deposition would occur from the perimeter embankment with spigot intervals of not less than 25 m and not more than 50 m. Conductor pipes laid on old conveyor belt pieces, similar to the style of conductor pipe shown in Figure 17-1 and Figure 17-2 below, are recommended to ensure that tailings discharged from the perimeter are deposited onto the tailings beach with minimal potential for erosion of the adjacent embankment. The diameter of the conductor pipe has to be designed to suit the size of the tailings distribution pipeline and the conveyor matting, to be placed down the embankment, also has to be adjusted to suit the size of the conductor pipe. Some operations may require not less than three pieces of used conveyor matting in the vicinity of the conductor pipe, to provide sufficient erosion protection to the adjacent embankment.

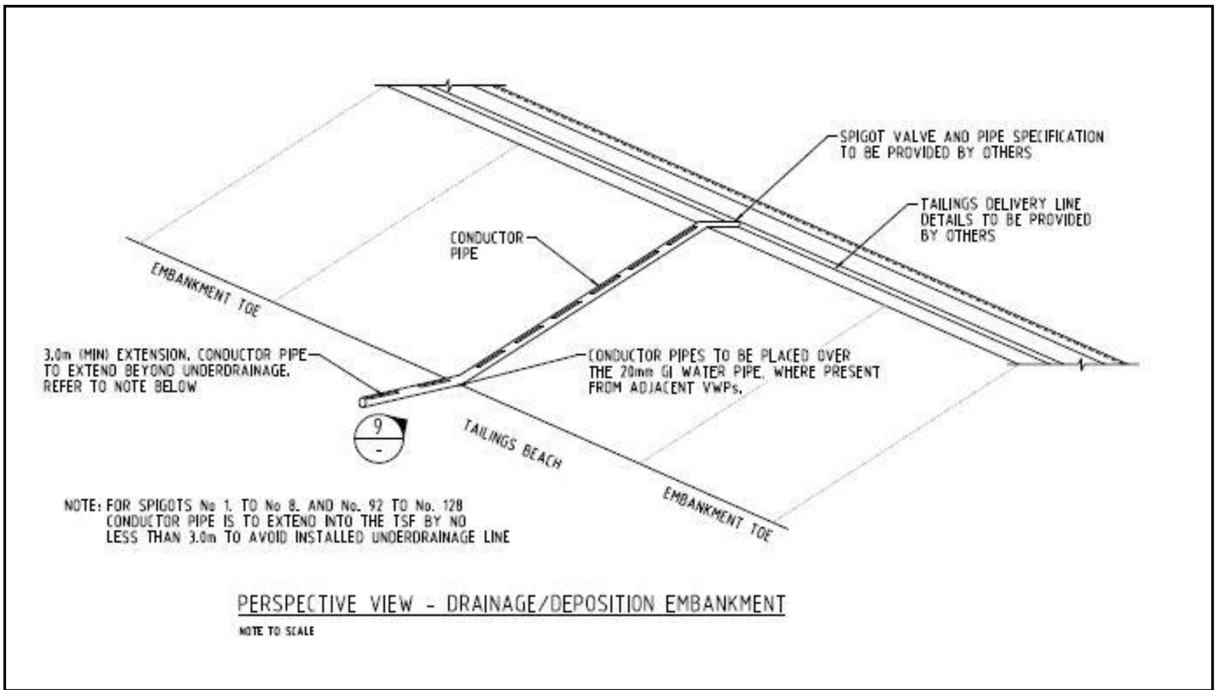


Figure 17-1: Conductor Pipe Detail.

Figure 17-2: Conductor Pipe Detail on HDPE-lined TSF.

17.2.4.2 **Decant System and Water Management**

A decant is to be installed in the centre of the TSF to facilitate surface water recovery. The decant system comprises a rock ring filter. Access to the actual decant structure for light vehicles and maintenance equipment will be via a decant access roadway, constructed from either in-situ materials or mine waste.

The decant is to be raised in conjunction with the raising of the perimeter embankments. Figure 17-3 shows a typical rock ring filter. The benefits of implementing a well-formed rock ring filter decant arrangement are as follows:

- The clarity of the water within the rock ring is a function of the thickness of the filter walls. Typically, the perimeter of the rock ring has a crest width of not less than 5 m excluding windrows.
- The rock ring is an efficient filter system, where the surface area of the rock (width and diameter within the ring) forms circuitous flow paths, which reduce the 'flow velocities' through the filter wall allowing the fines (silt and clay sized fraction) to settle out.
- The presence of the rock does not allow turbulence (wave action) to propagate through the filter wall and the area within the ring is sufficiently small to minimise the potential risk of wave action forming within the rock ring.
- The rock ring filter detracts birds from landing in the pond on the TSF.

Figure 17-3: Decant Rock Filter.

The supernatant pond level must be actively maintained, so as to be kept around the decant area and clear of the perimeter embankments. At no time should the supernatant pond be allowed to encroach within 100 m of the external engineered embankments, irrespective of the fact that a liner is being used on the upstream embankment slope.

The position and extent of the supernatant pond is controlled by the water recovery, which is to be maximised. The decant pumping system must be capable of recovering additional water during the wet season.

Whilst the TSF is designed as a "no-discharge facility", under standard operating conditions in accordance with ANCOLD (2019) Guidelines2, the final stage of the design will need to consider an emergency spillway, which provides a controlled discharge point following an extreme rainfall event which exceeds the available storage capacity within the TSF if it remains as an above ground facility. The emergency spillway needs to be designed to effectively discharge water captured within the TSF, equivalent to the rainfall intensity of the PMP, 24-hour event. It is appreciated that in some jurisdictions e.g. Western Australia, the regulator (DMIRS) prefers above-ground TSFs not to have spillways and for the upper surface to be designed to retain the closure design storm event.

Given the likely geochemical nature of the stored tailings, it is considered, at this stage, that the cover for the TSF must be a water-shedding structure.

17.2.4.3 ***Water Balance***

A preliminary water balance analysis was prepared using an excel spreadsheet based on average tailings deposition of 1.0 Mtpa. The spreadsheet calculates an estimation of the inflows and outflows from the TSF and determines the balance after plant water requirements have been met. Water shortfall or water in excess of requirements is indicated on a monthly and annual basis.

Water inflows to the tailings dam system consist of rainfall (runoff from the catchment surrounding the dam and incident rainfall on the impoundment area) and slurry water from the plant. Water outflows consist of evaporation from the supernatant pond and running beaches, evapo-transpiration from drying beaches, seepage, retention of water within tailings and water returned to the plant.

The following information was used for the preliminary site water balance:

- Average annual rainfall of 115 mm.
- Average annual evaporation of 1,735 mm.

The following assumptions were made for the water balance:

- Runoff co-efficient of 0.45 from the surface of the tailings.
- Decant pond to be maintained with a maximum radius of 25 m.
- Spigotting and running beaches would have a maximum area of 10,000 m^2.
- The maximum surface area of the TSF, 32 ha, was used for this water balance.
- Decant water recovery system must be designed for not less than minimum water recovery of 70% of the slurry water to achieve the minimum design in-situ dry density of 1.5 t/m^3.

Under average rainfall and evaporation, the preliminary water balance indicates the annual water return for the TSF in Year 1, where the tailings deposition rate is 942,000 tpa, the water return needs to be around 710,000 tpa, approximately 90 tph for 7,900 operating hours per annum, to maintain the water balance.

For Years 2 to 7 where the average tailings deposition rate is around 550,000 tpa, the water return needs to be around 415,000 tpa, approximately 52 tph for 7,900 operating hours per annum, to maintain the water balance.

Figure 17-4 and Figure 17-5 show the relationship between dry density and water recovery as relevant to this TSF design.

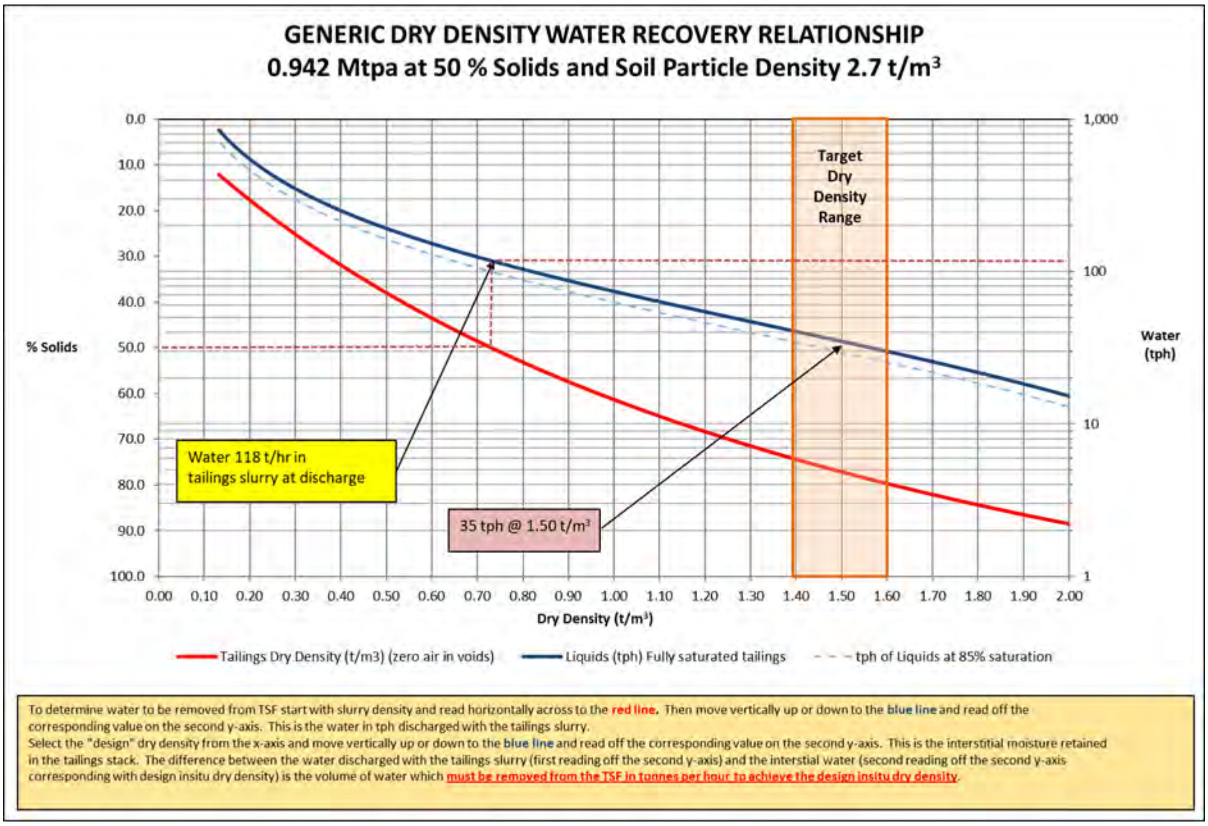


Figure 17-4: Generic Dry Density and Water Recovery Relationship at 0.942 Mtpa

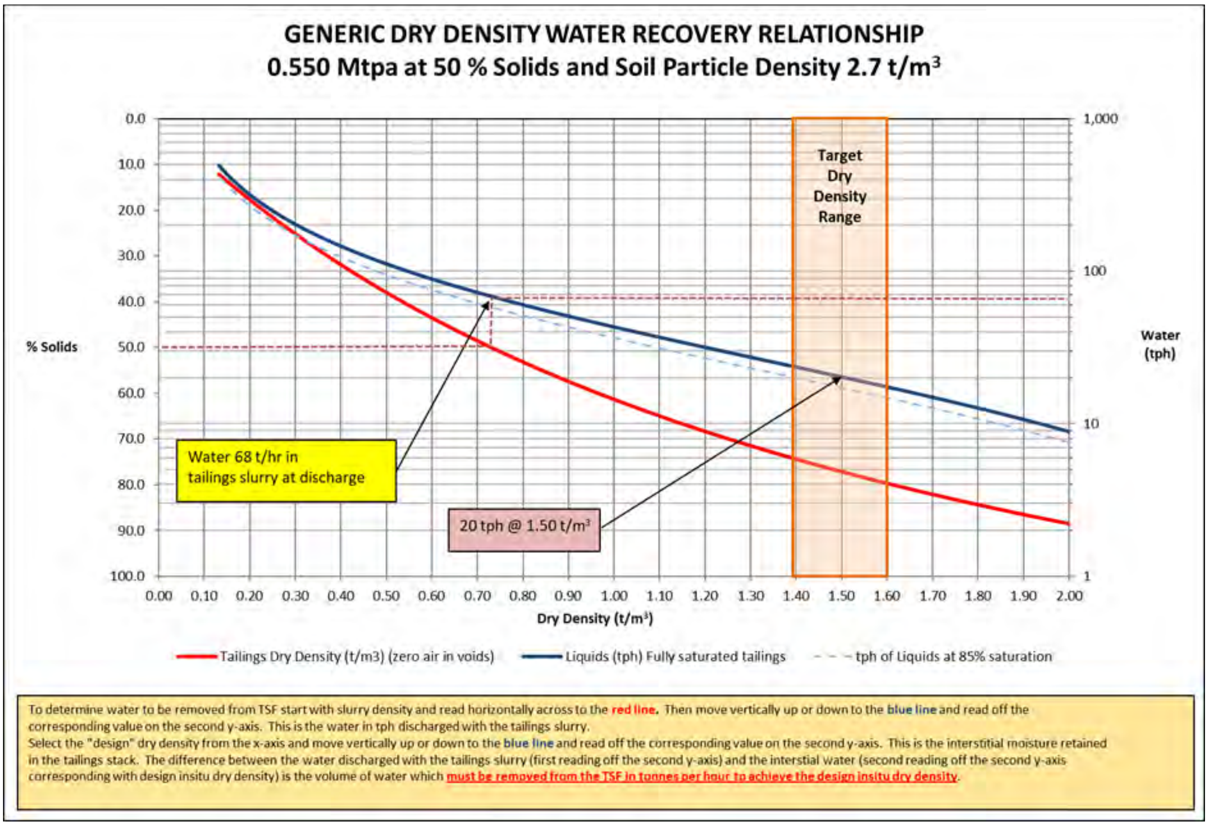


Figure 17-5: Generic Dry Density and Water Recovery Relationship at 0.550 Mtpa

17.2.4.4 Monitoring and Instrumentation

Groundwater monitoring bores are to be installed in the area downstream of the TSF to enable any deep-seated seepage beneath the perimeter embankment to be identified.

In addition to monitoring for seepage from the TSF, a series of monitoring points, comprising standpipe piezometers and Vibrating Wire Piezometers (VWP) will need to be developed internally within the TSF, below the HDPE-liner to check for seepage and enable monitoring of the phreatic surface and of pore pressures within the deposited tailings. This information will be required for evaluation of future embankment raises.

VWPs are to be installed as part of the starter embankment construction. Allowance needs to be made for a total of six VWPs. The VWPs are to be concentrated nominally 30 m and 50 m inside the facility from the upstream toe of the highest containment embankment, with the read-out cables extended up the perimeter embankment slopes to the starter embankment crest levels. Extending the read-out cables up the slopes to the crest reduces the requirement to install them through the embankment and thus reduces the potential for development of a seepage pathway. The read-out cables are to be installed in PVC electrical conduit, to ensure they are not damaged during deposition. A 0.5 m cover of clayey fill is to be placed above the piezometer and read-out cable to prevent the tailings damaging the cable. The cables on the base of the TSF are to be aligned, such that the discharging tailings will not occur directly onto them.

17.2.5 Closure and Rehabilitation

17.2.5.1 Progressive Rehabilitation

Given that the TSF is to be raised in the downstream direction for the deposition embankments, progressive rehabilitation of the downstream slopes of these embankments will not be possible throughout the life of the operation.

The suggested rehabilitation involves covering the downstream slope with nominally 500 mm of rock armour sourced from either the limited mine waste or designated borrow pit, to provide protection against erosion due to rainfall runoff. Once a finer-grained growth medium, such as topsoil, is added to the rock armour, the development of vegetation can commence. As part of the topsoil/growth medium placement process, the outer slopes should be contour-ripped. Timing of the rehabilitation works is to be confirmed during the operation of the facility.

It is recommended that any vegetation development on the outer slopes of the TSF comprises local grasses and small shrubs with shallow root zones. The establishment of trees should be avoided due to the increased potential for the roots to penetrate into the tailings.

The rehabilitation plan should be executed in consideration of the tailings raise strategy, such that no topsoil or vegetation establishment activities are undertaken on the slopes, which might be subject to other uses in the future. For example, if additional tailings are required for mine backfill, the embankments containing those tailings may have to be reconfigured. However, rock armour should be temporarily placed on those slopes to ensure erosion protection is in place for the period before construction of the adjacent use commences. The rock armour will need to be removed from the outer slope of the embankment as part of the construction process for the alternative use.

17.2.5.2 ***Post Deposition Rehabilitation***

Following completion of deposition within the TSF, the deposited tailings are to be left to dry and consolidate. It is expected that the water liberated by consolidation and rainfall runoff will continue to pool where the decant rock ring is located. Periodic removal of this water will be required.

A detailed closure design is to be developed as the operational life of the TSF approaches closure. The ANCOLD (2019) guidelines recommend the inclusion of a spillway, sized to discharge a Peak Maximum Flow (PMF) event as part of the closure plan. The requirement for the spillway, including the proposed design and location, will be addressed as part of the closure design works.

Upon confirmation that further perimeter embankment raising is not going to be required and once the deposited tailings have consolidated such that they have gained strength to support construction equipment, the surface of the facility is to be shaped and capped with a layer of clayey fill. Rehabilitating in this manner will provide a stable and low risk feature post closure. Contouring of the final surface and downstream perimeter embankments will aid in reducing the likelihood of rainwater run-off eroding the exposed surfaces.

After completion of reshaping and contouring the TSF surface and downstream embankment slopes, topsoil can be placed over the facility to revegetate with local plant species. Spreading of any available timber (small trees removed as part of the clearing works) over the surface of the facility will also aid in the establishment of vegetation. Contour ripping the exposed surfaces will also aid in enabling the vegetation to establish and reduce the risk of the topsoil being eroded by wind and runoff water.

17.3 Forward Works for TSF

17.3.1 Risk Based TSF Design

A desk-top risk assessment of the preliminary concept design for the TSF has been completed as part of this Scoping Study. Further risk assessment work may be required if the position of the TSF changes and that risk assessment would be executed in parallel with the work outlined in the following sections as part of progressing to detailed design and documentation for regulatory approval.

17.3.2 Geotechnical Investigations and Materials Testing

17.3.2.1 ***Construction materials***

As indicated in Section 17.2.3, TSF construction comprises a downstream zone which will be constructed using mine waste sourced from the proposed mixed open pit and underground operation and in-situ materials, if suitable, as required. Current indications are that adequate volumes of waste are available for the current LOM, refer to Table 17-4.

However, it should be noted that no cut-to-fill has been used in the design, although some excavation within the TSF Stage 1 footprint maybe required to satisfy construction requirements for Stage 1. Assuming the geotechnical investigation and testing confirms the suitability of the in-situ materials for construction use, each metre of cut from within the internal footprint of the TSF would yield approximately 100,000 m^3, which would slightly reduce the mine waste required for the Stage 1 perimeter embankment.

17.3.2.2 ***In-situ materials***

Geotechnical investigations of the site selected for the TSF and construction materials are required to verify the assumptions for the embankment design and construction. This geotechnical work is expected to comprise:

- Borehole and backhoe test pit investigations of the site of the TSF to ascertain the soil, rock and groundwater conditions and collect samples of potential construction materials.
- Particle size distribution tests with hydrometer testing of the soils.
- Atterberg Limit Tests of soils.
- Maximum and minimum density tests of soils.
- Standard Compaction Tests of the soil materials.
- Large shear box tests (300 mm x 300 mm x 150 mm).
- Large scale triaxial tests (150 mm diameter x 300 mm height).

17.3.2.3 ***Tailings Materials***

Tailings testwork has not been completed as part of this study and geotechnical testing and assessment of the tailings characteristics is required. The tailings testing work is expected to comprise the following as a minimum:

- Particle size distribution tests with hydrometer testing of the fine fraction of the tailings.
- Atterberg Limit Tests.

- Drained and undrained settling tests.
- Extended height consolidation testing.

*17.3.2.4 **Failure Impact Assessment***

A failure impact assessment will need to be executed in accordance with the relevant local standards.

*17.3.2.5 **Stability Analyses***

Stability analyses are to be undertaken in the next stage of the project to confirm embankment stability. Where seismicity is demonstrated to be of concern, further analyses using 2D Plaxis (dynamic analysis) may need to be executed to check that:

- The embankment for the TSF remains serviceable following a 1:1,000 AEP earthquake event.
- The embankment for the TSF does not fail catastrophically either as a result of instability with flow of materials, or loss of freeboard followed by overtopping following a Safety Evaluation Earthquake (SEE) 1:10,000 AEP.
- The embankment for the TSF does not fail post closure following an SEE.

*17.3.2.6 **Hydraulic Analyses***

Whist the concept presented in this study shows limited external catchment, where runoff might be diverted, hydrological studies will need to be executed to assess the potential inflows in the vicinity of the TSF, diversion requirements, if any, and confirm the freeboard and spillway requirements.

*17.3.2.7 **Detailed Design Works***

With the completion of the various analyses outlined above, the detailed design works can be executed. These works will need to include:

- Preparation of the Design Report which details the basis of design and the results of the various analyses etc., executed as part of the design works.
- Preparation of drawings and construction specifications.
- Preparation of management plans (Operations Manuals) for the operation and management of the TSF.
- Preparation of the instrumentation and monitoring details to allow the performance of the TSF to be compared with the design expectations.
- Preparation of the Closure and Rehabilitation details, refer to Section 17.2.5.

17.3.2.8 ***Construction Plans***

With the completion of the geotechnical investigations and detailed design works, there will be a need to develop the Construction Plan and QA/QC documentation.

17.3.2.9 ***Construction Reporting***

At the completion of the construction of the TSF embankment, a set of "as built" drawings will be prepared and together with the results of in-situ testing and external testing, will be reported to verify that the construction is in accordance with the design of TSF.

18 PROJECT INFRASTRUCTURE

The Hualilan project is a greenfields site that requires significant infrastructure to support the operations. The infrastructure required includes:

- Access roads
- Water supply
- Energy and power supply
- Accommodation village
- Process plant site buildings for administration, workshops, and warehouses
- Mining area buildings and other infrastructure to support mining operations
- Washdown facilities
- Fuel supply, storage, and distribution
- Communications and information technology
- Fire protection
- Surface water management infrastructure
- Logistics support for the project operations.

Figure 18-1 shows the overall site layout, including the access roads, bore field, process plant, village, mine area, and IWL. Figure 18-2 shows a close-up view of the process plant, IWL, workshops, store and office area.

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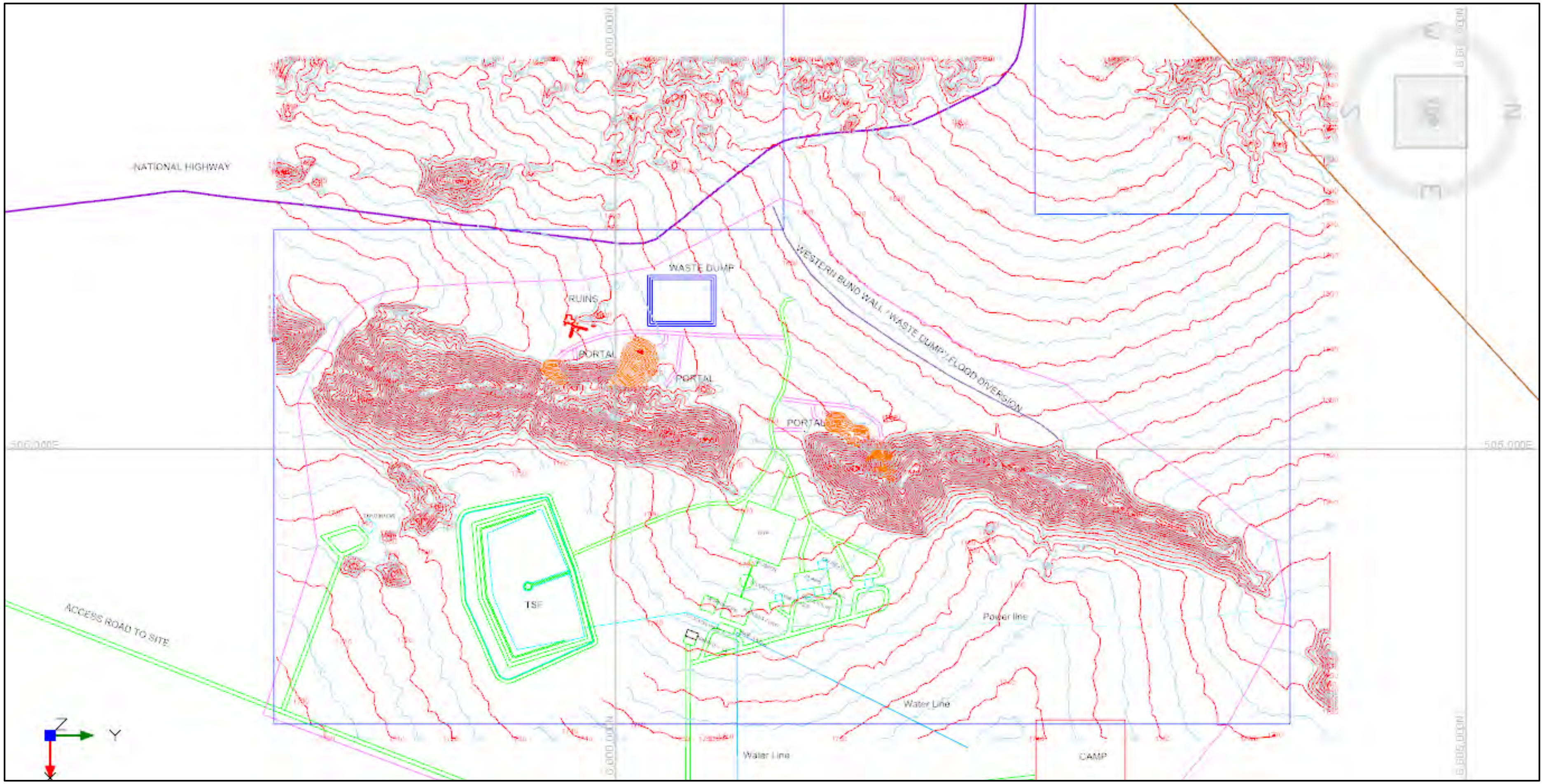

Figure 18-1: Overall Hualilan Site Layout

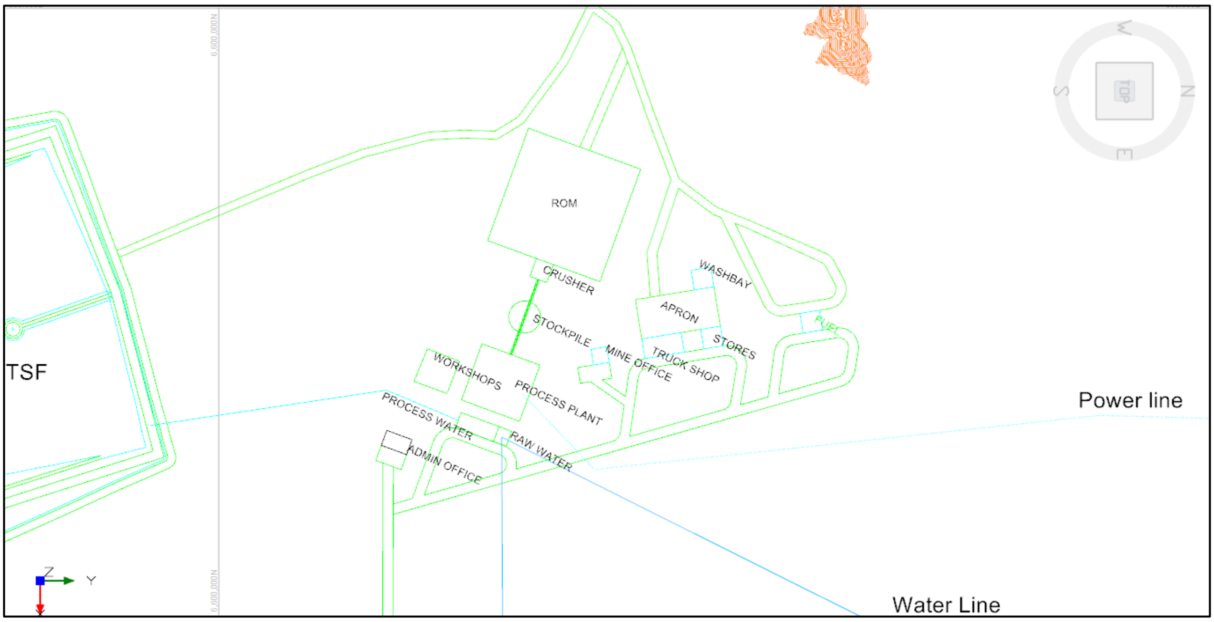

Figure 18-2: Overall Hualilan Site Process Plant Area Detail

18.1 Water Supply

The Company has drilled three exploratory water bores to prove sufficient groundwater resources for the project. In Figure 18-1 the bore can be seen to the northeast, east and southeast of the proposed process plant site. All three wells intersected ground water close to within 40 metres of surface over broad intervals in porous and permeable unconsolidated sediments. The water bores were drilled and flow testing completed as part of a successful hydrogeology study undertaken by the Company in 2023. This study indicated a maximum of 13 additional water bores (Figure 18-3), including monitoring stations, are required to supply the water needs for the project.

A local cost estimate was obtained for the costs of drilling these 13 additional bores for a combined 535 metres of drilling, their completion including submersible pumps and the cost of water pipelines to provide water to the project. This cost estimate of US$1.67 million (exclusive of VAT) did not include the cost of pumping from the bore field to the project or water storage reservoir. An additional allowance of US$0.3 million has been included for the construction of a reservoir and pumping.

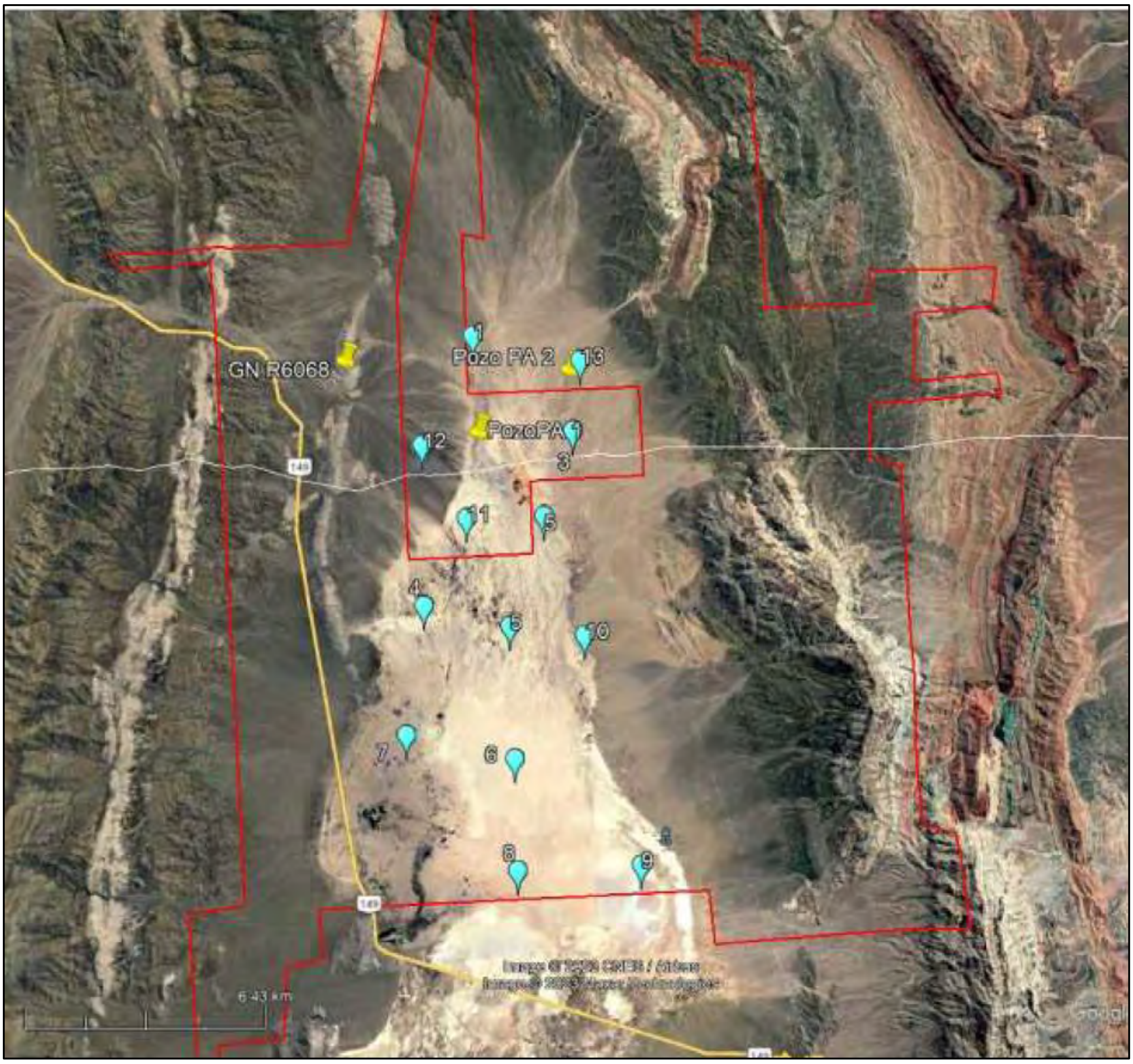

Figure 18-3: Proposed Water Bores

18.2 Power Supply

The estimated total power demand for the scoping study of 7.8 MW is outlined in Table 18-1 below.

Table 18-1: Power Usage

Category	Annual consumption Kwoh
Process Plant and comminution circuit	30,000,000 (30 kWh/t)
Underground mine	38,000,000 (38 kWh/t)
General usage	500,000 kWh
Total Annual Power Consumption	**68,500,000 kWh**
Power Demand	**7.8 MW**

The Company has entered into a partially binding Memorandum of Understanding (MOU) with YPF Luz for the supply of renewable power to the Hualilan Gold Project. YPF Luz is one of Argentina's largest energy companies, with installed generative capacity of 3.2 GMW including 468 MW of renewable energy capacity. YPF is a subsidiary of YPF S.A., Argentina's largest integrated energy company partially owned by the Argentinian Government, with reported EBITDA of US$398 million for the 2022 financial year. YPF has a focus in the renewable energy business and provides its customers with efficient and affordable electric energy, as well as tailor-made engineering solutions to transportation and distribution challenges. Additionally, YPF S.A., the main shareholder of YPF Luz, is the largest distributor of Biodiesel to the mining industry in Argentina.

Under the MOU YPF will present CEL with high level technical alternatives for the supply of renewable electric energy to the Hualilan project, from which CEL shall decide if it wants YPF to develop any of the alternatives presented. In order to develop the chosen alternative, the Parties will negotiate a binding agreement that will include YPF obligation to develop the alternative, and the agreements that the Parties may enter into (such as a Power Purchase Agreement "PPA") if the development is satisfactory. If the project reaches an Energy Supply price below 7 US cents/kWh the Company shall award YPF with a PPA for the Energy Supply to the Hualilan Project. This PPA can, at the Company's election, include all connection capital expenditure associated with the provision of power to the project removing the need for the Company to fund the upfront capital component associated with the provision of power to the project.

Additionally, the Company has received indicative quotes for connection to the grid via duel 33 kV Medium Voltage lines. This connection would be via a 43 km long route to the Bauchazeta power station (Figure 18-4). This estimate of US$4.5 M includes line costs of US$3.5 M at $40,000/km with an additional US$1 M for other infrastructure. Additionally, an amount of US$1.4 M has been included in underground mine costs for electrical sub stations.

Figure 18-4: Proposed Power Supply Route

18.3 Roads

18.3.1 Access Roads

Access to the Hualilan deposit from the city of San Juan, is via the double lane sealed highway, following travel itinerary below:

- National Route No. 40: San Juan – Talacasto Station. Distance: 52 km
- Provincial Route No. 436: Talacasto Station – Junction of National Route No. 149. Distance: 23 km
- National Route No. 149: Junction Provincial Route No. 436 – Deposit. Distance: 45 km.

The main road corridor is used for all types of cargo and substances, hazardous and non-hazardous, which is used by other regional operations and projects in the Iglesia Department.

18.3.2 On-Site Roads

Approximately 26 kilometres of access roads will be constructed as shown in Table 18-2. The roads will be constructed using blasted waste material from the open pit for a 50 cm base course, topped with 30 cm of crushed roadbase made from ex-pit limestone waste for the surface course. The sub-base will be constructed by pushing-up the existing cover material with a dozer and compacting with a drum roller.

Table 18-2: Site Roads

Road	Length (m)	Width (m)	Depth of fill (m)	Fill (m^3)
Main access road to highway	5,744	20	0.8	91,904
Explosives compound access and	1,596	20	0.8	25,536
Explosives compound to administration turn-off	2,515	20	0.8	40,240
Access road from administration turn-off to camp	2,059	20	0.8	32,944
Access road to administration	1,008	20	0.8	16,128
Access roads to fuel, stores, workshop, mine office	2,647	20	0.8	42,352
Central mine access roads	2,981	25	0.8	59,620
IWL to mine access road	1,259	25	0.8	25,180
Road around IWL	3,074	25	0.8	61,480
Mine roads to north	1,056	25	0.8	21,120
Mine roads to south	1,858	25	0.8	37,160
Total	**25,797**			**453,664**

18.4 Camp

The existing 100 person modular exploration camp will be relocated to the site of a proposed 120 bed permanent camp for the operating mine.

This permanent camp incudes an office complex including reception and waiting area, a conference room that can accommodate 18 people seated, a six person conference room, four executive offices, and two administrative offices capable of containing nine workspaces.

The 120 beds are arranged in six separate wings with the majority of rooms containing two beds per room with bathroom facilities in each wing. Some single occupancy rooms with self-contained bathroom facilities are also included. The camp contains full catering facilities for 120 people and dining room. It has a recreation area comprising several informal facilities, a gymnasium, basketball court and half size soccer field.

The design of the camp is shown in Figure 18-5 to Figure 18-7. Several tenders have been received for the construction of this camp with the majority around US$2 million. It is anticipated construction of the camp will start during the planned bankable feasibility study so the camp can be used to house workers during the construction stage. The estimated camp area is 33 ha.

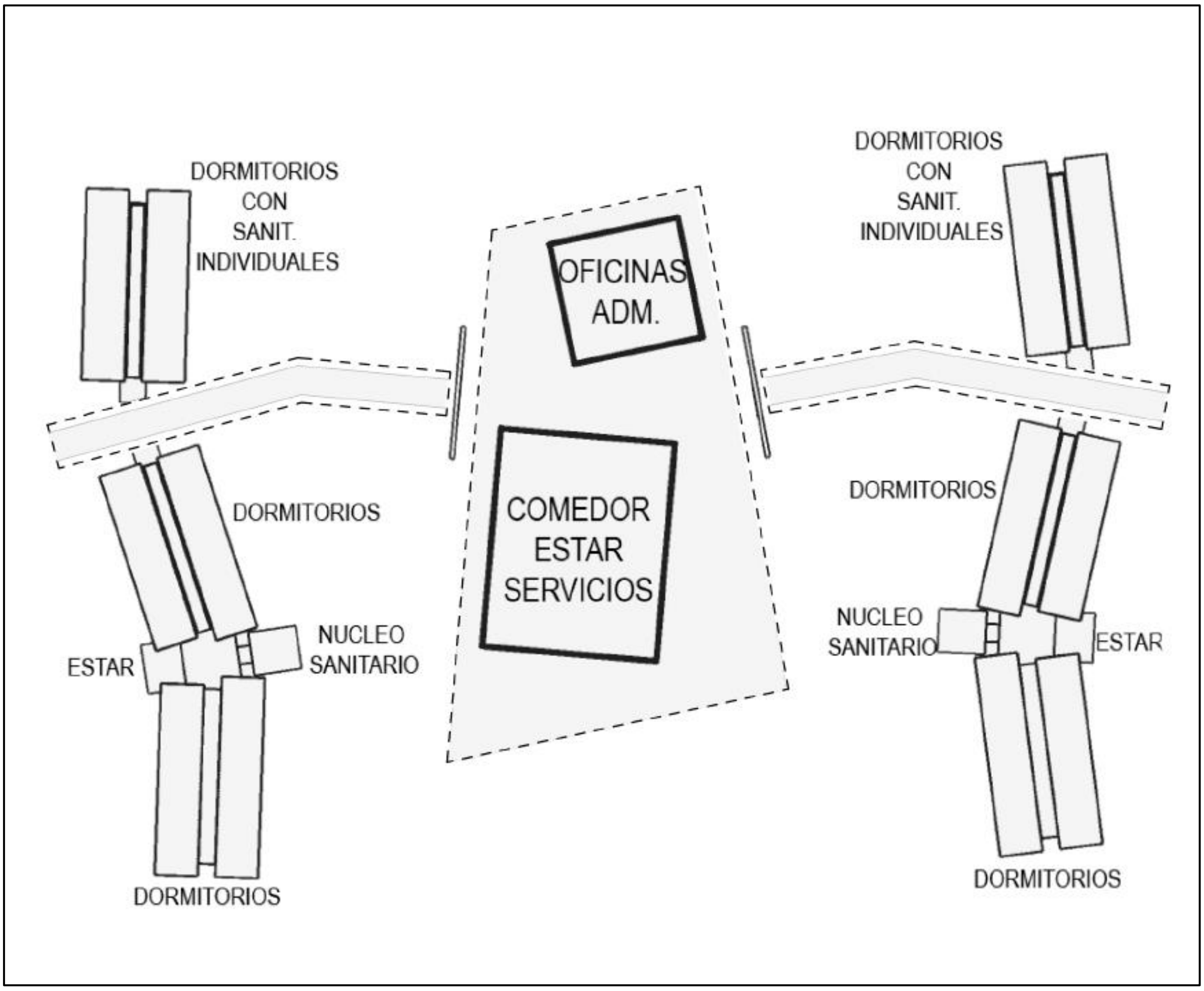

Figure 18-5: Plan showing entire camp and office complex

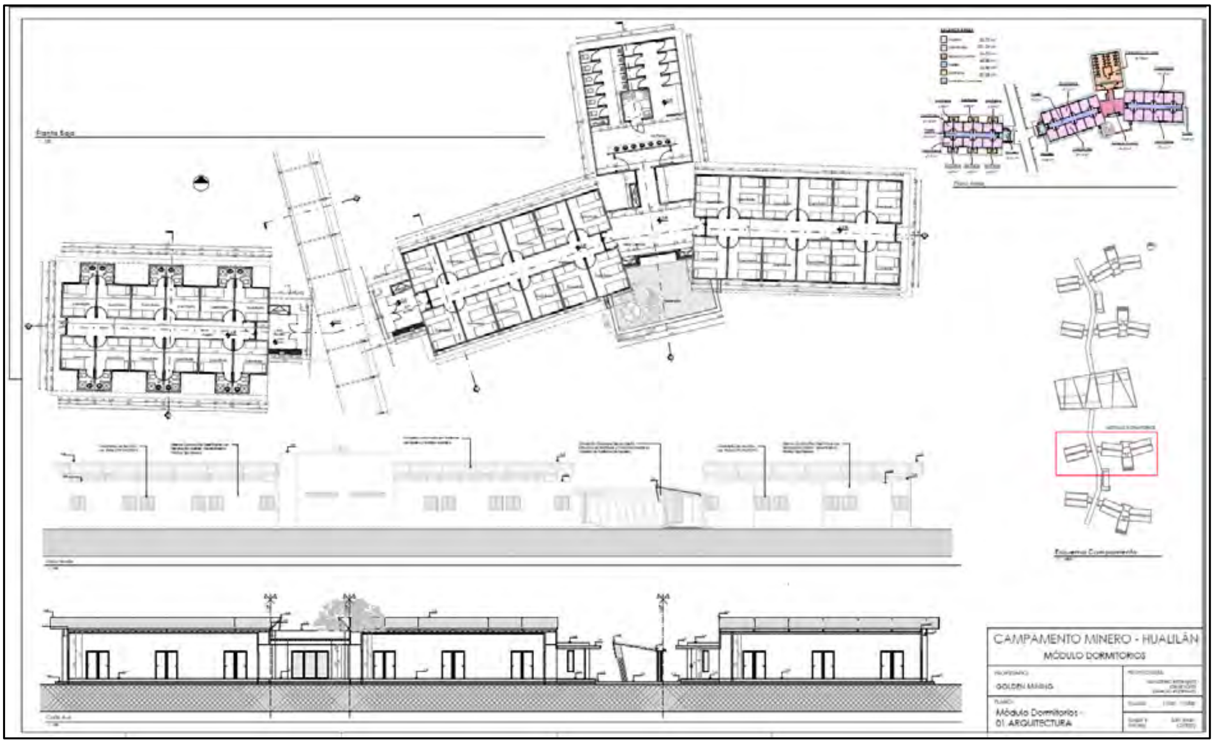

Figure 18-6: Plan showing one of the accommodation wings

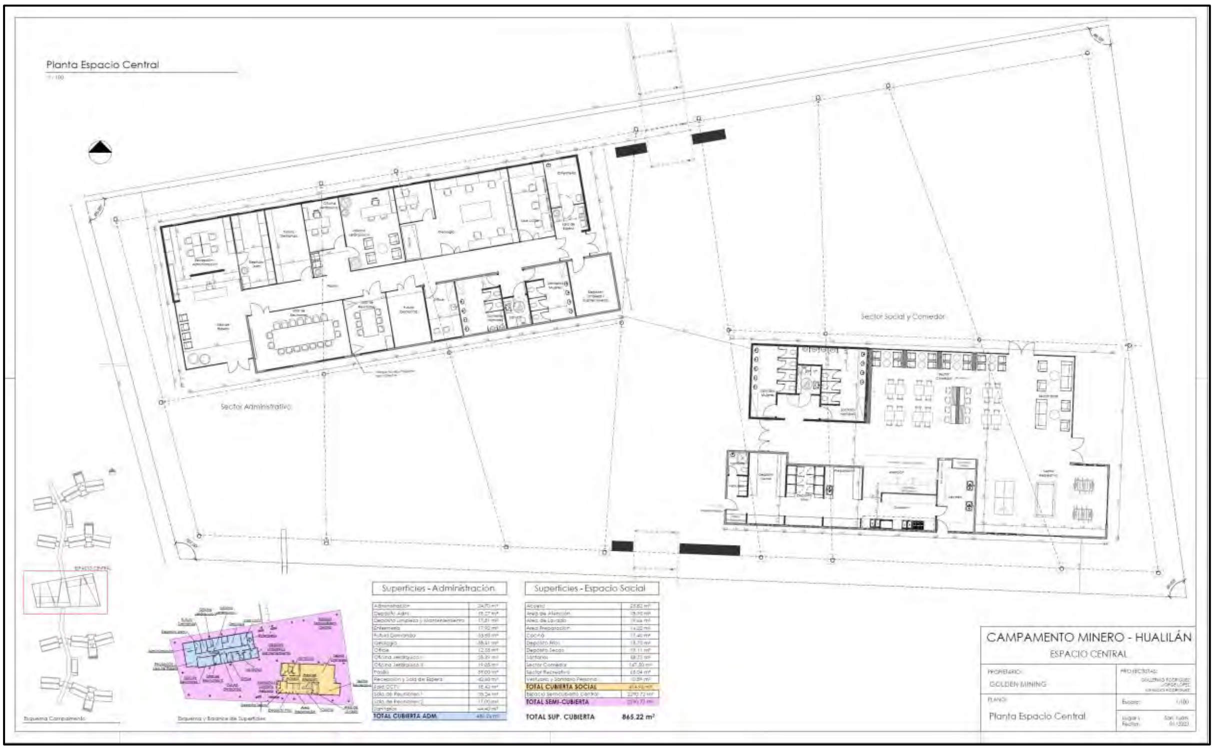

Figure 18-7: Plan showing office and recreation complex

18.5 Potable Water Supply

Raw bore water will be treated through a potable water plant and stored in a series of 20 x $10m^3$ dedicated tanks with dosing capability to provide a minimum of 5 days water at maximum camp occupancy. Water will be distributed as potable water for the camp, offices, mine site and process plant by a pump and series of open and buried pipelines.

18.6 Sewage Treatment

The wastewater treatment begins with the physical separation of large solids (garbage) from their stream using a grid system (meshes), although said waste can also be crushed by special equipment. Subsequently, desanding will be applied (separation of small, very dense solids such as sand) followed by primary sedimentation (or similar treatment) that separates the suspended solids existing in the wastewater.

This is followed by the progressive conversion of the dissolved biological matter into a solid biological mass using suitable bacteria, generally present in these waters. Once the biological mass is separated (a process called secondary sedimentation), the treated water can undergo additional processes (tertiary treatment) such as disinfection, filtration, etc. The final effluent may be discharged or reintroduced back into a natural water body (stream, river or bay or another environment). The final effluent to be discharged must meet the minimum requirements established in the Decree No. 2,107-MIyT-07, regulations of Provincial Law No. 348-L.

The sewage waste generated in the dining room and kitchen will pass through grease separators (traps) before entering the domestic solid waste treatment plant. The sludge generated in the effluent treatment plant will be analysed and depending on the characterisation, it will be treated as urban solid waste or as hazardous (waste stream Y18). If it is classified as urban solid waste (MSW), it will receive final disposal in an Environmental Technologies Park authorised by the State Secretariat of Environment and Sustainable Development. If it is a hazardous waste it will receive final disposal via thermo-destruction in a facility authorised by the provincial environmental authority.

18.7 Waste Management

18.7.1 Oils and Lubricants

For oils and lubricants the waste management plan includes the recovery of the used oils and lubricants, their temporary storage and periodic removal by an operator authorised by the provincial environmental authority. The operator will transport waste to its final disposal via thermo-destruction, after which a final disposal certificate will be issued for the purposes of traceability of this type of waste.

18.7.2 Truck and Heavy Equipment Washing Water

For truck and heavy equipment washing the washing area will have a waterproof concrete base to contain the wash water and prevent it from infiltrating the ground. The washing platform will be built with a slight slope that will allow the resulting waters to be conveyed to a pit where the solids will decant. The clear waters will continue to a chamber, where due to the difference in density, the oil will be separated from water. Periodically, solids (sludge or mud) and grease will be removed from the chamber receiving final disposal via thermo-destruction in a facility authorised by the provincial environmental authority. The water will be reused in washing equipment.

18.7.3 Domestic and Industrial waste

Domestic waste will be generated mainly in the dining room, administrative offices and in health services. Industrial waste generating activities relate to auxiliary operations, equipment maintenance, machinery used in mine operations and in the mineral treatment process. Industrial waste will also be generated in the warehouse sector. Solid domestic waste will also be generated, although in smaller quantities, in the industrial and mining facilities. Waste management will include collection, classification, temporary storage and transfer for the final disposal of domestic waste. The types of waste that fall into this category include food scraps, papers, cardboard, plastic, rubber, wood, glass, cans, dirt or dust resulting from cleaning or cleaning tasks, rags, etc. A waste yard for temporary storage will be available in the mining-industrial complex. These wastes will then be differentiated into recoverable and non-recoverable. The transportation of solid waste – recoverable and non-recoverable – will be carried out by transporters authorised by the State Secretariat of Environment and Sustainable Development and its final disposal will take place in an Environmental Technologies Park or in a recovery or recycler facility authorised by the province.

18.8 Main Offices, Plant, Workshops

18.8.1 Main Office

In addition to the office complex included in the new permanent camp, the current portable offices on site will be relocated to an area of 270 m^2 which has been provided for the main office. These existing offices include seated working space for approximately 30 personnel across six demountable offices.

18.8.2 ROM Pad

Based on the scoping study process plant requirements, the ROM pad has been sized to accommodate all low grade (Type A) gold predominant mineralised material, high grade (Type B) gold predominant mineralised material, and high grade (Type C) zinc predominant mineralised material. The stockpile area is 300 m x 300 m, providing 9 Ha of placement space.

18.8.3 Process Plant

The Process Plant design is detailed in Chapter 16. Based on benchmarking similar scale sequential float process plants, an area of 150 m x 150 m is anticipated for the processing plant. This area is considered large enough for the main components being the mill, flotation plant and float tail leach section.

18.9 Communications

18.9.1 Off-Site Communications

Current communications off-site are provided by a satellite link which provides high speed internet. An existing WiFi network has been created using this satellite link which allows off-site communications via mobile phones using WhatsApp. It should be note that in Argentina, and South America in general, the penetration of WhatsApp is significantly greater than in Australia with the majority of the population making phone calls and messaging via WhatsApp. This existing satellite link and WiFi has catered for over 120 persons in the camp when the company had nine drill rigs active.

During construction and operation this satellite link will be expanded. Additionally, satellite phones and potentially Starlink are expected to be available in Argentina by the time construction commences.

18.9.2 On-Site Communications

Communication on site will be made available through a combination of systems based on purpose. These systems include mobile phones using the existing satellite supported WiFi network for wireless data and voice, WhatsApp on the same network, UHF radios, satellite phones and Starlink which is reported to have availability in Argentina from 2024.

18.10 Mine Facilities

18.10.1 Explosives

The explosives magazine is located along the site's main access road adjacent to the TSF. The magazine will consist of a security fenced compound with CCTV as well as the following sheds for the ultimate type of explosives selected. These may include:

- An ANFO shed.
- Emulsion tanks and chemical gasser.
- Detonator storage and handling equipment.

The company will provide the explosives magazine compound, concrete slabs, lighting, power supply, and security.

18.10.2 Fuel Storage and Delivery

Diesel is expected to be delivered by road to the fuel bay. An upgraded design will incorporate storage to maintain operations for around three weeks should an interruption in supply occur.

18.10.3 Wash Bay

A heavy vehicle washdown pad will be located at the mine area adjacent to the heavy vehicle parking area and mine workshops. This washdown will provide for the effective removal of mud and dust before entering the workshop for maintenance.

18.10.4 Hazardous Waste Storage

A hazardous waste yard will be built between the camp and the process plant; its size has been based on benchmarking of similar operations. The yard must have a waterproof base, pit or secondary containment pool, fence or fence perimeter, it will be roofed, with security signage and fire-fighting elements compatible with the fire load.

Once in the hazardous waste storage area the waste is identified, classified and grouped according to the final destination. There are three alternatives: recovery (recycling), storage or disposal. The process is managed in coordination with the Secretary of State and Environment of the Province of San Juan.

18.10.5 Emergency Response

The current emergency plan in operation during exploration activities at the Hualilan project involves an ambulance and two trained paramedic staff being on site 24 x 7. This ambulance and crew are provided on contract by the regional health service at a cost of approximately US$6,000 per month. This service will be maintained during construction and production and its usage monitored – if required it can be upgraded.

18.11 Security

The current security infrastructure on site comprises a boom gate and guardhouse at the main camp access road. During operation additional security check points will be maintained at the mill site, camp entrances and the explosives shed, with surveillance cameras which will be monitored 24x7 installed throughout the operation.

19 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

19.1 Environmental Studies

Environmental baseline monitoring has been ongoing at the project since March 2021. This monitoring includes:

1. Air Quality
2. Noise and Vibration levels
3. Flora
4. Fauna
5. Weather

Additionally specific independent consultant reports have been prepared and lodged with the San Juan Department of Mines including:

- Archaeological report
- Report on Palaeontology

19.2 Environmental Work Completed

19.2.1 Air Quality

The air quality is measured by fraction of respirable particulate matter, i.e. particulate matter of aerodynamic diameter less than or equal to 10 µ (PM10) in the air and the content of some trace elements present in the air.

The following is a summary of background air quality monitoring campaigns carried out by Enviro Services Group SRL between November 2022 and January 2023. Under the framework of National Law No. 24.585/95 on Environmental Protection for Mining Activities, the PM10 particulate and certain chemical elements are evaluated at four monitoring stations (Table 19-1). The location of the monitoring stations is shown in Figure 19-1. The concentrations measured are below the quality guide level indicated in National Law Nº 24.585/95.

Table 19-1: Particulate Matter Concentrations PM10

Station	Guidance Level	Nov 2022	Dec 2022	Jan 2023
E	150 µg/m^3	25.1 µg/m^3	82.9 µg/m^3	4.3 µg/m^3
N	150 µg/m^3	11.0 µg/m^3	25.3 µg/m^3	1.61 µg/m^3
S	150 µg/m^3	11.5 µg/m^3	8.0 µg/m^3	4.3 µg/m^3
SE	150 µg/m^3	9.0 µg/m^3	34.2 µg/m^3	35.2 µg/m^3

Figure 19-1: Location of monitoring stations

The impact on air quality from particulate matter (PM-10) was evaluated by project activities. This fraction of particulate material was selected because it has two characteristics:

1. It can be maintained in atmospheric suspension and transported outside the Project.
2. It is small enough to be inhaled and can penetrate the respiratory tract reaching the alveolar zone of the lungs, which is the site of gas exchange.

In addition, PM10 samples were chemically analysed to determine Lead (Pb), Arsenic (As), Mercury (Hg), Cadmium (Cd) and silica (SiO_2) content.

- For all measurement stations, Pb concentrations do not exceed the guideline level (1.5 µg/m^3) and are below the detection limit of the analytical method used by the laboratory (0.0064 µg/m^3).
- The concentration of As, Hg, Cd and Silica is not regulated in the National Law Nº 24.585/95. Concentrations recorded in all the recording stations and for all the monitoring campaigns are below the detection limit of the analytical method used by the laboratory (0.0064 µg/m^3).

19.2.2 Noise Levels

Background environmental noise measurements were carried out on 14 October 2022 at different points in the Departments of Iglesia, Jáchal and Ullum and in the perimeter of the future mine. Measurements were made with a "slow" dynamic response, taking continuous measurements over at least 10 minutes duration at morning, afternoon and evening measurement times. Table 19-2 shows the results. Taking as a reference the levels suggested by the World Bank of 70 dBA, no record exceeds the reference level.

Table 19-2: Noise measurement results

Point	Reference	Reference value	Morning value	Afternoon value	Evening value	Qualification
1	Villa Iglesia	70 dBA	62 dBA	59 dBA	57 dBA	Normal
2	Projected Camp	70 dBA	48 dBA	47 dBA	46 dBA	Normal
3	Mine Area Barycentre	70 dBA	57 dBA	59 dBA	54 dBA	Normal
4	Ciénega de Gualilán	70 dBA	61 dBA	63 dBA	60 dBA	Normal
5	Baños de Talacasto	70 dBA	62 dBA	63 dBA	61 dBA	Normal
6	Talacasto station	70 dBA	63 dBA	64 dBA	62 dBA	Normal

19.2.3 Hydrology and Hydrogeology

Hydrology and Hydrogeology studies are outlined in Chapter 14.

19.2.4 Flora and Vegetation

During the months of March 2021, May 2022 and November 2022 the sampling of flora and vegetation, by Dr. Héctor J. Villavicencio, was conducted in order to characterise these ecosystem variables. The main objectives of the study were:

- Survey the plant diversity of the study area to make an inventory for the area, with special emphasis on endemic species or species of special interest of conservation for the region.
- Characterise the plant communities in the surveyed area, indicating composition, richness and diversity for each determined unit, which were mapped.
- Prepare a report on the information collected.

For the purposes of characterising the biotic components, four environmental units or sectors have been recognised, which have distinctive features and are represented - spatially - in the project footprint and water study area, respectively. Figure 19-2 shows the four zones that are described as follows:

- Pedemontane descent of the La Invernada and Talacasto mountain ranges: represented by an extensive pedemontane plain that covers the East flank of the La Invernada mountain range and the West flank of the Talacasto mountain range. In this unit there is evidence of fluvial action, mechanical disintegration, and mass removal phenomena. The drainage network is made up of a series of fluvios and interfluvios, which are temporary channels, fed by rainfall during the summer season.
- Pampa de Gualilán: includes the northern portion of this geoform, which develops between the mountain ranges of La Invernada, to the west, and Talacasto, to the east, which in its most depressed sector presents fine sediments of beach or barreal environment.
- Sierra de Hualilan: includes rocky outcrops formed by calcareous and clastic sediments from the Paleozoic.
- Medanal: located to the east of the mining project, on the pedemontane slope of the Talacasto mountain range.

The environmental units considered present altitudinal heterogeneity and morphostructural variants, aspects that are accompanied by changes in vegetation patterns. The typical expression of the Phytogeographic Provinces of Monte and Cardonal is observed, as well as a sector of ecotone with the Phytogeographic Province of Puna.

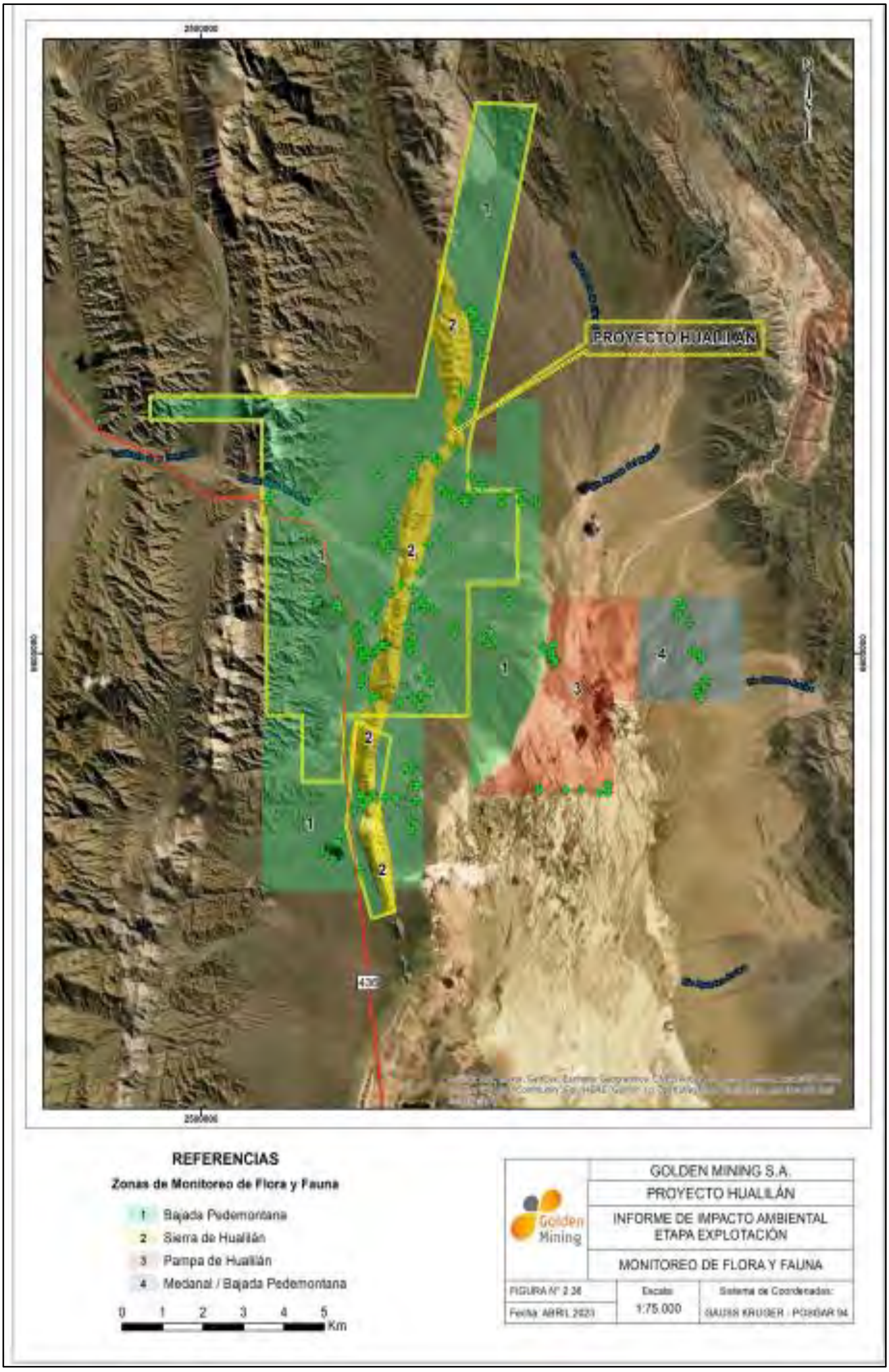

Figure 19-2: Flora and fauna monitoring zones

Most of the area corresponds floristically to the phytogeographic provinces of Monte and Cardonal with some ecotonal species of the Puna, as the communities found, as well as the altitudinal, geomorphological and edaphic variables. The presence of 97 plant species belonging to 32 families (according to PlanEAr "Plants Endemic to Argentina") was confirmed.

- One species, (Maihueniopsis recurvata) was categorised as plants of restricted distribution, but with scarce populations or on which one or more threat factors (habitat destruction, overexploitation, biological invasions, etc.) are presumed to be acting.
- Three species (Aloysia castellanosii, Heliotropium ruiz-lealii, Artemisia mendozana) were categorised as plants restricted to a single province, or with small areas shared by two or more contiguous political provinces.
- 15 species were categorised as common plants, although not abundant in one or more of the phytogeographic units of the country (case of taxa with disjunct distribution): Solanum hastatilobum, Bredemeyera colletioides, Rhodophiala mendocina, Sclerophylax arnottii, Tunilla corrugata, Zuccagnia punctata, Lecanophora heterophylla, Trichocereus candicans, Tephrocactus aoracanthus, Tephrocactus alexanderi, Pterocactus aff tuberosus, Dolichlasium lagascae, Gymnophyton polycephalum and Denmoza rhodacantha.
- Six species (Echinopsis leucantha, Trichocereus strigosus, Atriplex aff crenatifolia, Prosopis torquata, Monttea aphylla, Lippia junelliana), were categorised as abundant plants, present in only one of the large phytogeographic units of the country.
- Six species (Gochnatia glutinosa, Hyalis argentea, Atriplex lampa, Prosopis alpataco, Larrea cuneifolia) were categorised as plants with restricted distribution, but with scarce populations or which are presumably threatened by one or more factors (habitat destruction, overexploitation, biological invasions, etc.).

No irreversible impacts are expected on ecological processes in the project footprint and areas surroundings. No mitigation measures are planned for the flora resource given that, in the Hualilan project, there is no impact, either directly or indirectly, on sensitive vegetation units. Additionally, it should be noted that in the footprint of the project and water study area there are no high plains or wetlands which are subject to an additional set of regulations.

19.2.5 Fauna

During the months of March 2021, May 2022 and November 2022 the monitoring of invertebrate fauna, by Drs. Juan C. Acosta and Héctor J. Villavicencio, was conducted in order to characterise this component of the Project. The main objectives of the study were to:

1. Survey the diversity of vertebrates in the study area to make a list general, with special emphasis on endemic species or species of special conservation interest for the region.

2. Characterise faunal communities indicating richness, equity, dominance and diversity for each given unit.
3. Estimate the relative abundances and/or relative frequencies of species in their associated environment.
4. Categorise species according to local or national endemism, international conservation status, presence in CITES, legal protection.
5. Determine seasonal altitudinal and latitudinal displacement patterns.
6. Characterise and map vertebrate communities by natural units identified in the area.
7. Identify and map priority conservation areas and critical habitats as defined by the IFC (International Finance Corporation).

This fauna Monitoring Program is ongoing.

19.2.5.1 ***Amphibians***

The presence of one specimen of an amphibian species typical of the Monte desert, the frog *Pleurodema nebulosum* (Figure 19-3), was detected and its conservation status is "Not threatened". The individual was recorded in the dunes area in a temporary watercourse under a rock. In view of this, and to determine its distribution within the area of influence of the project, it is necessary to conduct censuses after the rainy season, when the population is active in the temporary pools in summer to reproduce.

Figure 19-3: View of a specimen of Pleurodema nebulosum

19.2.5.2 ***Reptiles***

Nine reptile species were recorded, making up the total richness for the spatial units for the spring season (November 2022). Sampling should continue in summer when the herpetofauna is in full reproductive activity and thus expand the ectotherm community.

Calculations have been made for the purpose of estimating the representativeness of the species as a function of the sampling date and the environment. In Table 19-3 the mean of the measures of Central Tendency of the relative abundance by species is presented, where:

- Kruskal-Wallis "H" test. *P<0.05: significant differences. (The Kruskal-Wallis H test is a rank-based nonparametric test that can be used to determine if there are statistically significant differences between two or more groups of an independent variable on a continuous or ordinal dependent variable.)

- Site 1 refers to: Sierra de Hualilan
- Site 2: Medanal
- Site 3: Pampa de Gualilán
- Site 4: Pedemontane descents of Sierra de la Invernada and Sierra de Talacasto.

Table 19-3: Relative abundance of reptiles in the area in the studied sites

Environment	Site 1	Site 2	Site 3	Site 4	H test
Leiosaurus jaguaris	0.10	0.02			P<0.05
Liolaemus olongasta		0.12	0.02	0.01	P<0.05
Homonota horrida	0.03				------
Aurivela longicauda		0.9			------
Pseudotomodon trigonatus		0.10			------
Philodryas psamophidea		0.02			------
Philodryas trilineata	0.03				------
Bothrops ammodytoides		0.01			------
Siagonodon borrichianus				0.01	------

Liolaemus olongasta (Figure 19-4) and *Leiosaurus jaguaris* (Figure 19-5) showed statistically significant differences in abundance between environments. The rest of the species were present in only one environment.

Figure 19-4: View of a specimen of Liolaemus olongasta

Figure 19-5: View of a specimen of Leiosaurus jaguaris

Relative abundance data becomes valuable if protocols are repeated and compared over time. If it is possible to monitor this variable, the value of the description of its variation over time and by site is important to determine whether there are population fluctuations over time.

The results show different presence and abundance among environments due to the differential distribution in relation to the use of microhabitats and the biogeographic linkage of the different species. There are species from montane, montane-puna ecotone, rocky and flat environments without rocks. Due to previous surveys nearby (Acosta, Laspiur, Blanco, & Villavicencio, 2016) (Acosta & Blanco, 2017) there are probabilities of surveying more species, especially snakes, so it is necessary to continue monitoring. At this stage there are no local endemics or species compromised in terms of conservation.

19.2.5.3 ***Birds***

The bird community recorded was represented by 33 species, from 18 families. Among these families, the best represented was Thraupidae (with five species), followed by Tyrannidae (with four species). Among the species, *Phrygilus gayi* (Figure 19-6), *Zonotrichia capensis* (Figure 19-7) and *Diuca diuca* were the best represented species in the whole area and in great abundance. Meanwhile, other species that occur in numerous flocks in specific sectors of the area were represented by *Rhopospina fruticeti*, *Aeronautes andecolus*. *Rhea pennata* was also well represented.

Figure 19-6: View of a specimen of Phrygilus gayi

Figure 19-7: View of a specimen of Zonotrichia capensis

The avifauna recorded in the project during the sampling showed a great heterogeneity in its distribution and different possibilities of satisfying different requirements concerning the vital activity of the individuals in the monitored environments. It has become evident that the species recorded make equivalent use within the entire study area, reducing competition for resources.

19.2.5.4 *Mammals*

The presence of 11 mammal species was confirmed: 10 native and 1 exotic. These are detailed in Table 19-4 and photographic examples of three are provided in Figure 19-8 to Figure 19-10.

Table 19-4: Mammals species in area of the project

	Order	Family	Specie
Native species	Carnivora	Canidae	Licalopex culpaeus
		Felidae	Puma concolor
		Mephitidae	Conepatus chinga
	Rodentia	Ctenomydae	Ctenomys sp
		Cricetidae	Abrothrix olivacea
		Muridae	Phyllotis xanthopygus
		Caviidae	Microcavia australis
		Abrocomidae	Abrocoma cinerea
	Cingulata	Dasypodidae	Chaetophractus villosus
	Cetartiodactyla	Camelidae	Lama guanicoe
	Chiroptera	Vespertilionidae	Myotis dinellii
Exotic species	Lagomorpha	Leporidae	Lepus europaeus

Figure 19-8: View of a specimen of Myotis dinellii.

Figure 19-9: View of a specimen of Microcavia australis.

Figure 19-10: Photo-trapping of puma in rocky areas of Sierra de Hualilan

In Table 19-5 species of native mammals confirmed for the study area and their conservation status according to the criteria of:

- Red List of Mammals of Argentina: Extinct (EX), Data Deficient (DD), Critically Endangered (CR), Vulnerable (VU), Potentially Vulnerable (NT), Least Concern (LC).
- IUCN: Near Threatened (NT); Least Concern (LC); Not Evaluated (NE); Data Deficient (DD); Extinct (Ex); Extinct in the Wild (Ew); Critically Endangered (CR); Endangered (EN); Vulnerable (VU).
- C.I.T.E.S: Category I: species in prohibited international trade, Category II: species in regulated international trade, Category III: species in regulated international trade, requiring the cooperation of other countries to prevent unsustainable or illegal exploitation.

Table 19-5: Confirmed native mammal species for the study area and their conservation status

Species	Categorisation 2019 Argentina	UICN 2016	CITES
Lycalopex culpaeus	NT	LC	II
Phyllotis xanthopygus	LC	LC	
Abrothrix olivacea	LC	LC	
Ctenomys sp.	LC	LC	
Microcavia australis	LC	LC	
Abrocoma cinerea	LC	LC	
Lama guanicoe	LC	LC	II
Conepatus chinga	LC	LC	
Chaetoprhactus villosus	LC	LC	
Puma concolor	NT	LC	II
Myotis dinellii	LC	LC	

The native mammal fauna of the area shows moderate richness and uneven abundance depending on the faunal group due to the harshness of the environment in general (poor and unstable soils, low biological productivity and trophic supply). Eleven mammals were recorded. There was evidence of high abundance of red foxes, possibly related to the learning of these populations to human presence and access to food of anthropogenic origin.

19.2.6 Fauna Monitoring Program

The Fauna Monitoring Plan described in the EIA indicates that it will be carried out during the construction, operation and closure stages with the objectives of advancing the knowledge and characterisation of the biotic components of the project footprint, water study area and assessing the effectiveness of the detailed prevention measures.

- Selection of environments: Monitoring will be carried out in four environmental units: Pedemontane descent of the La Invernada and Talacasto mountain ranges, Pampa de Gualilán, Sierra de Hualilan and Medanal.
- Faunal biological indicators: to assess conditions, changes or trends (species richness, abundance, dominance, algal diversity, beta and gamma, etc.) for the assessment and monitoring of faunal biodiversity in the project area.
- Sampling frequency: Two annual samplings, one in spring/summer and the other in autumn/winter, in order to record seasonal and annual patterns (between different sampling years and with the baseline studies) of the biological indicators of vegetation/fauna, to understand and interpret the dynamics of the biotic component.
- Biodiversity sampling: Biodiversity time-series surveys or monitoring are essential elements for data collection, allowing the detection of new species, species turnover and species disappearance for the system under study. This monitoring is suggested in the best practices guide for biodiversity management in the mining industry, a

document prepared by the Argentine Chamber of Mining Entrepreneurs (CAEM). It is therefore expected to continue monitoring in order to obtain multi-temporal parameters and patterns of species richness, abundance, and diversity in order to obtain general trends as well as to complete the biodiversity lists.

19.2.6.1 ***Amphibians***

Because amphibians depend on water for reproduction, they are an important group to monitor as any modification of their environments impacts their life cycles and the consequences are easily quantifiable. Amphibians will be sampled at all monitoring sites. In each of the spatial units, an active diurnal search for amphibians, larvae and egg-laying will be carried out.

In addition, nocturnal (dusk) searches will be carried out, consisting of random transects with torches in the same environments surveyed during the day, in order to census active adult individuals at night. Acoustic surveys will also be carried out to detect nocturnal species. Although desert amphibians depend on occasional rainfall for their activity, they can move at night even without rainfall, in humid areas or under stones or crevices.

19.2.6.2 ***Reptiles***

Stratified sampling will be carried out according to the spatial units defined in the baseline. Within each stratum, three transects (200 m long) will be randomly distributed and two pitfall traps will be placed in each of them.

The count of individuals in each transect will be done by visual encounter following the criteria of (Ojasti & Dallmeier, 2000) and Heyer (2001). The surveys will be both non-extractive and extractive in order to make precise taxonomic determinations. In the case of captures, sliding snares and forks will be used, and specimens will be released after identification.

19.2.6.3 ***Birds***

The following will be carried out: i) Censuses in linear vehicle transects, for direct observation of a continuous route and with length control in internal circuits and ii) Censuses in observation stations, with time control, in areas suitable for the use of this technique.

In the sampling stations and transects, it will be assumed that all individuals found within them are counted, and that all observations are independent.

At the sampling stations, the counting method of (Brandolin, Martori, & Avalos, 2007) will be used, selecting areas in which to stay for 10 minutes. Interspersed with each sampling station, transects will be used with the aim of identifying individuals that might be hidden. In this way, running 25 m from the central point of the station, new species can be recorded.

The count will begin after a short period of time (no less than one minute) after arriving at the centre of the station, to allow the disturbance to cease, and the birds will be identified during the determined time (Salinas, Arana, & Pulido V, 2007). The bandwidth and radius, respectively for both methods, will be the distance at which birds will be detected and correctly identified in each habitat (visibility criterion), using 10x50 dioptre binoculars and cameras.

The observation stations and transects will be geo-referenced before starting the surveys and will be fixed. For identification purposes nests, droppings, roosts, footprints, pellets and song recordings will also be recorded. Only direct observations or sightings will be used to calculate relative abundance, all other signs or traces will be used to determine richness. The surveys will be non-extractive.

The number of individuals will be recorded as accurately as possible (Ralph, et al., 1995) (Bibby, Jones, & Marsden, 1998) using for identification the field guide and songs of (Narosky & Yzurieta, 2010) and Bernabé López-Lanús (2020). In addition, for the nomenclature of bird orders and families, (Remsen Jr, et al., 2020) will be consulted.

19.2.6.4 ***Mammals***

Sampling will combine different methodologies based on the characteristics of different groups of mammals and different types of environments. Stratification will be based on communities, size, detectability and abundance of species.

There are several methods for mammal recording and they can generally be divided into direct and indirect methods. As a direct method the field methodology consists of direct observation and counting of large mammals. For small mammals, trapping techniques are used and records of signs of activity such as tracks, carnivore faeces, caves, pellets of raptors and bones in general found in the environment are used (Aranda, 2000) (Ojasti & Dallmeier, 2000).

For the identification of captured specimens and bone remains (skulls, mandibles, teeth, etc.) found in carnivore faeces, (Olrog & Lucero, 1981), (Redford & Eisemberg, 1992) and the keys of (Braun & Diaz, 1999) Díaz (2000) and the descriptions of (Jayap, Ortiz, Teta, Pardinas, & Delias, 2006) will be followed.

19.3 Archaeological Heritage

As part of the baseline studies and for the purpose of identifying and characterising sites of heritage value - historical, archaeological and paleontological - field surveys were carried out covering the footprint of the project. These were complemented by a review and analysis of the available bibliographic background.

19.3.1 Sites of historical and archaeological value

In October 2022, archaeological monitoring of the area linked to the Hualilan mining project, the town of Hualilan (originally Gualilán) located in the Department of Ullum, Province of San Juan, was carried out.

For the purposes of the field work, the respective authorisation was obtained from the Secretary of Culture of the Province of San Juan, authority in charge of the application of heritage laws, as stated in Resolution No. 0185-SC-2022.

Among the main conclusions reached in the report prepared by Dr. Catalina T. Michieli and Arq[3]. Carlos Gómez Osorio - which included an exhaustive review of bibliographic background - it is worth mentioning point four of the Conclusions, which states: "It is also completely arbitrary and unfounded to affirm that in the area of the mine and its immediate surroundings there are real estate remains and/or archaeological materials from pre-Hispanic (i.e., indigenous) times, as it was proved in the first survey of 2004 and ratified in each of the sites in this monitoring". The report concluded that none of the existing structures on site have any archaeological significance due to extensive periods of later modification.

19.3.2 Sites of Paleontological Value

The Hualilan project is located in the Central Precordillera where the calcareous and clastic sedimentary sequences of the Lower and Middle Palaeozoic - which outcrop in the Hualilan Sierras - are characterised by their fossiliferous content.

In view of the background of the region, a field survey of the footprint of the project was carried out, involving geological, stratigraphic, structural and paleontological observations by Dr. Susana Heredia and Dr. Ana Mestre (Heredia & Mestre, 2022)(IIM - CONICET).

The paleontological survey confirms that the units are not important as paleontological sites since it observed remains that are present in other sectors of the Precordillera.

19.4 Landscape

The perception of a given landscape results from the identification of a certain number of elements - generally abiotic, biotic and anthropic - which are organised and structured giving rise to easily distinguishable characteristic configurations such as mountainous, rural, desert, or urban landscapes.

[3] Denotes a qualification in architecture.

In order to determine the landscape value of the project area, a field survey was carried out in November 2022, with cartographic support (topographic charts and satellite images).

19.4.1 Landscape Units

The reconnaissance of the terrain made it possible to identify three landscape units which due to their physiognomy and general characteristics, coincide with the main geoforms of the relief. These are shown in Figure 19-11 and described as follows:

- Sierra de La Invernada: includes the landscapes that develop on the eastern flank and its pedemontane slope, elongated in a southern direction, with irregular topographic features, presence of anthropic components represented by the route of Provincial Route Nº 436 and combined with those of abiotic and vegetational character. There are frequent wide views and panoramic conditions towards the surrounding reliefs (Cordón del Peñón, Sierras de Hualilan and Pampa de Gualilán).
- Sierras de Hualilan: includes landscapes related to the mountain domain, elongated in a N-S direction, irregular topographic features, steep slopes and narrow ravines. It has a marked anthropic component related to mining activities - past and present - and subordinate abiotic and vegetational components.
- Pampa de Gualilán: includes the landscapes on the eastern flank of the Sierras de Hualilan and the Pampa de Gualilán. It is a broad landscape, with morphological features dominated by a sub-horizontal to horizontal topography. The human component is restricted to the route of the LEAT 500 KV (Linear de Extra Alta Tensión – high tension power line), with abiotic elements dominating. There are frequent wide views and general panoramic conditions towards the surrounding reliefs represented by the Hualilan and Talacasto mountain ranges.

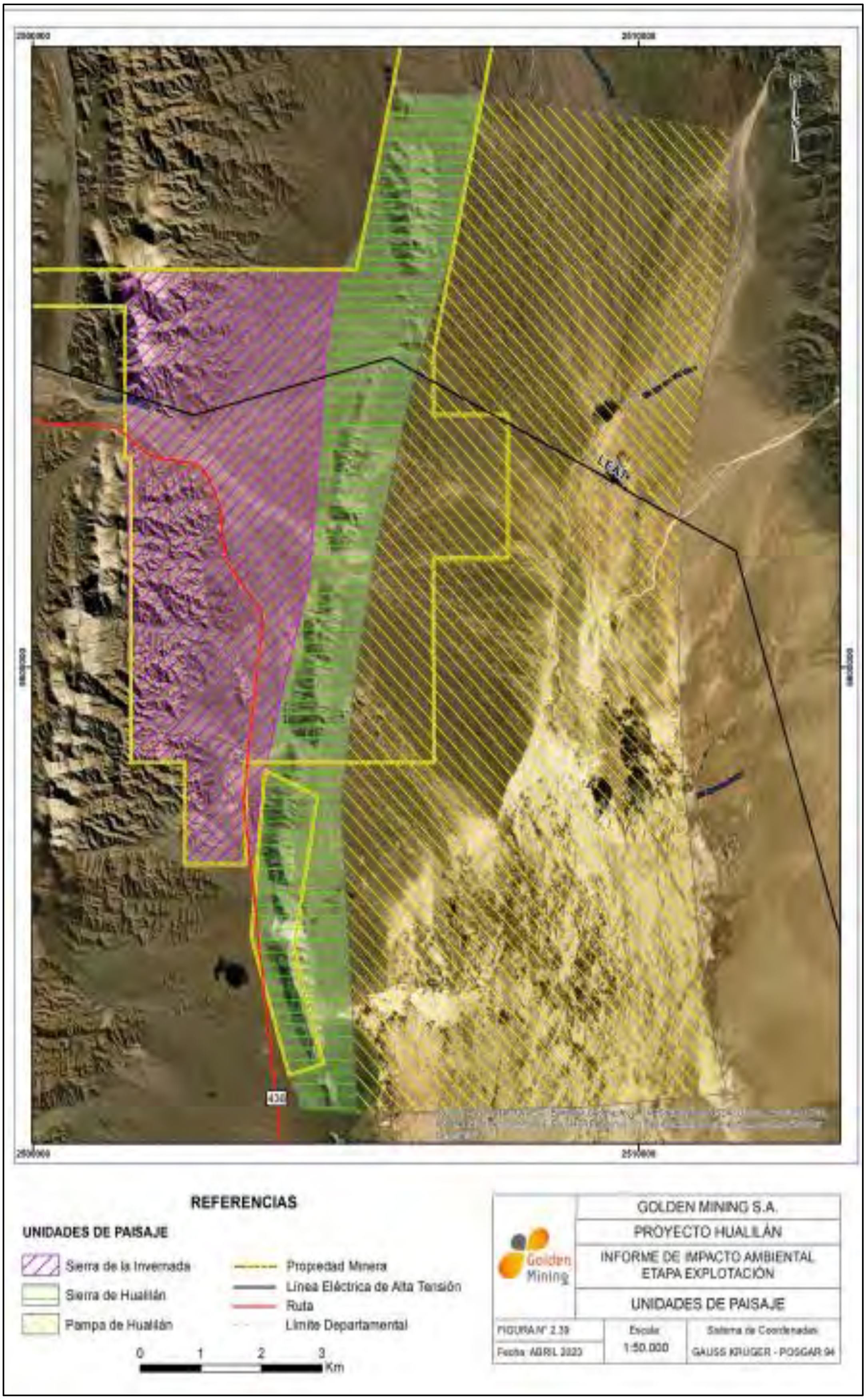

Figure 19-11: Landscape units

19.4.2 Visual Basins

*19.4.2.1 **Sierra de La Invernada Landscape Unit***

The landscape of this area can be divided into 2 types. The eastern flank of the Sierra de La Invernada presents an abrupt relief, of great amplitude and vegetation corresponding to the Monte. To the east is the pedemontane slope, with a sub-horizontal morphology, of marked amplitude and with the presence of shrub and herbaceous vegetation characteristic of the Monte. See Figure 19-12.

Figure 19-12: View in a N-NW direction of the landscape unit Sierra de La Invernada, showing the eastern flank of the mountainous domain and the and the pedemontane slope.

In both areas it is a landscape of natural features, where the only anthropic intervention is given by the layout of the Provincial Route Nº 436. In the winter season, due to the scarce contribution of rainfall, the vegetation makes gray and ochre colours predominate. Conversely, in the summer season, the vegetation plays an important role in the landscape due to the greenish tones of the vegetation cover.

The morphology of the area as a whole, mostly sub-horizontal, defines visual territories of rounded (visual angles of 360º and visual ranges exceeding 10 km) or semi-rounded shapes flanked in all directions by mountain ranges - such as the Cordón del Peñón to the north and the Sierras de Hualilan to the east - or by flat areas such as the Pampa de Gualilán to the east.

19.4.2.2 ***Sierras de Hualilan Landscape Unit***

The landscape of this unit is characterised by a marked N-S orientation, irregular topography, steep slopes and narrow ravines that segment the mountain core. See Figure 19-13.

The shrub, sub-shrub and herbaceous vegetation is typical of the Monte. The shades that this component gives to the landscape depend on the rainfall regime, dominating the ochre during the winter season and different shades of green in the summer.

Figure 19-13: View E-NE of the Sierras de Hualilan landscape unit, which is characterised by its southern orientation and steep characterised by its southerly orientation and steep slopes.

It is a landscape with notable evidence of anthropic intervention as a result of the exploration and mining activities carried out in the district from the 19th century to the present.

The morphology of the area defines visual territories of semi-rounded shapes delimited to the west by the Sierra de La Invernada, to the NW by the Cordón del Peñón and to the east and south by a relatively flat area represented by the Pampa de Gualilán.

The relief features allow an observer circulating through the main direction of observation, represented by Provincial Route Nº 436, to move in any direction and mostly not alter its visual territory.

It is a unit of low compactness, with views interrupted by very few obstacles and limits defined by the surrounding relief. The visual basins are subrounded, with a wide visual domain and range (visual angles of 360º and visual ranges greater than 10 km).

19.4.2.3 ***Pampa de Gualilán Landscape Unit***

The landscape of this unit can be divided into two types. The lower part, of beach environment, presents a flat morphology of great amplitude and practically devoid of vegetation cover. The foothill sectors link towards the Hualilan Sierras to the west, and towards the Talacasto Sierra to the east, where the vegetation cover is characteristic of the mount.

Both areas integrate a landscape of natural features, with a low level of anthropic intervention, represented by the LEAT 500 KV and its service roads.

The regular and mostly sub-horizontal or flat topography of the relief defines territories of rounded shapes (360º visuals), flanked in all directions by mountain ranges (Sierra de La Invernada, Sierras de Hualilan, Sierra de Talacasto, Sierra de La Crucecita, etc.). See Figure 19-14.

Figure 19-14: View in a S-SE direction of the Pampa de Gualilán landscape unit, which is characterised by its flat morphology and great amplitude.

The morphological and topographic attributes of this unit determine a low compactness, being territories whose visuals are interrupted by the effect of very few obstacles and with limits perfectly defined by the surrounding reliefs. The visual basins are rounded, with a wide visual domain and range (visual angles of 360º and a visual range of more than 10 km).

19.5 Protected Areas and Priority Sites

Figure 19-15 shows the location of the Hualilan project in relation to the Natural Protected Areas (NPA) of the Province of San Juan, concluding that the project is distant from any NPA of national, provincial and/or municipal jurisdiction.

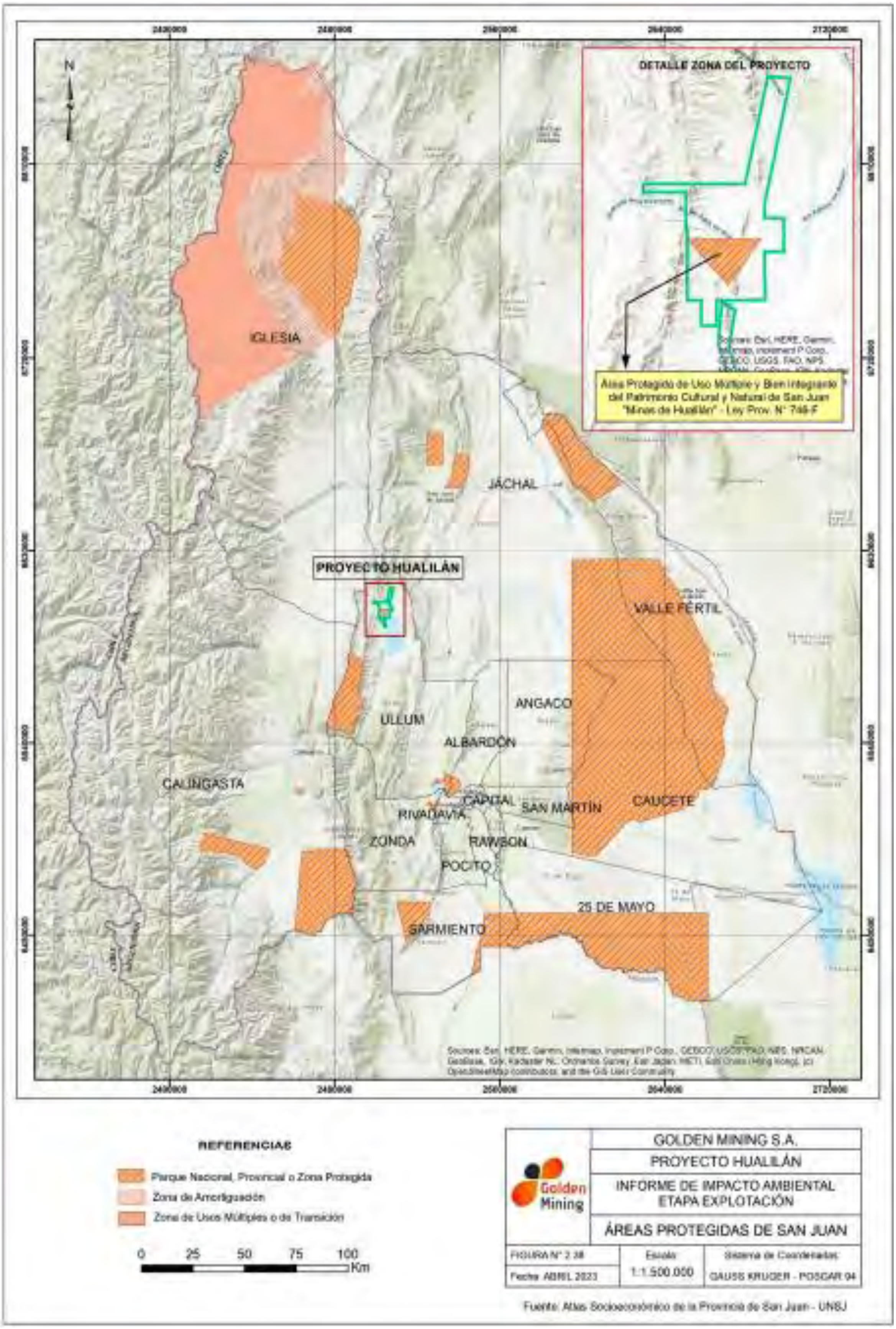

Figure 19-15: Protected areas in the area of influence of the project

However, it is pertinent to mention two of them due to their proximity to the existing access roads that connect the towns that are part of the social study area (Villa Ibáñez, in the Department of Ullum and the towns of Villa Iglesia, Las Flores and Rodeo in the Department of Iglesia, respectively).

Lomas de Las Tapias Natural Park and Protected Landscape, created by Provincial Law No. 729-L, is located in the Ullum Department, at a distance of approximately 20 km from the city of San Juan and 7.5 km, as the crow flies, from Villa Ibáñez. It covers an area of 5,000 hectares.

The objective of this NPA is to preserve a typical landscape of huaquearías and the paleontological site of vertebrate fossils of Chasiquense Mammal from the Middle Pliocene.

Estancia Don Carmelo Private Multiple Use Reserve, created by Provincial Decree No. 1,220/93, is located approximately 130 km from the city of San Juan and 15 km, as the crow flies, from the centre of the Hualilan project.

It covers an area of about 40,000 hectares that extends from the San Juan River on the south, the Sierra del Tigre on the west, the Sierra de La Invernada on the east and an imaginary line that runs from E - W to the height of the Cerro Morterito on the north.

The objective of this NPA is to conserve specimens of the Monte, Puna and Cuyo District eco-regions of the High Andean Province and the fauna represented by the Andean condor, choique, black-chested eagle, peregrine falcon, guanaco, red fox, puma, Andean hare, chinchillón, lizards and Andean toad.

19.6 Potential Emissions, Waste and Effluents Generated by the Project

19.6.1 Atmospheric Emissions

Table 19-6 presents a summary of the emissions into the atmosphere that will be generated as result of the mining of the project. This table specifies: the source of origin, the quality, flow or emission, variability and its management (control and discharge method).

The emissions that are identified for both the construction and operation stages of the project include:

- Particulate matter
- Engine combustion gases
- Blasting gases
- Gases from the process plant.

Due to the nature of the project, emissions into the atmosphere will mainly be of material particulate and gases.

Table 19-6: Potential Emissions

Emission Type	Stage	Source	Characteristics	
			Quality	Flow
Particulate Matter	Construction, Operation, Closure	Dust generation through traffic, mining activities and wind	Particulates <30µm	Variable
Vehicle Emissions	Construction, Operation, Closure	Internal combustion engines	CO_2, CO, NOx	Minor
Emissions from diesel generators	Operation	Internal combustion engines	CO_2, CO, NOx	Minor
Blasting gases	Operation	Blasting	CO_2, CO, NOx, SO_4	Minor
Process plant emissions	Operation	Plant operation	TBD	Minor
Laboratory – Sample Prep	Operation	Crushing and classification	TBD	Minor
Laboratory – Assay Process	Operation	Assays	TBD	Minor

19.6.2 Noise and Vibrations

During the operation stage, noise will be generated mainly by the mining of the deposit and the operation of the process plant. In the first case, it will come mainly from blasting and truck traffic, as well as vehicles that transit to and from the project. In the process plant, the noise will be generated mainly due to the operation of the primary crusher and ore mill. Auxiliary equipment, such as generators, electrical devices, compressors and pumps, will also be sources of noise. Due to their location and the measures that will be implemented for noise control, its influence will be important in the workplace.

Blasts are a particular case: they constitute a source of noise of short duration (few seconds) but of great intensity that, depending on the amount of explosive that is used and the detonation sequence, can reach levels of 140 dBA within the pit area. The mining of the project considers the number of blasts per day and the sound pressure level that would be similar to the indicated value.

The operation of mining trucks constitutes another important source of noise, which unlike blasting, it will emit noise intermittently. The sound pressure level that is generated, will depend on the speed of circulation and whether they transit empty or loaded. It is expected that the equivalent noise level contributed by the traffic of these trucks does not exceed 85 dBA.

Crushing and grinding is also a source of significant noise generation, which can achieve noise pressure levels not exceeding 120 dBA.

19.6.2.1 Noise and Vibration Control

The measures that will be implemented to attenuate noise and vibrations due to the ore blasting include:

i. The operators in this area of operation must use mandatory personal hearing protection equipment.
ii. Adequate delays to minimise the propagation of air waves that generate noise and vibrations.

The measures that will be implemented to attenuate noise and vibrations due to traffic of trucks include the installation of noise silencers in the exhaust pipes.

19.6.2.2 Conclusions

The noise generated by the mining operation and the plant will have its effect only locally. Calculations predict that at more than 2 km the increase in noise from the plant will be imperceptible with an increase less than 3 dBA.

At 2 km, the noise level from blasting would be approximately 125 dB. Although this noise will be of high intensity, its duration will be a few seconds.

Modelling indicates that the particle velocity caused by blasting will be below the residential structure safety criteria recommended by the U.S. Bureau of Mines (2 in/s, or 50.8 mm/s). According to this value, it is expected that the vibrations due to the blasting will have a local influence.

In view of the relative geographic location of the Hualilan project, it is not considered at present that analysis of the potential incidence of increases in noise and vibrations on populations or settlers in particular is warranted. Additionally, with the results obtained from noise modelling, it is expected that there will be no effects on the local fauna existing in the "project footprint".

19.6.3 Mine Waste

Mining waste is derived from the exploration and mining stages from extraction, beneficiation and/or mineral and rock processing operations. This type of waste will be produced during the construction and operation, being represented by:

- Unmineralised material that will be removed when the uncovering and mining of the pits is carried out and in the framework of the construction of the facilities of the mining-industrial complex.
- Tailings or tailings resulting from the mineral beneficiation process at the processing plant.

Both types of waste are included in the definition of mining waste (subsections a and b of point ii. Types of mining waste) of the document prepared by the Secretary of Mining of Argentina called "Rational Management of Mining Waste. General guidelines for rational management of mining waste throughout the life cycle of a mine" according to Resolution No. 181/2021 published in the Official Gazette of the Nation on 06-24-21 (Secretary of Mining of Argentina, 2021).

19.6.3.1 ***Static environmental testing***

Both the non-mineralised waste and the tails have been evaluated to determine their degree of reactivity and geochemical behaviour in the long term. The first evaluation, carried out by the Mining Research Institute of the Faculty of Engineering of National University of San Juan (IIM - UNSJ), consisted of the geochemical characterisation of 14 samples representative of the waste through the application of static methods (ABA and NAG tests), which determine qualitatively if the material is capable of generating acidity or not. Geochemical characterisation tests were carried out on two (2) representative samples of the process tails.

In addition, studies carried out measurements of pH in paste and solubility tests. The samples from exploration boreholes correspond to volcanic rocks of dacitic composition and clastic sediments (sandstones and shales).

Additional static environmental testing was undertaken at SGS Laboratories in Lakefield Canada in the form of modified acid base accounting (ABA) and net acid generation (NAG) tests out on the rougher tailings of the High grade A (≥1.5 g/t Au, ≥1.5% Zn) and Low grade intrusion-hosted (<1.5 g/t Au, <1.5 % Zn) composite.

The ABA test (Acid-Base Balance) involves a set of tests to determine the potential for generating acid (PA) and the neutralisation potential (PN) of the sample. The relationship between PN and PA is used to calculate the potential rate of neutralisation (TPN), which is compared to standard criteria to provide an indicator of the theoretical acid generating potential of a sample.

The TPN values (PN/PA) constitute a criterion for evaluating the results of the acid count – widely accepted basis in the literature. TPN values >3 indicate low to no potential for generation of acidic waters; TPN values between 1 and 3 indicate an uncertain potential for acid generation and TPN values <1 indicate high potential for generating acidic waters (Ministry of Mining – Mining Council, 2002).

The NAG (Net Acidity Generation) test aims to replicate, at a laboratory level, the process of natural weathering and oxidation suffered by rocks and minerals exposed to the elements. It is based on the addition of hydrogen peroxide to the sample in order to oxidize the sulphides.

19.6.3.2 *Static environmental test results – unmineralised waste*

Samples with a NAG pH >4.5 are defined as NPGA (No Acid Generating Potential) and samples with a NAG pH <4.5 are defined as PGA (Acid Generating Potential).

The results of the 14 samples obtained from the static tests are listed below:

- 1 sample was PGA according to both tests
- 8 samples were NPGA according to both tests
- 5 samples were PGAI according to the ABA test (TPN <3), that is, the alkaline elements are not sufficient to neutralise the acidity generated and PGA according to the NAG test.

The pH in paste is an indicator of the total acidity stored in the sample, as well as the extent in which this occurred. The paste pH values of all the samples evaluated showed a Basic pH, that is, at the time of analysis they did not contain acidity due to oxidation (pH>5.5).

Solubility tests, in shaken bottles, indicate low sulphate values for all the samples. The dilution of heavy metals, in the 14 samples, does not exceed the guideline levels of quality of water for human drinking established in Table No. 1 of Annex IV of the National Law No. 24,585/95.

Regarding the dilution of mercury (Hg), it should be noted that, although it exceeds the water quality guide levels for the protection of aquatic life in surface freshwater established in Table No. 2 of Annex IV of National Law No. 24,585/95, it does not apply considering the characteristics of hydrology at surface of the "project footprint" where the drainage network is made up of regular temporary channels).

Challenger plans to increase the number of samples, belonging to different sectors of the main geological domains, in order to carry out static tests. Those samples that have registered PGA and/or PGAI will undergo kinetic tests (wet cell test) for the purposes of determining the rate of acid generation and the variation, over time, of the quality of the filtered water.

19.6.3.3 *Static environmental test results - tailings*

The NAG test results conducted by the (IIM - UNSJ) suggest that both samples are potentially non-acid generating. However ABA (Acid-based Accounting) testing classifies only the F10 Ro Tails sample (flotation tails from the intrusion-hosted mineralisation) as potentially non-acid generating. Appendix 6 – SGS Report explains this in more detail.

The results of the NAG tests conducted by SGS in Lakefield are presented in Table 19-7. Since the final pH of the two samples exceeded 7.0 without the addition of sodium hydroxide, the results of the NAG test suggest that the two samples may not be acid generating.

Table 19-7: Results of Net Acid Generation Tests

Sample ID	Final pH	Vol NaOH to pH 4.5 mL	Vol NaOH to pH 7.0 mL	NAG (pH 4.5) kg H_2SO_4/t	NAG (pH 7.0) kg H_2SO_4/t
F28 Zn Ro Tails (HG B Comp)	8.63	0	0	0	0
F10 Ro Tails (low grade Comp)	10.1	0	0	0	0

The results of the modified ABA tests are shown in Table 19-8. The high sulphur content in the High grade B tailings resulted in an acid generation potential (AP) requiring 215 t $CaCO_3$/1000 t of tailings to neutralise the acid. The neutralising potential (NP) of the sample was 245 t $CaCO_3$/1000 t of tailings. The resulting NP/AP ratio of 1.14 is considered too low to classify the High grade B rougher tailings as non-acid generating. In contrast, the rougher tailings of the Low grade composite yielded a NP/AP ratio of 32.2, which clearly classify this stream as non-acid generating. It should be noted that the NAG and ABA test are static tests, and the results should only be considered indicative. Dynamic environmental tests such as humidity cell tests will be required to properly quantify the acid generation potential of the tailings and possible leaching of elements from the sample.

Table 19-8: Results of Modified Acid Base Accounting Tests

Sample ID	NP t $CaCO_3$ /1000 t	AP t $CaCO_3$ /1000 t	Net NP t $CaCO_3$ /1000 t	NP/AP Ratio	S %	Acid Leachable SO_4 - S %	Sulphide %	C %	CO_3 (HCl) %
F28 Zn Ro Tails (HG B Comp)	245	215	29.8	1.14	7.22	0.34	6.88	3.29	16.3
F10 Ro Tails (low grade Comp)	40.3	1.25	39.0	32.2	0.039	<0.04	<0.04	0.64	3.08

19.6.4 Industrial Waste

Industrial waste (as defined in article 2 of National Law No. 25,612 Management Comprehensive Industrial Waste and Service Activities) is any element, substance or object, in solid, semi-solid, liquid or gaseous state, obtained as a result of an activity of service or for being directly or indirectly related to the project. Industrial waste is classified, according to its hazard, as hazardous and non-hazardous depending on whether they present any of the dangerous characteristics included in Annexes I and II of National Law No. 24,051 on Hazardous Waste to which the Province of San Juan adheres through Provincial Law No. 522-L.

Industrial waste will be generated mainly during the operation stage of the project, in activities related to auxiliary operations, equipment maintenance and machinery used in mine operations and in the mineral treatment process.

Industrial waste will also be generated in the warehouse sector and to a lesser extent in the chemical - metallurgical laboratory. Waste will also be generated during construction and closure tasks but in smaller volume.

Industrial waste classified as non-hazardous and recoverable or recyclable (paper, cardboard, plastic, glass, wood, scrap metal, etc.) will be delivered to an Environmental Technologies Park (P.T.A.) or to an authorised recycler.

This category includes: used tyres from mining equipment and light vehicles, which will be delivered to an authorised operator (recycler); and the debris that results from the dismantling and demolition tasks of mining infrastructure. The latter type of waste will be generated during the mine closure stage and will possibly be used as fill in sectors to be infilled or, failing that, it will be placed in the waste dumps.

Among the main hazardous waste, examples are empty pressurised gas tubes, containers with traces of organic and inorganic chemicals, contaminated metal parts, batteries, Waste Electrical and Electronic Equipment ("WEEE"), Solder Scraps, Solids contaminated with hydrocarbons, used oils and lubricants, aqueous-based solutions with hydrocarbons, engine filters, sludge or mud, earth, debris and refractory material, waste laboratory, etc.

Hazardous industrial waste generated during the construction, operation and closure stages will be collected in a waste yard designated for this purpose in the mining-industrial complex. The yard must have a waterproof base, pit or secondary containment pool, fence or fence perimeter, it will be roofed, with security signage and fire-fighting elements compatible with the fire load.

Hazardous waste will be managed – transport + final disposal – in accordance with the provisions in Provincial Law No. 522-L and Regulatory Decree No. 1,211/07, through carriers and operators authorised by the Secretary of State for the Environment and Sustainable Development of San Juan and/or Ministry of Environment and Sustainable Development of the Nation respectively.

More specific Industrial Waste considerations are outlined below.

*19.6.4.1 **Oils and lubricants***

In the mining stage, the use of heavy machinery and equipment will be intensified, in relation to that used during exploration or construction. The equipment used will remain in the operations and construction areas. Maintenance regarding oil changes, controls of grease and oil levels and other preventive measures, will be carried out in the workshop that will be set up for this purpose.

The management plan, which will be implemented for this type of effluent, includes the recovery of the used oils and lubricants, their temporary storage and periodic removal by an operator authorised by the provincial environmental authority for the purposes of proceeding to its final disposal via thermo-destruction, after which a final disposal certificate will be issued for the purposes of traceability of this type of waste.

For the temporary storage of used oils, a tank or cistern will be enabled, which will be installed in the maintenance area. The tank or cistern will be provided with a pan of security to contain a volume equivalent to 110% of its capacity, it will have the corresponding signs indicating its content. The facility will have a fire protection compatible with the fire load of the tank or cistern. This tank will also store used oils and hydraulic fluids that are recovered in maintenance of on-site machinery such as crushers, generator sets and engines.

*19.6.4.2 **Mechanical maintenance wash water***

The washing of both surface and underground mining trucks, among other mobile machinery, is an activity that will be carried out during preventive maintenance regularly. The washing operation will be carried out in an area specially conditioned for this task.

For the management of washing water, the techniques and procedures followed will be those used usually in the industry. The washing area will have an impermeable concrete base to contain the washing water and prevent it from infiltrating into the ground. The wash deck will be built with a slight slope that will allow the resulting water to be drained to a well or ditch collection where the solids will be decanted.

The clear waters will continue to a chamber, where due to the difference in density, the fats will be separated. and oils from water. Periodically, the solids of the well (sludge or mud) and the grease and chamber oils will be mucked out. The water will be reused in washing equipment.

The solids (sludge or mud) will be temporarily disposed of in a waterproofed drying area which will have a drainage system, in order to reduce by evaporation, the content of water. Finally, they will be collected in the hazardous waste yard, in the case of contaminated soils with hydrocarbons.

The grease and oils that are recovered from the chamber will be disposed of in the storage tank or cistern storage of used oils, which will be managed as hazardous waste (current of waste Y8 or Y9) receiving final disposal via thermo-destruction in an operator authorised by the provincial environmental authority.

19.6.4.3 ***Laboratory wash water***

During the operation of the project, an analytical and metallurgical laboratory will be installed to control the quality of the different process flows. As a result, it will periodically be necessary to wash the containers and elements in which the tests are carried out and there will be a liquid residue that will contain various dissolved chemicals used in analytical procedures.

The precise quantity and quality of this residue is unknown. However, the volumes generated will be small, approximately 30 m^3 per month, compared to the volumes being managed in the processing plant. Therefore, their management considers their collection and their reuse in the mineral treatment process.

19.6.5 Residential Waste

Residential wastes fall into the category of waste comparable to urban solid waste (MSW) as defined in paragraph d of art. 7 of Provincial Law No. 1114-L and in accordance with what is regulated in the Law National No. 25,916 Comprehensive Management of Household Waste.

They will be generated mainly in the dining room, administrative offices and in health services. Solid domestic waste will also be generated, although in smaller quantities, in the industrial and mining facilities.

During the construction, operation and closure stages of the project, a comprehensive plan will be implemented for waste management that will include collection, classification, temporary storage and transfer for the final disposal of domestic waste.

The types of waste that fall into this category include: food scraps, papers, cardboard, plastic, rubber, wood, glass, cans, dirt or dust resulting from cleaning or cleaning tasks, sewage, etc.

A waste yard for temporary storage will be available in the mining-industrial complex. These wastes will be differentiated into recoverable and non-recoverable. The transportation of solid domestic waste – recoverable and non-recoverable – will be carried out by transporters authorised by the Secretary of State for Environment and Development Sustainable (SEAyDS) and its final disposal will take place in an Environmental Technologies Park (P.T.A.) or a recycler authorised by the SEAyDS.

The generation rate of domestic solid waste is variable and depends on the quantity of personnel involved in the work. An average generation rate of 0.50 kg/person/day, has been assumed allowing the following quantities of domestic waste to be estimated – depending on the employed personnel:

- Construction stage: 175 kg/day
- Operation stage: 120 kg/day
- Cessation stage: 25 kg/day.

More specific Domestic Waste considerations are outlined below.

19.6.6 Sewage

Sewage effluent treatment (PTEC), which must be authorised by the Department of Hydraulics and whose discharge parameters must verify the maximum established in the Decree Nº 2.107-MIyT-07, regulating Provincial Law Nº 348-L.

During the construction, operation and closure phase of the project, sewage will be generated mainly in the camp toilets, administrative offices, industrial facilities and by those located in the work fronts.

A supply of 200 L/day/person was considered, with a recovery of 80% and an endowment of staff for the construction, operation and closing stages of 350, 240 and 50 people respectively. The sewage and domestic effluents generated in the camp toilets, offices administrative, kitchen and industrial facilities, will be collected and taken to processing plants.

In the construction stage, it is planned to use chemical toilets on the work fronts. Residual liquid from these chemical baths will be collected and will receive final disposal in a waste treatment plant.

The sewage generated in the kitchen of the dining room during the construction stages, operation and closure will pass through grease separators (traps) before entering the production plant treatment.

The sludge or sludge generated in the effluent treatment plants (PTEC) will be analysed and depending on their characterization, they will be treated as urban solid waste (MSW) or as waste hazardous.

If it is classified as urban solid waste (MSW), it will receive final disposal in a Technology Park Environmental (P.T.A.), authorised by the Secretary of State for the Environment and Sustainable Development, while if it is a hazardous waste it will receive final disposal via thermo-destruction in by an operator authorized by the provincial environmental authority.

19.6.6.1 ***Pathogenic residues***

It is expected that the infirmary will generate waste during the construction and operation stages. Pathogenic, represented by materials and implements already used, such as gauze, syringes, medicine containers, gloves, etc. The estimated generation rate of this type of waste will reach 0.02 kg per day.

The handling of these includes the separation of the sharp elements and their immediate disposal in containers specially prepared for this, provided by the operator in charge of the final disposal of these waste streams. This type of waste will be managed – transportation + final disposal – as regulated in Provincial No. 522-L and Regulatory Decree No. 1,211/07

19.7 Environmental Liabilities

There are no environmental liabilities associated with the project.

19.8 Closure and Abandonment Stage

Among the main objectives of the closure plan for the Hualilan project, the following can be mentioned:

- To achieve requirements for human safety, such as through the permanent sealing of surface exits that connect with the underground workings (portals, shafts, ventilation shafts)
- Comply with the regulatory requirements and commitments assumed by Challenger for the closure of the Hualilan project.
- Achieve a site closure condition that protects the environment and public safety.
- Achieve physical and chemical stability of those sectors of the project that have been impacted or modified by mining activities.
- Rehabilitate those areas of the mining-industrial complex that have not been intervened by permanent works and integrate into the landscape, as far as possible, those sectors that have been intervened by permanent works at the site.
- Ensure that the post-closure socio-economic and community relations requirements for the social study area are satisfactory.
- Avoid and/or limit the need to opt for an active scenario at the time of abandonment of the operation. In other words, the objective is to achieve a state that minimises the maintenance or operation - of a permanent nature - of some of the technical components of the site after the moment of abandonment.

The closure plan presented is of a conceptual nature and the final closure plan will be prepared and submitted to the mining environmental authority with the First Update of the Exploitation EIA.

19.8.1 Closure Plan

The conceptual closure objectives and activities, for each of the components of the mining operation, are described below in addition to the activities to be carried out in the post-closure stage.

*19.8.1.1 **Open pits***

Warning signs and a perimeter fence made of rock or rockfill shall be placed around the perimeter of the pits to prevent risks and the accidental entry of people and/or animals of a certain size.

All equipment involved in open pit mining will be removed and disposed of at a site designated for that purpose or, failing that, these assets will be sold.

The post-closure monitoring period will include frequent visual inspections of slopes and ridges for a period of four years from the end of the operation, progressively decreasing until no unfavourable physical or environmental conditions are detected.

*19.8.1.2 **Underground***

All ancillary equipment and facilities - water, air, transport, communication systems - will be removed, dismantled and disposed of at a site designated for this purpose or, failing this, these assets will be sold.

The monitoring of rock plugs or permanent seals of underground workings will include frequent visual inspections for a period of four years from the end of the operation, which will be progressively reduced as geotechnical and geochemical stability conditions of the plugs or seals are verified.

Georeferenced maps will be generated with the location of workings and signage of blocked access activities. This information must be published and updated to the mining authority.

*19.8.1.3 **Mine service facilities***

Closure activities include the removal of machinery, equipment and utility structures by Challenger Gold or its contractors. Reinforced concrete foundations - floors and slabs - will be demolished and the waste or rubble will be deposited in waste dumps. The plastic membranes and/or geomembranes used for containment purposes will be removed and, depending on whether or not they are hazardous waste, they will be disposed of at an Environmental Technologies Park in the case of solid urban waste or via thermo-destruction at an authorised facility in the case of hazardous waste.

Mine service facilities that have been closed will be monitored by frequent visual inspections during the initial post-closure period. No additional inspections should be required after the fourth year after completion of the work, assuming that no unfavourable conditions have been detected during the initial monitoring period.

*19.8.1.4 **Waste dump facilities***

Long-term physical and chemical stability will be ensured and waste dumps will be integrated into the existing relief in the best possible way.

At the end of the operation the dump slopes will be re-profiled at an angle of 24° (2.25H:1V). The construction of a self-evaporative cover is also suggested, which will both help to reduce the potential for wind erosion and to manage summer storm events which can be of high intensity.

Post-closure monitoring will include groundwater quality and air quality measurements at gradient points above and below this structure. At the same time, geotechnical surveys will also be conducted on the slopes and ridges of the waste rock dumps.

*19.8.1.5 **Tailing storage facility***

A self-evaporative cover will be placed on the TSF to manage direct precipitation using non-reactive pit waste. The geomembrane used as protection in the water retention sump and deposited solids will be removed; the sump will be backfilled with suitable material, as will the diversion channels constructed for this structure.

Post-closure monitoring will include groundwater quality and air quality measurements at gradient points above and below this structure and geotechnical inspections of the slopes and their crest will also be conducted.

*19.8.1.6 **Processing plant and stockpile***

The soils contaminated by the processing plant will be excavated and subsequently removed to the spoil heap or to an acceptable location where they will be permanently disposed of.

The area remaining after excavation will be backfilled with soil or earth material, which will be spread and compacted as much as required to develop a final closure surface compatible with the existing relief.

Plant closure will include dismantling and removal of all structures, equipment and materials. Foundations including floors and slabs will be demolished and the waste or debris deposited in the landfill or other suitable permanent disposal location.

Soil contaminated by hydrocarbons will be removed and deposited - on a temporary basis - in the hazardous waste yard, for subsequent disposal via thermo-destruction by a facility authorised by the Secretary of State for the Environment and Sustainable Development.

19.8.1.7 ***Offices, energy supply and related infrastructure***

Closure activities for these facilities include dismantling and removal of structures, equipment and materials. Tanks or cisterns will be drained, rinsed, dismantled and removed from the site. Floors and slabs shall be demolished, and waste or debris placed in the dump or other acceptable facility for permanent disposal.

Soil that could become contaminated, in whole or in part, with hydrocarbons will be removed and deposited - on a temporary basis - in the hazardous waste yard, for subsequent disposal via thermo-destruction by a facility approved by the Secretary of State for the Environment and Sustainable Development.

Uncontaminated soils will be deposited in landfills. The remaining areas will be filled and compacted according to landscape conditions.

19.8.2 Post-closing Stage or Abandonment

In general, geotechnical, geochemical, biological, environmental and socio-economic monitoring programmes implemented during the operational period should continue during the closure stage and be updated as required in the post-closure period.

Requirements for post-closure monitoring, including methods, frequencies, durations and site sampling may be modified based on trends detected during the mining operation or closure period and/or according to regulatory requirements or requests.

In this regard, these requirements will be defined during the detailed engineering phase with emphasis on:

- Facilities and areas used for handling or storage of hazardous materials such as process solutions and hydrocarbons.
- Physical stability of large slopes.
- Potential generation of acid rock drainage.
- Surface water control structures.

Initial post-closure monitoring will be conducted twice a year for four years after completion of the closure works. Based on the results of the initial post-closure monitoring the control routine would be decreased and gradually phased out, subject to validation by the mining environmental authority.

20 SOCIAL AND COMMUNITY IMPACT

20.1 Area History

San Juan was settled in the late 16th and early 17th centuries by small numbers of Spanish agriculturists, Dominicans and Jesuits from Chile. A part of the old Cuyo region, it remained a sparsely populated area exporting wine and dried fruits that were produced in its irrigated valleys. In 1776, control over San Juan passed from the Chilean captaincy general to the Viceroyalty of the Río de la Plata. San Juan declared its own status as a province in 1825.

The first settlers of the area that today is known as the Ullum department, were the aborigines of the Ullum-Zonda community, at the beginning of the first millennium. This was an area dominated by the Spaniards without other population settlements of interest, from the foundation of San Juan until the independence of Argentina.

From its beginnings, the San Juan economy was based mainly on dispersed agricultural activities and the occupation of the territory was slow. The search for gold and silver began in the period of Inca occupation and gained momentum at the end of the 19th century, with mining at Hualilan. Subsequently mining has become an important part of the local economy with San Juan the largest supplier of limestone in the country and the province hosting several large gold and base metal mines.

20.2 Towns and Villages in Area

The Hualilan project is located in the Ullum Department in the northwest of the Capital city of San Juan. The Ullum department has an area of 4,391 km^2 and its population is 6,463 according to the last Census carried out in the year 2022.

The nearest towns and villages to the project are all located to the north of the project on National Highway 149. These are:

- Iglesia located 44 kilometres on national route 149 is the nearest town to the Project. Iglesia has a population of 366. It offers tourist accommodation.
- Las Flores, located 51 kilometres north of the Project on national route 149, has a population of 900. Los Flores offers a petrol station and supermarket, accommodation and restaurant.
- Rodeo located an additional 17 km north of las Flores on route 149 is the nearest large town to the Project. With a population of approximately 5,000, Rodeo offers a far greater range of services including medical facilities and a police station.

The main population centres south of the project are:

- Villa Ibáñez, Ullum located 109 km along RP No. 54 and then No. 40 – Route No. 149
- Capital, San Juan located 121 km along Route Nº 40 and then Route Nº 149

Villa Ibáñez and San Juan City are the two populated centres with political and institutional links to the Project.

20.3 Stakeholders

20,000 Ha of land surrounding and including the Project has been acquired by the Companies 100% owned Argentinian operating subsidiary Golden Mining S.R.L. The primary stakeholders are inhabitants of the Ullum Department and the Province of San Juan. Under Argentine Mining Law all royalties flow to the Province and mining projects are governed at the Provincial level. The other key stakeholder is the Federal Government which benefits via tax revenue and export duties.

20.4 Local Human Resources and Skill Levels

The project area is located a 1.5 hour drive by double laned sealed highway from San Juan City, the capital of the province with a population of over 1 million people. A pool of high-quality labour, both university trained and non-university trained, is available in San Juan.

San Juan city is a major gold and base metal mining hub in Argentina. It contains head offices of many mining service providers including drilling companies, assay laboratories, geophysical contractors, and mining contractors. The province has also seen two of its major gold mines Casposo (which has been on care and maintenance since 2019) and Gualcamayo (which is currently treating low grade stockpile material and has submitted a closure report for approval) reach the end of their mine life which has created a pool of skilled San Juan based mine workers. Both operations were initially open pit mines before transitioning to underground mining thus this pool of skilled mining labour includes both open pit and underground miners/drillers, treatment plant workers and maintenance mechanics. Additionally, Barrick Gold's Veladero Mine has undertaken several rounds of cost reduction via staffing reductions further deepening the pool of skilled mining workers available in the province.

Additionally, San Juan City contains the San Juan School of Mines which is a high-quality university specialising in mining. Via its partnership with the San Juan School of Mines the Company has rotated many of the students through its operation to provide work experience and have been impressed with their training and level of knowledge. This University provides well trained graduates in mining-related disciplines.

20.5 Community Needs and Expectations

Through the conducting of in-depth interviews the opinions, perceptions and expectations of community representatives of the Ullum department were gathered. Eight representatives of relevant sectors of the community were interviewed, namely: Entrepreneurs, Social Organisations, Commerce, Agribusiness, Sports, Education, Health and Local Politics.

The main community needs and problems identified by those surveyed were:

- health services provided in the Department
- drinking water service
- unemployment
- insecurity
- public lighting and electricity
- lack of job opportunities
- paving and improvement of streets
- waste collection
- educational services

Many of these needs are related to the basic services and infrastructure of the communities that are generally taken care of by the municipality, while others are of a structural nature and require sustained public policies to be minimised or eliminated.

The highest ranked need identified by the survey respondents is health services, infrastructure, supplies and medical care, which were the aspects they highlighted as needing improvement.

Additionally, the interviewee's perception of the difficulties, needs, changes, inequalities, expectations and opinions about the Hualilan project were examined in depth.

The interviewees mentioned as significant the loss of work culture and cultural identity, the presence of non-native labour force and the lack of stable employment, which is the identity axis of people and therefore of daily life. They also indicated the lack of opportunities, addictions, lack of security and lack of future expectations, as well as the need for improvements in all public services.

Regarding the labour market, they emphasised the need for workers to have more stable jobs to improve their living conditions. They also mentioned economic activities that require promotion and support in the department, such as tourism, agribusiness and gastronomy. There was agreement that the Hualilan mining project raises the community's expectations due to the possibility of the new sources of employment that it would generate.

Regarding government service and infrastructure the following were identified:

- the need for basic infrastructure works
- economic support to the different sectors
- investment and labour training.

Expectations in relation to the Hualilan project's impact on services and infrastructure are high, as they consider that support could be received from the company.

Regarding future expectations for the Ullum department, the community leaders foresee possibilities of growth, generation of new jobs and strengthening and/or exploitation of other economic activities, for which they consider it a priority to overcome certain limitations in their community such as: the prevailing work culture and conformism of the people, training and education of human resources and access to quality basic services for its inhabitants.

20.6 Community Relations Program, Assistance and Projects

20.6.1 Community Relations Program

Challenger has developed a community relations program, which began to be implemented during the exploration stage and will continue throughout the useful life of the mine.

The main objective of this program is to generate two-way communication channels between the company and the community, particularly from the Ullum Department, for the purposes of:

- Informing the community about the environmental impacts of the project in the stages of construction, operation and closure.
- Prevention and mitigation measures for those impacts.
- The opportunities generated by the new activity to be located in the area.
- Managing community expectations and concerns.

In general terms, this program considers the execution of:

- Citizen Participation Plan (hereinafter, PPC)
- Communications and Disclosure Plan (hereinafter, PCD)
- Community Development Support Plan (hereinafter, PADC).

20.6.1.1 ***Citizen Participation Plan (PPC)***

The PPC was carried out through two stages: one of diagnosis and another in which the project was presented to community representatives of the Ullum Department.

The presentations of the project consisted of meetings by interest groups among which representatives of the state, political, economic and/or productive sectors could be distinguished, along with press, religious institutions and social organisations.

To prepare the project presentation meetings, the following were carried out:

- Identify the community stakeholders interested in the project
- Identify perceptions about the project.

With the information obtained, a dissemination plan for the project was defined, which responds to different objectives, among which can be mentioned:

- Regularly disseminate information to the community, which allows developing perceptions and realistic expectations about the project.
- Provide an opportunity to local communities and interested citizens to express their opinions and concerns in relation to the project.
- Respond to stakeholder concerns and address these concerns and problems arising to the extent practicable and reasonable.

The information provided by the community will be kept for the purposes of promoting the positive effects of the project and mitigate the negative effects, through knowledge of the community's concerns about the project. The information received has served to propose measures to mitigate negative impacts.

The surveys and meetings with the different stakeholders from the community of the Ullum Department, were carried out in the months of February and March 2023. These will be ongoing during the future stages of the project.

20.6.1.2 ***Communications and Dissemination Plan (PCD)***

The objective of this stage will be to continue the Citizen Participation activities so that the consultation and dissemination are procedures applied to the development of the project.

This plan will be carried out through periodic meetings with the community of the social study area of the project, during the construction, operation and closure stages, in order to keep the public informed about the development of the project and aspects related to it, as well as anticipate conflict situations and look for ways to address and resolve them.

The information meetings that are scheduled regarding the progress of the project will continue. In principle, it is estimated that, during its construction, a monthly cycle of meetings will be held and during the production stage the Company will carry out a cycle of annual meetings. Considering that the beginning of construction produces great expectations, it is estimated that it will be necessary to hold meetings prior to the start of activities and monitor how the relationships between the company, subcontractors and the community are developed, so to foresee conflict situations.

During the life of the project, new developments will inevitably arise, which may merit the call for additional public meetings. A Community Relations Officer will have the responsibility to determine, according to the PCD, when a special circumstance warrants public disclosure and will take appropriate measures to inform in a manner timely to community leaders and the interested public.

Meetings on planning to support community development throughout the life of the mine, will be held with local authorities to address the how Challenger will support community development. The purpose of these meetings will be to talk about the interests and concerns of the community, identify projects that will benefit them, and determine the progress or success of the programs that have been implemented.

Once the agreements or commitments adopted in the meetings are made, it will be necessary to define responsibilities of all parties involved and establish a roadmap for monitoring of agreements.

*20.6.1.3 **Community Development Support Plan (PADC)***

The social license is of particular importance in the development of a mining operation, so the key to obtaining and maintaining it will be adherence to sound social practices.

Challenger is committed to integrating social principles into all aspects of the project, which will specifically lead the development and implementation of a plan to support community development with special emphasis on Department Ullum throughout the life of the project. These principles are:

- Include the evaluation of social, economic and cultural issues in project management.
- Support the personal and professional development of their workforce and their families.
- Promote adherence to responsible social practices by suppliers and contractors.
- Contribute to sustainable community development over time and with the standards of the industry.
- Actively encourage the participation of stakeholders in the establishment and implementation of a social program based on consistent interaction and dialogue.
- Provide the necessary resources to implement their role and monitor the progress of the social program developed in coordination with the community.
- Work with public departments, NGOs, social organisations, interest groups and the community to promote community development.
- Disseminate the content and objectives of the community program to all managers, employees and their families in order to encourage their participation in the program.

Challenger currently supports the community in several ways including:

- 2000 families from the Ullum community benefited from a drinking water well, which represents more than USD$30,000 of investment.
- Geology students trained via our local internship program with San Juan Universities with some assisted via our scholarship program with the San Juan School of Mines

- New local businesses supported via CEL's local supplier development plan in Argentina.
- Locals employed with a 95% retention rate.
- Special needs students were sponsored to participate in the Ullum science fair in Argentina.
- Children benefited from educational programs in Argentina.
- The San Juan Mining Ministry confirms that Challenger Exploration and Barrick Gold are the only exploration companies operating in San Juan that meet the government's +80% local employment quota.

21 MARKET STUDIES AND CONTRACTS

The information summarised in this subsection is derived from a concentrate market study conducted by Reach Partners, a third-party concentrate marketing specialist company based in London. The Company has expanded this work done by Reach with discussions with concentrate off-takers, other concentrate marketing groups, and concentrate traders. As part of these expanded discussions additional indicative concentrate off-take terms were provided by some off-takers. These discussions all supported the data and indicative pricing provide by Reach Partners.

21.1 Product Supply and Demand

The Hualilan project will produce three types of concentrate:

- Gold-Silver concentrate;
- Lead-gold-silver concentrate; and,
- Zinc Concentrate with gold and silver credits.

Additionally, Hualilan will produce gold and silver as dore from the Flotation Tails Leach (FTL).

Pricing for metals contained in the concentrate as well as the gold and silver dore will be based on market price at the time of sales when the concentrate and dore is received at the smelters.

The assumptions made for the purposes of this report include the following:

- The zinc concentrate produced by the mineral processing facility will be sold to smelters in Europe. The gold and silver credits will be payable as metal credits as per normal industry practice. The transportation and insurance costs have been accounted for separately to the Smelting and Refining costs.
- The gold-silver concentrates produced from the mineral processing facility will be sold to smelters in either Asia or Europe. The transportation and insurance costs have been accounted for separately to the Smelting and Refining costs. The payability and transport costs/ refining costs (TC/RCs) have been accounted for in the Smelting and Refining costs for this project.
- The lead-gold-silver concentrates produced from the mineral processing facility will be sold to smelters in either Asia or Europe. The transportation and insurance costs have been accounted for separately to the Smelting and Refining costs and payability.
- Gold and Silver produced by the FTL will be sold as dore with payability of 99.5% for gold and 97.5% for silver.
- Gold, zinc, silver, and lead are readily traded commodities and the sales terms for them are generally standard in nature. For the purposes of this study, it is assumed that the products will be sold freely and at standard market rates.

The metal price assumptions used in this Scoping Study are based on consensus pricing from a number of banking institutions to arrive at a reasonable long-term estimate. Metal price estimates are considered conservative and are based on supply and demand fundamentals.

The metal prices used in the economic evaluation of this project are summarised in Table 21-1 and compared to rolling 5-year average and spot prices.

Table 21-1: Commodity Price Assumptions

Assumptions used in this Scoping Study	Rolling 5 Year Average (from September 2018)	Spot Price (November)
Gold – US$1750 oz	$1710	$1980
Silver – US$20 oz	$20.72	$23.00
Zinc – US$1.15 lb	$1.28	$1.15
Lead – US$0.94b	$0.93	$0.98

21.2 Product Specifications

Detailed specifications of the concentrates produced at Hualilan are outlined in this section. The project is expected to produce approximately 116,000 ounces of gold, 440,000 ounces of silver and 9,175 tons of zinc, and 474 tonnes of lead over seven years of mine life under the high grade case evaluated in the Scoping Study.

The gold, silver, zinc, and lead produced will be spread across the various concentrates and gold/ silver dore as outlined in Table 21-2. Payable metals in each concentrate are highlighted in bold.

Table 21-2: Summary of Concentrate Streams

Product	Au (g/t)	Ag (g/t)	Zn (%t)	Pb (%t)	Cu (%)	Fe (%t)	S (%)
Zn-Au-Ag concentrate	**5-10**	**180**	**50**	0.3	1.0	11.5	34
Pb-Au-Ag concentrate	**150**	**700**	5.9	**63**	0.3	7.1	15.5
Au-Ag concentrate	**55-65**	**250**	3.5	1.1	0.8	30	41

The detailed composition of the various concentrates is summarised in Table 21-3 and outlined in the sections following.

Table 21-3: Summary of concentrate off-take terms

Product	Payability (%)	TC/RC (US$/t)	Penalties (US$/t)
Zn-Au-Ag concentrate			
Zinc	84	160	7
Gold	63		

Product	Payability (%)	TC/RC (US$/t)	Penalties (US$/t)
Silver	35		
Pb-Au-Ag concentrate			
Lead	95	125	nil
Gold	95		
Silver	95		
Au-Ag concentrate	92.9		
Gold	95	100	nil
Silver	60		

21.2.1 Zn-Au-Ag Concentrate

The Zn-Au-Ag concentrate is produced from the rougher zinc flotation stage (followed by three to four cleaning stages) from Material Type C (≥1.5 g/t Au, ≥1.5% Zn & <1.5 g/t Au, ≥1.5% Zn). Table 21-4 shows the range of composition in the various flotation and flotation variability tests completed. Table 21-5 provides a detailed analysis of the composition of the concentrate. The project will produce approximately 18,350 tonnes of this concentrate annually.

Table 21-4: Zinc Concentrate – key components average composition

Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	S (%)
48-50.5	9.6-12.3	84-178	0.8-1.3	0.1-0.5	33.9-34.6

Table 21-5: Detailed analysis of zinc concentrate (flotation tests F26 and F28)

Zn (%)	Au (g/t)	Ag (g/t)	Cu (%)	Pb (%)	S (%)	F (%)	Cl (%)	Hg(g/t)
50.5	9.6	178	1.33	0.48	33.9	**<0.005**	**<0.01**	**3.2**
Al (g/t)	**As (g/t)**	**Ba (g/t)**	**Be (g/t)**	**Bi (g/t)**	**Ca (g/t)**	**Cd (g/t)**	**Co (g/t)**	**Cr (g/t)**
<400	<30	<3	<0.05	<20	1680	2910	9	<200
Fe (%)	**K (g/t)**	**Li (g/t)**	**Mg (g/t)**	**Mn (g/t)**	**Mo (g/t)**	**Ni (g/t)**	**Sb (g/t)**	**Se (g/t)**
11.5	<214	<30	135	3810	6	26	25	<40
Si (g/t)	**Sn (g/t)**	**Sr (g/t)**	**Ti (g/t)**	**Ti (g/t)**	**U (g/t)**	**V (g/t)**	**Y (g/t)**	**P (g/t)**
1970	<20	<10	45	<30	<400	<4	<8	<200

This concentrate is likely to be sold to the European or the Korean market as it contains significant Au and Ag credits. The Au and Ag payabilities on Zn concentrates sold into China are low, plus the Cd content of this concentrate is approaching the 3000 ppm limit for exports into China.

Typical terms from a buyer will be:

- Payabilities of:
    - Zn: Gross payability of 85%, subject to an 8-unit (%) deduction. Thus, for this concentrate the post deduction Zn payability is 84.2%.
    - Au: One unit (g/t) of Au content shall be deducted from the final Au content and 70% of the balance shall be paid. Thus for this concentrate the post deduction Au payability is 63%.
    - Ag: Three units (oz) of Ag content shall be deducted from the final Ag content and 70% of the balance shall be paid. Thus, for this concentrate the post deduction Ag payability is 35%.
- Current spot TC/RC costs for this concentrate are US$150-$170/t with US$160/t used for budget estimates.
- There is a penalty charge of US$7.0 per Dry Metric Ton (DMT) of concentrate due to Fe content of 11.5% being above the penalty limit. For each 1.0% by which the final Fe assay exceeds 8.0%, seller pays a penalty charge of US$2.0 per DMT of concentrate.
- Cadmium penalties can apply to concentrates with the level of Cd of the Zn-Au-Ag concentrate can be US$1.50/lb Cd however no cadmium penalties are anticipated based on the market study and indicative payabilities provided.
- There are no other deleterious elements in this concentrate.

21.2.2 Pb-Au-Ag Concentrate

The Pb-Au-Ag concentrate is produced from the Cu-Pb rougher Flotation Stage (followed by two cleaning stages) from Material Type C (≥1.5 g/t Au, ≥1.5% Zn & <1.5 g/t Au, ≥1.5% Zn). Table 21-6 shows the range of composition in the various flotation and flotation variability tests completed. Table 21-7 provides a detailed analysis of the composition of the concentrate. The project will produce approximately 950 tonnes of this concentrate annually.

Table 21-6: Lead-Gold Concentrate – key component average composition

Pb (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	S (%)
63.4-64.7	81.8-178	637-765	0.2-0.4	5.6-6.3	15.4-15.9

Table 21-7: Detailed analysis of Pb-Au concentrate (Flotation test F26 and F28)

Pb (%)	Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	S (%)	P (g/t)	P (g/t)	P (g/t)
64.7	178	765	0.41	5.59	15.9	<200	<200	<200
Al (g/t)	**As (g/t)**	**Ba (g/t)**	**Be (g/t)**	**Bi (g/t)**	**Ca (g/t)**	**P (g/t)**	**P (g/t)**	**P (g/t)**
6,140	<30	50	0.27	169	51,500	<200	<200	<200

Fe (%)	K (g/t)	Li (g/t)	Mg (g/t)	Mn (g/t)	Mo (g/t)	Ni (g/t)	Sb (g/t)	Se (g/t)
7.1	2,630	<30	4,000	5,040	18	42	107	<40
Si (g/t)	**Sn (g/t)**	**Sr (g/t)**	**Ti (g/t)**	**Ti (g/t)**	**U (g/t)**	**V (g/t)**	**Y (g/t)**	**P (g/t)**
n/a	<20	47	481	<30	n/a	11	15.4	<200

- This concentrate can be sold globally to smelters in all regions.
- Typical terms from a buyer will be:
  - Pb: Gross payability of 95%, subject to a minimum 3-unit (%) deduction. Thus, for this concentrate the post deduction Pb playability will be 94%.
  - Au: Gross payability of 95% subject to a minimum deduction of 1 gram/dwt hence payability remains 95%
  - Ag: Gross payability of 95% subject to a minimum deduction of 50 grams/dmt which equates to a payability of 92.9%. One off-taker quotation during the market study did not include the 50 gram deduction for Ag
- Indicative TC's for this concentrate inclusive of RC's are approximately US$125/t for this concentrate.
- Additionally, indicative TC's (not including RC's) of US$100/t with additional refining charges of US$15/oz Au and US$1/oz Ag were quoted during the market study.
- This concentrate contains no penalties for any deleterious elements.
- For Budget purposes TC's of US$125/t inclusive of RC's were used.

21.2.3 Au-Ag Concentrates

There will be three separate Au-Ag concentrates produced:

1. From gravity concentrate and various cleaner tails from the Zn and Cu-Pb rougher stages in the Type C material. This will contribute approximately 25% of the Au sold in Au-Ag concentrate.
2. Produced via the combination of the gravity concentrate and the bulk sulphide flotation concentrate (after cleaner stages) from the Type B (Au≥1.5 g/t, Zn <1.5%) material. This will contribute approximately 75% of the Au sold in Au-Ag concentrate.
3. Produced via the combination of the gravity concentrate and the bulk sulphide flotation concentrate (after cleaner stages) from the Type A material (Au <1.5 g/t, Zn <1.5%). This will contribute a relatively minor amount of the Au sold in Au-Ag concentrate.

These concentrates will be combined to form one concentrate. Approximately 55,000 tonnes of this combined Au-Ag concentrate will be produced annually, likely allowing sufficient concentrate tonnes for bulk shipping and handling. The composition of the combined concentrates is shown in Table 21-8.

Detailed analysis on the two main individual concentrates that comprise the combined Au-Ag concentrate have been undertaken. These results are provided in Table 21-9 (from the Type C material) and Table 21-10 (from the Type B material prior to the addition of the gravity concentrate).

Table 21-8: Combined Gold-Silver concentrate key components average composition

Au (g/t)	Ag (g/t)	Cu (%)	Zn (%)	Pb (%)	S (%)
50-60	250	0.8	3.5	1.1	41

- The likely destination for this concentrate is China, where the smelters do not penalise for the presence Zinc and Lead. Typically, European smelters penalise Lead above 0.3-0.5% and Zinc above 2%. Combined Pb-Zn penalties of US$3.00/dmt for each 1% combined lead zinc are insufficient to rule out European smelters.
- Typical terms for a Chinese buyer will be as follows:
    - Au: Gross payability of 95%
    - Ag: Gross payability of 60%
- TC of US$100/tonne (including RC's).
- This concentrate contains no other penalties for any deleterious elements if sold in China.
- The Au-Ag concentrate is extremely low in Arsenic, with the Arsenic content below the level of detection in all assaying conducted to date. The concentrate may prove attractive to traders and high-arsenic concentrate producers for the purpose of blending with high arsenic concentrates. This could result in the concentrate attracting a premium however this has not been budgeted for in the Scoping Study.
- For the financial analysis in the Scoping Study TC/RC of US$100/t have been assumed.

Table 21-9: Detailed analysis of Type C Au-Ag concentrate (Flotation test F26 and F28)

Au (g/t)	Ag (g/t)	Zn (%)	Cu (%)	Pb (%)	S (%)	F (%)	Cl (%)	Hg(g/t)
117.9	286.4	6.49	0.77	1.30	26.96	<0.01	1.1	7.0
Al (g/t)	**As (g/t)**	**Ba (g/t)**	**Be (g/t)**	**Bi (g/t)**	**Ca (g/t)**	**Cd (g/t)**	**Co (g/t)**	**Cr (g/t)**
6649.4	94.2	62.4	0.2	12.8	62548.7	151.4	15.1	969.7
Fe (%)	**K (g/t)**	**Li (g/t)**	**Mg (g/t)**	**Mn (g/t)**	**Mo (g/t)**	**Ni (g/t)**	**Sb (g/t)**	**Se (g/t)**
30.0	2834.0	<30	3100.1	5613.9	29.6	187.2	88.6	<40
Si (g/t)	**Sn (g/t)**	**Sr (g/t)**	**Ti (g/t)**	**Ti (g/t)**	**U (g/t)**	**V (g/t)**	**Y (g/t)**	**P (g/t)**
n/a	<20	51.0	326.8	<30	n/a	13.8	8.1	<200

Table 21-10: Detailed analysis of Type B Au-Ag rougher concentrate (Flotation test F5)

Au (g/t)	Ag (g/t)	Zn (%)	Cu (%)	Pb (%)	S (%)	F (%)	Cl (%)	Hg(g/t)
28.0	113	**13.9**	**0.3**	**6.2%**	**42.8**	0.022	20	<0.3
Al (g/t)	**As (g/t)**	**Ba (g/t)**	**Be (g/t)**	**Bi (g/t)**	**Ca (g/t)**	**Cd (g/t)**	**Co (g/t)**	**Cr (g/t)**
1,510	<30	7.5	0.28	<20	46,400	1130	<5	65
Fe (%)	**K (g/t)**	**Li (g/t)**	**Mg (g/t)**	**Mn (g/t)**	**Mo (g/t)**	**Ni (g/t)**	**Sb (g/t)**	**Se (g/t)**
13.7	344	<40	2460	7,130	<5	<20	<30	<30
Si (g/t)	**Sn (g/t)**	**Sr (g/t)**	**Ti (g/t)**	**Tl (g/t)**	**U (g/t)**	**V (g/t)**	**Y (g/t)**	**P (g/t)**
n/a	<20	32.6	60.1	<30	<50	<4	1.7	<200

Note – the head grade of this sample was 4.3% Zn 0.7% Pb compared to the expected average grade of 1.1% Zn and 0.1% Pb in the Type B material resulting in a significantly higher Zn and Pb content in this concentrate.

21.3 Marketing Plan

The Company has not yet entered into any contracts for concentrate sales at the time of writing. The concentrate marketing plan has been developed in conjunction with Reach Partners, who were contracted by Challenger to prepare a market study specific to the concentrates that will be produced at Hualilan. Additionally, the Company has had several informal discussions with concentrate off-takers, concentrate marketing groups and concentrate traders. These discussions all supported the Reach Partners data and indicative pricing provide by Reach Partners.

The Company envisages that it will sign definitive off-take agreements for the concentrates after the completion of the Bankable Feasibility Study after running an extensive concentrate offtake tender process.

21.4 Concentrate Packaging and Transport

Concentrate and dore transportation contracts will be negotiated and finalised during the BFS phase of the project. Total freight costs including overland freight by truck, rail freight, port costs, ocean freight, insurance, and assay fees are estimated to be US$150/t.

For the purposes of this Scoping Study it is assumed that the Zn-Au-Ag and Pb-Au-Ag concentrates will be loaded in bags and placed in shipping containers at site for transport. The combined Au-Ag concentrate is likely to be produced in sufficient quantities to allow for bulk shipping and handling, however a decision on bulk shipping will be made during the completion of the planned PFS with transportation via shipping containers assumed in the current study.

The shipping containers and bulk concentrate will be trucked on 20 feet shipping containers 113 kilometres from the project along National Highway 149 to the road-to-rail intermodal facility at Albardon, to the north of San Juan City. The haul is along a double lane sealed highway which can accommodate B-double trucks. The road haul roundtrip will be approximately 5 hours and at the estimated peak rate of production of 70,000 tpa of concentrate a relatively small fleet of 4 haul trucks will be owned and operated by the third-party logistics service provider. Contracts for the haul from Hualilan to San Juan City will be negotiated and finalised during the BFS. Indicative prices of US$9-US$11 per tonne inclusive of loading onto rail have been provided by potential contractors with US$11/t used for budget purposes. These prices were provided by a concentrate logistics study undertaken by Steinweg and local supplier quotes.

If bulk transport of the Au-Ag concentrate is used, an intermodal facility with a 1,000 tonne capacity concentrate storage shed, owned and operated by a third-party logistics service provider, adjacent to the Albardon Station will be used. On arrival each haul truck will be inspected, weighed and will unload the concentrate to the shed floor. On exiting the storage shed, the empty haul truck will be weighed, washed down and inspected prior to its release for the return trip to the project. The shed will be kept under negative pressure to limit the loss of dust to the outside environment. To load the concentrate onto the train, a front-end loader will remove the wagon covers. The front-end loader will load the concentrate directly into wagons in the storage shed.

The concentrate will be exported through Terminal Puerto Rosario. Rosario Port is approximately 1,120 km by rail from Albardon. The existing rail network has sufficient capacity to take the container traffic at the estimated peak rate of production, and Rosario Port has both container and bulk handling facilities. Indicative pricing of US$30-US$35 per tonne via 20 feet shipping containers has been provided to the Company by the rail operator inclusive of container loading and unloading, with US$35/t used for budget purposes.

21.5 Port Storage and Concentrate Shipping

Puerto Rosario is a major goods shipping centre in Argentina servicing the Santa Fe area which produces much of Argentina’s grain exports. The port can accommodate Panamax sized vessels and has existing capacity, including all existing loading facilities that will be required for concentrate exports either via container or in bulk, given the development of additional grain terminals up and down river.

Port Storage and concentrate shipping contracts will be negotiated and finalised during the BFS phase of the project. It has been assumed that secure storage facilities will be leased at Puerto Rosario.

If bulk transport of the Au-Ag concentrate is used the wagons will be unloaded in a 3,000 tonne capacity concentrate storage shed which will be established. The storage shed will be kept under negative pressure to limit the loss of dust with escaped dust collected and placed on the stockpile. Terminal Puerto Rosario uses rotating-container technology to load concentrates and other bulk minerals at its general cargo berth.

The Company engaged Steinweg to undertake a logistics study for the transport of concentrate from site to market. Additionally, the Company's Import/Export agents based in Buenos Aries provided a series of third party quotes for concentrate transport from site to various international ports. Sea freight cost of around US$75/wmt have been used in this study and are regarded as a reasonable assumption for financial modelling purposes. Port costs including concentrate weighing, assaying, and insurance will be in the range of US$20-30/t with US$29/t used for budgeting.

21.6 Payment Process

Indicative payment terms have been provided as part of the concentrate marketing study undertaken by the Company. These indicative terms are summarised below.

Zn-Au-Ag Concentrate

- Delivery: CIF Main ports of Korea, China, Japan.
- Payment: Basis CIF delivery, interest applicable on earlier payments at approximately 3M SOFR +3%. Up to 90% could be paid against Bill of Lading
- Quality Control: Weighing, Sampling and Moisture Determination would be conducted at port of discharge.

Pb-Au-Ag concentrate

- Delivery: CIF Main ports of Korea, China, Japan.
- Payment: Basis CIF delivery, interest applicable on earlier payments at approximately 3M SOFR +3%. 90% could be paid against Bill of Lading
- Quality Control: Weighing, Sampling and Moisture Determination would be conducted at port of discharge.

Au-Ag Concentrate

- Delivery: FOB port of load.
- Payment: Basis CIF delivery, interest applicable on earlier payments at approximately 3M SOFR +3%. Up to 90% could be paid against Bill of Lading
- Quality Control: Weighing, Sampling and Moisture Determination would be conducted at port of discharge.

22 CAPITAL COST

22.1 Capital Cost Estimates

22.1.1 Basis of Estimate

The capital cost estimate was prepared by Mining Plus and various independent external consultants retained by Challenger for capital cost estimates.

There was limited use of benchmarking with costs generally sourced from vendor quotes/indicative prices or detailed first principle cost analysis using vendor quotes based on the preliminary project design. Where benchmarking was used to provide any capital costs the primary source was the Mining Plus internal cost database augmented by Challenger's consultants databases. Where benchmarking has been used to provide capital cost estimates this has been specifically stated in this chapter.

The cost estimate is expressed in Q3 2023 US$ and used the USD/ARS exchange rate at the time the quotation was provided (average 200 ARS) for any in-country costs provided in ARS. In practice in Argentina most cost quotes are generally provided in USD and converted into ARS based on the prevailing USD/ARS rate. The costs do not include allowances for escalation or exchange rate fluctuations. All costs are exclusive of the Argentinian value added tax (VAT) which is applied separately in the financial model used for economic evaluation.

The capital cost estimate for this scoping study has a target accuracy range of ±15% where costs have been sourced from vendor quotes or first principles analysis. The costs developed by benchmarking have a target accuracy of ±35%, unless stated otherwise. The following areas were included in the estimate:

1. Open Pit Mine (open pit mine development, equipment fleet, pre-stripping/pioneering and supporting infrastructure and services)
2. Underground Mine (underground development, equipment fleet, paste backfill plant and supporting infrastructure and services)
3. Process plant (gold-silver, zinc-gold-silver, and lead–gold-silver concentrates), conventional 1,000,000 tpa concentrator and Flotation Tails leach circuit with supporting plant infrastructure and services)
4. TSF
5. On-site infrastructure (earthworks, sitework, roads, water treatment and distribution, camp and other general facilities)
6. Off-site infrastructure
7. Owners Costs including EPCM, spares, first fills, transport costs and import costs.
8. Indirect costs

9. Other Pre-production Costs (other operating costs prior to commercial production/ processing)
10. Contingency (applied at +15%)

Total capital costs are US$133.7 million not including US$18.4 million of capitalised mining costs. Total pre-development capital costs of US$152.1 million are summarised in Table 22-1.

Sustaining capital costs are related to the following:

- UG mine equipment acquired after processing commences.
- Underground Development (inclined and vertical)
- UG Infrastructure construction
- Process Plant maintenance
- Surface and UG Infrastructure maintenance (maintenance costs for ventilation, paste plant and underground mining equipment have been included as operational mining costs)
- TSF ongoing costs.

Table 22-1: Summary Capital Cost Estimate

Description	Pre-production Capital Costs US$ M	Sustaining Capital Cost US$ M	Total Capital Cost US$ M
1. Open Pit Development (inc. Truck Shop, Wash Bay, Tyre Bay)	5.8		5.8
2. Underground Development (inc. paste plant)	21.8	45.0	66.8
3. Process Plant	59.0	8.9	67.9
4. TSF	5.4	3.2	8.6
5. On-site Infrastructure	8.7	1.5	10.2
6. Off-site infrastructure	0.0	0.0	0.0
7. Owners Costs	15.6		15.6
8. Indirect Costs	2.7		2.7
9. Contingency	14.7	0.5	15.2
Total Capital Expenditure	**133.7**	**59.0**	192.7
10. Other Pre-production Costs[3]	18.4		18.4
Total	**152.1**	**59.0**	211.1

1. All figures are rounded to reflect the relative accuracy of the estimate.
2. Totals may not sum due to rounding as required by reporting guidelines.
3. Pre-production costs are operating costs that occur prior to the mill operating.

22.2 Open Pit Mine Capital Costs

This item accounts for the capital costs associated with the open pit mine, haul roads, and support mine infrastructure and services. The Open Pit Mine Capital cost estimate is summarised in Table 22-2.

Table 22-2: Open Pit Mine Capital Cost Estimate

1 OPEN PIT MINE CAPITAL COSTS		US$
111 Open Pit Mine Development		
	1111 Pre-Stripping	876,000
	1112 Pioneering	876,000
112 Open Pit Mining Equipment		
	1123 Contractor Operated	0
113 Open Pit Mine Infrastructure		
	1131 Site Preparation and Haul Roads	450,000
	1132 Dewatering	0
	1133 Truck Shop	2,500,000
	1134 Wash Bay	550,000
	1135 Tire Bay	585,000
Sub Total Mine Capital Costs		**5,837,000**
Contingency		**612,750**
TOTAL OPEN PIT MINE		**6,449,750**

It is assumed that all open pit mobile mine development equipment will be supplied by the contractor, thus, the capital cost estimate does not consider the purchase of mining equipment. Auxiliary mine fleet will be leased with monthly rental costs sourced from multiple equipment suppliers in San Juan were used for this study. The combined open pit and underground Auxiliary Fleet is detailed in Section 15.7.

Mine capital costs include a 0.4 Mt pre-production stripping of unconsolidated material that does not require drill and blast and $US1.2 million open pit pioneering. No contingency was applied to these costs. Pre-stripping and pioneering will begin six months prior to process plant and other construction to ensure there is sufficient waste material for roads, flood control, ROM Pad, and site earthworks prior to the commencement of general construction.

The site preparation and haul roads costs were based on earthworks and quantities estimated from the preliminary general site layout with unit costs developed from a first principles cost model using supplier quotes. The cost of the Truck-Shop, Wash-Bay and Tyre Bay was benchmarked from similar operations in South America with costs sourced from the Mining Plus internal cost database. A contingency of +15% was applied to these costs.

22.3 Underground Mine Capital Costs

The Underground Mine Capital costs are summarised in Table 22-3.

Table 22-3: Underground Mine Capital Costs

2 UNDERGROUND MINE CAPITAL COSTS		US$
223 Pre-Production Underground Mine Development		
	2231 Portal Ground Support & Catch Fence	355,555
	2232 Portal Electrical Infrastructure Establishment	533,333
	2233 Decline Development (15% contingency included)	8,214,490
	2234 Vertical Development (15% contingency included)	3,031,929
222 Underground Mining Equipment		
	2223 100% Lease finance available/rental	-
223 Mine Infrastructure		
	2231 Underground facilities including change house and explosives magazine, refuges	310,000
	2232 Ventilation - Cooling	2,916,667
	2233 Dewatering/Pumping	603,000
	2234 Paste Plant	5,166,667
	2235 Raw water establishment	66,667
	2236 Compressor for Underground	200,000
	2237 PPE	75,000
	2238 Cap-lamps & Chargers	66,667
	2239 Self Rescuers	56,667
	2240 On-Site Power Supply & Transmission (underground)	166,667
Sub Total Mine Capital Costs		**21,763,309**
Contingency		**1,577,533**
TOTAL UNDERGROUND MINE		**23,340,842**

22.3.1 Pre-Production Development

The estimated underground mine development costs were based on development quantities obtained from the preliminary conceptual mine design and schedule which is outlined in Table 22-4. All inclined (Ramp) and vertical development is allocated to capital with the remaining lateral development and boxhole rises allocated to operating costs.

Development rates were derived via a ground up first principles cost model developed by Challenger's Underground Mining Consultants. This model used input costs provided by supplier quotations in Argentina with typical industry productivity provided by equipment vendors discounted to reflect local operating productivity. The first principles development cost model assumes expatriate labour is used for all Jumbo, Loader, and Service/Maintenance operators for the initial three years of operation until training of local operators is complete. This cost model has an internal 20% contingency built into the model.

Table 22-4: Underground Mine Development Physicals

	Year-0	Year-1	Year-2	Year-3	Year-4	Year-5	Year-6	Year-7	Total
LATERAL DEVELOPMENT									
Ramp	2,905	2,674	1,958	1,519	1,114	501	-	-	**10,671**
Stockpile	285	362	298	272	155	129	-	-	**1,501**
Vent Drive	748	686	636	819	228	206	-	-	**3,322**
Escapeway Access	779	422	471	381	307	124	-	-	**2,485**
Xcut- Waste	691	914	805	808	368	278	-	-	**3,864**
Ore Drive-Waste	492	2,840	3,373	3,759	1,097	919	-	-	**12,479**
XCut-Ore	12	36	38	22	7	48	-	-	**161**
Ore Drive-Ore	442	2,033	2,862	2,960	1,745	719	-	-	**10,761**
Slot Drives	5	370	553	437	640	525	537	138	**3,205**
Backfill Development	8	592	896	688	1,024	840	864	208	**5,120**
Total Lateral Development	**6,367**	**10,928**	**11,889**	**11,663**	**6,685**	**4,290**	**1,401**	**346**	**53,569**
VERTICAL DEVELOPMENT									
Escapeway Raise	683	621	558	432	245	213	-	-	**2,753**
Raise Ventilation	616	631	518	418	235	198	-	-	**2,615**
Total Vertical Development	**1,299**	**1,252**	**1,076**	**850**	**480**	**411**	**-**	**-**	**5,368**
Boxhole Rises	11	1,469	1,976	1,859	2,553	2,055	2,042	560	**12,526**
Total Development	**7,677**	**13,649**	**14,941**	**14,373**	**9,717**	**6,756**	**3,443**	**906**	**71,463**

22.3.2 Mining Equipment Fleet

It is assumed that the underground mining operation is owner operated. An initial estimate on sizing of the mining equipment fleet was done using the preliminary underground mine physicals developed for the scoping study. After this indicative underground mining fleet was developed Challenger entered into discussions with potential equipment suppliers.

During discussions with equipment suppliers, indicative terms for 100% lease finance of the specialised underground equipment fleet were provided. Based on these discussions it has been assumed that the underground Development, Production, Load and Haul fleet, and Integrated Tool Carriers (IT's) will be purchased via 100% lease finance.

Additionally, discussions were held regarding potential full-service maintenance solutions provided by the equipment vendor. At this stage it has been assumed that maintenance will be the responsibility of Challenger. Discussions are ongoing and Challenger would expect the next step to involve entering into an Equipment Supply Partnership Agreement with its preferred underground mining equipment supplier.

Challenger sourced long term rental costs for the Auxiliary Fleet from several rental operators in San Juan. There is a deep mining equipment rental market in the province of San Juan. For study purposes it is assumed that the Auxiliary Fleet will be sourced via long term equipment rental. In San Juan, a standard long term rental agreement has the advantage of including full equipment maintenance and, for heavy machinery, the provision of an operator. Additionally, the Company sourced rental quotes for items from the underground mining fleet, the majority of which is available via rental, which provides additional flexibility.

The underground Mining and Auxiliary Fleet is shown in Table 22-5. The table includes monthly lease finance and/or rental costs sourced by equipment suppliers. This table also includes the outright purchase price of this equipment for comparison purposes. The availability of 100% lease finance options and long-term rental removes US$19.6 million from the Underground Mine Development Capital cost estimates for the project with a trade-off in higher operating expenses.

Table 22-5: Underground Mining and Auxiliary Fleet

Unit	Qty Pre Production	USD Unit Cost	Unit Life (months)	Monthly Lease Finance or Rental Cost	Comment	Total Capex USD if purchased
Development						
Jumbo	3	1,562,525	60	**$33,738**	lease finance	$4,687,575
Charge Up (ANFO Loader)	2	565,715	48	**$14,549**	lease finance	$1,131,429
Production Rigs						
Production Drill Rig	1	1,478,166	60	**$31,916**	lease finance	$1,478,166
Load and Haul Fleet						
17T Loader	1	1,087,972	48	**$27,981**	lease finance	$1,087,972
17T Loader (Tele-remote)	**1**	1,234,639	48	**$31,753**	lease finance	$1,234,639
UG Truck 55T	2	1,305,893	48	**$33,585**	lease finance	$2,611,786
Charge Up (ANFO Loader)	1	565,715	48	**$14,549**	lease finance	$565,715
Auxiliary Fleet						
ITC (Service Crew)	3	372,680	48	**$9,585**	lease finance	$1,118,040
ITC (Paste Crew)	1	372,680	48	**$9,585**	lease finance	$372,680

Unit	Qty Pre Production	USD Unit Cost	Unit Life (months)	Monthly Lease Finance or Rental Cost	Comment	Total Capex USD if purchased
Services Truck	1	565,715	48	**$15,000**	rental	$565,715
Stores Flat Bed Truck	1	308,000	48	**$20,000**	rental	$308,000
Grader	1	352,000	48	**$14,000**	rental	$352,000
Shotcrete Sprayer	1	609,715	48	**$16,200**	rental	$609,715
Agitator Truck	1	622,285	48	**$10,800**	rental	$622,285
Stores Flat Bed Truck	1	308,000	48	**$20,000**	rental	$308,000
Grader	1	352,000	48	**$14,000**	rental	$352,000
Watercart	1	748,587	48	**$15,005**	rental	$748,587
Light Vehicle (4 x 4)	20	73,333	48	**$3,144**	rental	$1,466,667
Total		**Total Capital Cost if Equipment purchased outright**				**$19,620,969**

22.3.3 Underground Mine Infrastructure

The capital cost of the underground facilities, including change house and underground explosives magazine, was estimated by Challenger's underground mining consultants using their internal cost database. Mine ventilation and electrical capital cost was estimated based on a preliminary sizing of the ventilation system composed of drifts, raises and ventilation. Dewatering and pumping costs were based on preliminary sizing based on budgeted water production during mining with a contingency of 25% built into the pumping model. The cost of the Paste Plant was provided by Challenger's underground mining consultants using their internal cost database. It has been assumed that the open pit Truck-shop, Tyre Bay and Wash Bay will be utilised for both Open Pit and Underground operations.

22.3.4 Underground Sustaining Capital Costs

The underground mining sustaining capital costs are summarised in Table 22-1. Costs such as ventilation and pumping, underground mining and auxiliary fleet maintenance after the commencement of production have been included in mine operating costs for this study.

It is assumed that the mining fleet will be replaced every 4 years with the exception of jumbos and production drill rigs which have a 5-year life. As the replacement mining equipment fleet is contemplated via lease finance, only the cost transport of the replacement mining fleet from Buenos Aires Port to site and import duty (at 1%) has been applied to sustaining capital.

All inclined (decline/ramps) and vertical development is allocated to sustaining capital after the commencement of commercial production; with the remaining horizontal development allocated to operating costs. The installation of 4-person (US$175,000) and 24 person (US$225,000) underground refuge chambers, as well as stench gas warning systems

(US$75,000) are allocated to sustaining capital, as these chambers will be installed after the commencement of processing and commercial production.

22.4 Processing Capital Costs

The Treatment Plant capital cost estimate for Gravity plus CIL process option was developed by Challenger's external metallurgical consultants on the same basis as the capital cost estimate for the Gravity + Sequential Flotation + FTL process option. This involves the existing comminution and gravity circuit with no flotation circuit with the gravity tails direct to CIL.

The process capital cost estimate is summarised in Table 22-6. Contingency of 15% has been applied to the process plant capital cost. A complete listing of process plant capital costs are provided in

Table 22-7 for the Gravity, Flotation and FTL option and Table 22-8 for the Gravity and CIL option.

Table 22-6: Treatment Plant Capital Costs

3 PROCESSING PLANT	US$
331 Construction Overheads	3,660,868
332 Bulk Earthworks	504,573
333 Primary Crushing	5,105,068
334 Milling & Classification	14,530,402
335 Bulk Flotation and Regrind	-
336 Leaching & Adsorption	7,757,591
337 TSF & Decant Return	1,641,895
338 Concentrate Dewatering and Handling	-
339 Metal Recovery & Refining	4,639,025
341 Reagents	2,047,283
342 Services	3,577,458
Sub Total Processing Plant	**43,464,162**
Contingency 15%	**6,519,624**
TOTAL PROCESSING PLANT	**49,983,786**

Table 22-7: Full Listing of Treatment Plant Capital Cost Estimate (Gravity, Flotation and FTL)

3 PROCESSING PLANT		US$
331 Construction Overheads		
	3311 Mechanicals Total	835,900
	3312 Platework Total	597,072
	3313 Structural Total	597,072
	3314 Concrete Total	466,480
	3315 E&I Total	806,101

3 PROCESSING PLANT		US$
	3316 Piping Total	358,243
332 Bulk Earthworks		
	3323 Earthworks Total	682,378
333 Primary Crushing		
	3331 Mechanicals Total	2,338,545
	3332 Platework Total	809,478
	3333 Structural Total	972,453
	3334 Concrete Total	350,655
	3335 E&I Total	337,933
	3336 Piping Total	296,004
334 Milling & Classification		
	3341 Mechanicals Total	10,083,116
	3342 Platework Total	327,394
	3343 Structural Total	1,534,944
	3344 Concrete Total	546,551
	3345 E&I Total	725,766
	3346 Piping Total	1,312,631
335 Bulk Flotation and Regrind		
	3351 Mechanicals Total	10,540,724
	3352 Platework Total	155,424
	3353 Structural Total	456,694
	3354 Concrete Total	388,942
	3356 E&I Total	857,270
	3357 Piping Total	1,408,116
336 Leaching & Adsorption		
	3361 Mechanicals Total	2,718,511
	3362 Platework Total	2,951,362
	3363 Structural Total	1,046,488
	3364 Concrete Total	247,620
	3365 E&I Total	453,983
	3366 Piping Total	339,627
337 TSF & Decant Return		
	3371 Mechanicals Total	131,261
	3372 E&I Total	18,789
	3378 Piping Total	1,491,845
338 Concentrate Dewatering and Handling		
	3381 Mechanicals Total	1,855,093
	3382 Platework Total	255,614
	3383 Structural Total	590,488
	3384 Concrete Total	99,646
	3385 E&I Total	305,613

3 PROCESSING PLANT		US$
	3386 Piping Total	233,644
339 Metal Recovery & Refining		
	3391 Mechanicals Total	1,030,607
	3392 Platework Total	142,008
	3393 Structural Total	328,049
	3394 Concrete Total	55,359
	3395 E&I Total	169,785
	3396 Piping Total	129,802
341 Reagents		
	3411 Mechanicals Total	1,407,983
	3412 Platework Total	499,710
	3413 Structural Total	498,215
	3414 Concrete Total	253,866
	3415 E&I Total	259,566
	3416 Piping Total	151,586
342 Services		
	3423 Mechanicals Total	858,187
	3422 Platework Total	21,300
	3423 Structural Total	95,766
	3424 Concrete Total	175,295
	3425 E&I Total	2,323,400
	3426 Piping Total	103,510
Sub Total Processing Plant		**59,029,461**
Contingency		8,854,419
TOTAL PROCESSING PLANT		**67,883,880**

Table 22-8: Full Listing of Treatment Plant Capital Cost Estimate (Gravity and CIL)

3 PROCESSING PLANT		US$
331 Construction Overheads		
	3311 Mechanicals Total	835,900
	3312 Platework Total	597,072
	3313 Structural Total	597,072
	3314 Concrete Total	466,480
	3315 E&I Total	806,101
	3316 Piping Total	358,243
332 Bulk Earthworks		
	3323 Earthworks Total	504,573
333 Primary Crushing		
	3331 Mechanicals Total	2,338,545
	3332 Platework Total	809,478

3 PROCESSING PLANT		US$
	3333 Structural Total	972,453
	3334 Concrete Total	350,655
	3335 E&I Total	337,933
	3336 Piping Total	296,004
334 Milling & Classification		
	3341 Mechanicals Total	10,083,116
	3342 Platework Total	327,394
	3343 Structural Total	1,534,944
	3344 Concrete Total	546,551
	3345 E&I Total	725,766
	3346 Piping Total	1,312,631
335 Bulk Flotation and Regrind		
	3351 Mechanicals Total	
	3352 Platework Total	
	3353 Structural Total	
	3354 Concrete Total	
	3356 E&I Total	
	3357 Piping Total	
336 Leaching & Adsorption		
	3361 Mechanicals Total	2,718,511
	3362 Platework Total	2,951,362
	3363 Structural Total	1,046,488
	3364 Concrete Total	247,620
	3365 E&I Total	453,983
	3366 Piping Total	339,627
337 TSF & Decant Return		
	3371 Mechanicals Total	131,261
	3372 E&I Total	18,789
	3378 Piping Total	1,491,845
338 Concentrate Dewatering and Handling		
	3381 Mechanicals Total	
	3382 Platework Total	
	3383 Structural Total	
	3384 Concrete Total	
	3385 E&I Total	
	3386 Piping Total	
339 Metal Recovery & Refining		
	3391 Mechanicals Total	2,576,518
	3392 Platework Total	355,020
	3393 Structural Total	820,123
	3394 Concrete Total	138,397

3 PROCESSING PLANT		US$
	3395 E&I Total	424,463
	3396 Piping Total	324,505
341 Reagents		
	3411 Mechanicals Total	938,655
	3412 Platework Total	333,140
	3413 Structural Total	332,143
	3414 Concrete Total	169,244
	3415 E&I Total	173,044
	3416 Piping Total	101,057
342 Services		
	3423 Mechanicals Total	858,187
	3422 Platework Total	21,300
	3423 Structural Total	95,766
	3424 Concrete Total	175,295
	3425 E&I Total	2,323,400
	3426 Piping Total	103,510
Sub Total Processing Plant		**43,464,162**
Contingency		**6,519,624**
TOTAL PROCESSING PLANT		**49,983,786**

22.5 Tailing Storage Facility (TSF)

Costs for the Integrated Waste Landform (IWL) TSF were estimated based on a complete listing of all materials required for construction provided by LMGS, the external consultants that designed the TSF. Unit costs were obtained from a combination of supplier quotes and a first principles cost model for the cost of earthworks and supervision.

The TSF design contemplates double lining of the TSF with 1.5 mm HDPE Liner and installation of Polytex HDPE 7000 Flownet over the designated double lined TSF footprint, leak detection layer, and upstream batters. It is assumed that this HDPE liner and Flownet would be imported with import and shipping costs provided by Challenger's external shipping consultants based in Buenos Aires. It was assumed that all earthworks material (approximately 4.9 Mt) required for the TSF will be provided from the Open Pit Waste.

The TSF will be completed in an initial lift prior to commencement of processing that will provide sufficient storage capacity for two years with a final lift completed by the end of Year Two. The initial lift is assigned to pre-production capex with the final lift assigned to sustaining capex.

Capital cost estimates for Lift 1 and the Final Lift are provided in Table 22-9 and Table 22-10.

Table 22-9: TSF Initial Lift Capital Cost Estimate

4 TSF	Stage 1 Initial Lift (Capital)	US$
441 Tailings Disposal		
	4411 TSF earthworks/supervision	**1,496,700**
	4412 HDPE Liner	1,639,213
	4413 Tailings pipeline/pump	81,875
442 Reclaim Water		
	4420 Reclaim Water	2,194,870
Sub Total TSF Capital Costs		**5,412,658**
Contingency		**811,899**
TOTAL TSF CAPITAL COSTS		**6,224,556**

Table 22-10: TSF Final Lift Capital Cost (Sustaining) Estimate

4 TSF	Stage 2 Final Lift (Sustaining Capex)	US$
441 Tailings Disposal		
	4411 TSF earthworks/supervision	**2,991,240**
	4412 HDPE Liner	158,597
	4413 Tailings pipeline/pump	
442 Reclaim Water		
	4420 Reclaim Water	
Sub Total TSF Capital Costs		**3,149,837**
Contingency		**472,476**
TOTAL TSF CAPITAL COSTS		**3,622,312**

22.6 On Site Infrastructure Capital Costs

Infrastructure-related capital costs are detailed in Table 22-11 and include:

- Water sourcing, treatment and onsite distribution including fire water distribution.
- Internal access roads, bulk excavation and siteworks fencing and lighting towers.
- On-site camp and other general facilities including warehouse, explosives magazine and security infrastructure.
- Fresh water supply
- Effluent and Waste Management facilities
- Communications infrastructure
- Software.

Table 22-11: On-site Infrastructure Capital Cost Estimate

5 ON SITE INFRASTRUCTURE		US$
551 Site Preparation		
	5511 Diversions & Drainage	65,000
	5512 Fencing	610,406
	5513 Excavation Bulk and siteworks	876,000
	5514 Lighting Towers	90,000
552 On-Site Roads		
	5520 On-Site Roads	500,000
	5523 Culverts	50,000
553 On-Site Non-Process Facilities		
	5531 Permanent Accommodation	2,000,000
	5533 General Offices Included above)	0
	5534 Assay Laboratory	0
	5536 Security Infrastructure	200,000
	5537 Warehouse	600,000
554 On-Site Bulk Storage		
	5541 Water - Potable	80,000
	5542 Fuel	166,667
	5543 Explosives	300,000
555 On-Site Services/ Utilities		
	5551 On-Site Potable Water	170,000
	5552 On-Site Process Water/bore field	2,000,000
	5553 On-Site Fire Water	500,000
	5554 On-Site Effluent	75,000
	5555 Recycling / Sorting Facility and domestic waste	100,000
	5556 Survey Drone	0
	5557 Survey Total Stations and Other Survey Equipment	175,000
556 On-Site Communications		
	5561 Communications & Networks	103,000
557 On-Site Ming Planning and other software		
	5571 Survey/Mining/Geology/Geotech	250,000
Sub Total On-Site Infrastructure		**8,711,073**
Contingency		**1,006,661**
TOTAL ON SITE INFRASTRUCTURE		**9,717,734**

The site preparation and haul roads costs are based on earthworks quantities estimated from the general site layout. The site layout includes 26 km of site roads (including Haul Roads which have been accounted for in Open Pit Mine Capital Costs costs). A bottom-up cost analysis was completed to derive the cost of road construction for budget purposes with road aggregate provided from the limestone portion of mining waste which will be crushed on site.

The design includes 19.5 km of fencing with quantities estimated from the general site layout. Costs applied to the 19.5km of fencing were derived from supplier quotes. The capital cost estimate accounts for site preparation/earthworks based on massive earthworks quantities estimated from the preliminary general site layout and sketched sections and unit costs sourced from the first principles analysis based on supplier quotes. Flood diversion will require 64,000 t of material which will be sourced from the open pit pre-strip.

The cost for the permanent accommodation (120 bed camp including, kitchen, recreational facilities, offices/conference rooms and sporting facilities) is based on the results of a tender process completed for the construction of this facility. The existing 100 bed temporary camp will be used for overflow accommodation during open pit mining. No contingency has been applied to this cost.

Costs for security infrastructure, the warehouse and on-site bulk storage have been benchmarked using the Mining Plus internal cost database with costs augmented by Challenger's external consultants.

Costs for on-site services including the completion of a bore field for process water, on-site effluent treatment and drinking water have been based on reputable local supplier quotes. The company has existing survey drones with an allowance of US$175,000 made for additional survey equipment.

Challenger has entered into a Memorandum of Understanding (MOU) with YPF Luz ("YPF") for the supply of renewable power to the Hualilan Gold Project. The MOU contemplates that the PPA can, at Challenger's election, include all connection capital expenditure associated with the provision of power to the project thus removing the need for Challenger to fund the upfront capital component associated with the provision of power to the project. Accordingly, it has been assumed that capital costs associated with provision of power to site will be included in the PPA.

Challenger has received indicative quotes for connection to the grid via duel 33 kV Medium Voltage lines. This estimate of US$4.5 M includes line costs of US$3.5 M with an additional US$1 M for other infrastructure. Additionally, an amount of US$0.6 M has been included in the underground mine capital cost estimates for electrical sub stations and other infrastructure with an additional US$0.8 M allowed for the installation of underground electrical infrastructure in underground mining costs.

22.7 Off Site Infrastructure Capital Costs

No off-site infrastructure is required for roads given the proximity of the National highway 400 m from site. It has been assumed that concentrate will be trucked in containers hence no off-site concentrate handling or port facilities are required with small storage facilities for containers to be rented. The provision of all utilities and services has been allowed for in On-Site Infrastructure.

22.8 Owner's Costs

Owner's capital cost estimates are provided in Table 22-12.

Estimates for project EPCM costs were provided by Challenger's Metallurgical Consultants based on benchmarking of similar operations with costs from their internal proprietary costs database. An allowance of US$3.15 million has been made for spares and first fills including Process Plant first fills.

An allowance of US$170,000 has been made for the cost of transport of the underground mining fleet that will be lease financed for its import into and delivery to site from Buenos Aires Port. An amount of US$2.8 million has been budgeted for the cost of the shipping and importation of the Processing Plant to Buenos Aires Port and transport to site. The shipping costs are based on the quantities of containers required to transport the process plant, with rates for shipping from China to Buenos Aires and subsequent transport to site provided by supplier quotes sourced by Challenger's shipping and import consultants based in Buenos Aires.

Import duty of 1% is payable on items imported for mining. Items imported for mining are exempt from other import duties which include VAT withholding (20%); income tax down-payment (6%) and gross income withholding (2.5%). Import duty on the US$55 M cost of the processing plant, excluding local concrete and earthworks, and the US$19.6 M capital cost of the underground mining fleet purchase via lease finance has been budgeted.

The cost of the owner's project team during project development and execution has been included in section 22.10.

Table 22-12: Owner's Capital Cost Estimate

7 OWNER'S COSTS		US$
771 Project Cost		
	7711 Project Cost and EPCM	8,787,052
772 Spares & Inventory		
	7723 Spares & Inventory	2,000,000
	7722 First Fills (excl process plant)	225,000
	7723 Process Plant First Fills & Consumables	900,000
773 Transport Costs		
	7731 Transport Costs Plant to site	2,782,500
	7732 Transport Costs Capital equipment to site	170,000
	7733 Import Duty (plant and capital equipment)	753,836
Sub Total Capex on site infrastructure		15,618,388

7 OWNER'S COSTS		US$
Contingency		1,431,133
TOTAL OWNER'S COSTS		**17,049,523**

22.9 Indirect Costs

Indirect capital Cost Estimates are shown in Table 22-13.

Table 22-13: Indirect Capital Cost Estimate

8 INDIRECT COSTS		US$
881 Temporary Facilities		
	8811 Temporary Facilities	76,200
882 Temporary Services		
	8823 Temporary Services	
883 Temporary Camp		
	8831 Temporary Camp and Catering	2,400,000
884 Commissioning		
	8841 Training/Induction + Management Plans	181,333
Sub Total Capex on site infrastructure		**2,657,533**
Contingency		**398,630**
TOTAL INDIRECT COSTS		**3,056,163**

22.10 Contingency

The contingency was established applying the following percentage factors associated with a Scoping Study level estimate.

- +15% on Open Pit and Underground Mine capital costs
- +15% on the Process Plant
- +15% on the TSF and On-Site infrastructure
- +15% on the Direct and Indirect and Owner's costs
- +20% contingency was embedded in the Underground Development rates
- No contingency has been applied to capitalised operating costs, capital cost items sourced from a tender process, or government duties and taxes.

22.11 Other Pre-Production Capital Costs

Other Pre-Production Capital Costs are operating costs that occur prior to the mill operating. These capital cost estimates in are shown in Table 22-14.

Pre-production costs include mining of 2.4 Mt of material (both waste and material to be processed) in the 12 months prior to the commencement of processing. Open pit contractor mobilisation costs are US$300,000.

Some 3,462 m of lateral development, which produces 24,870 t of development ore, and 802 t of stope production is capitalised at a cost of US$9.8 million in the 12 months prior to the commencement of processing/commercial production. The lateral development and stoping estimates are based on the lateral development and stoping quantities obtained from a preliminary conceptual mine design and schedule with development rates derived via the ground up first principles cost model developed by Challenger's Underground Mining Consultants.

A cost of US$1.2 million has been included for the mine owners and processing teams prior to commercial production. No contingency has been applied to these capitalised operating costs.

Table 22-14: Other Pre-Production Capital Cost Estimates

9 PRE-PRODUCTION COSTS		US$
	9911 Open Pit Mining (ore + waste)	6,360,000
	9912 Open Pit Contractor Mobilisation	300,000
	9913 Lateral Development and stoping (15% contingency included)	9,817,285
	9914 Mine owner's team (OP/UG/Processing/GA)	1,920,000
Sub Total Capex on site infrastructure		**18,397,285**
Contingency		**0**
TOTAL INDIRECT COSTS		**18,397,285**

23 OPERATING COSTS

23.1 Basis of Estimate

The operating cost estimate is based on contractor operated truck and shovel open pit mining, owner operated underground mining via longitudinal sub-level open stoping (SLOS) with paste backfill, conventional flotation with Float Tail Leach and deposition in a Tailing Storage Facility.

Unless specifically stated in this chapter, operating cost estimates have been derived from first principles costs analysis prepared by external consultants, rather than by benchmarking. These cost estimates include local labour rates derived from San Juan industry standards, costs sourced by vendor/ supplier quotations both in Argentina and externally, and productivity rates that reflect the local workforce and conditions.

Unless otherwise stated this estimate has an expected accuracy range of ±15% and is expressed in Q3-2023 US$. The estimate includes the underground mining, processing and G&A operating costs. It excludes escalation, currency fluctuations, off-site costs, interest charges and taxes. No contingency has been included in the operating costs.

The operating estimate is expressed in Q3 US$ and used USD/ARS exchange rate at the time the quotation was provided for any in country costs provided in ARS. In practice, in Argentina, most quotes are generally provided in USD and converted into ARS based on the prevailing USD/ARS. This includes diesel, equipment hire for both general and specialised mining equipment, reagents and consumables. The exceptions are Government provided services such as grid power and in-country labour. Generally, the rate of increase in the ARS price tracks the decline in the ARS/USD rate for power and labour, however there is a 1-3 month lag in the repricing of ARS denominated costs. For any ARS denominated input costs such as grid power and labour Challenger has used the 2022H2/2023H1 prices as converting the current local ARS costs into a USD denominated price at the prevailing ARS rate would artificially lower these input costs on a USD basis.

Summary mine operating cost estimates are provided in Table 24-1.

Table 23-1: Summary of Mine Operating Cost Estimates

Unit operating Costs	Unit	Unit Cost
Open pit Mining (ore/waste)	**US$/t moved**	**3.00**
Underground Mining	US$/t mined	34.74
Underground Development	US$/t mined	28.29
Underground Mining Total	**US$/t mined**	**63.03**
Processing (≥ 1.5% Zn) - 7% total ore	**US$/t processed**	**16.31**
Processing (≥ 1.5 g/t Au, <1.5% Zn) - 59% total ore	**US$/t processed**	**12.12**
Processing (<1.5 g/t Au) - 34% total ore	**US$/t processed**	**9.26**

Unit operating Costs	Unit	Unit Cost
G&A	**US$/t processed**	**5.38**

23.2 Open Pit Mining Costs

Open Pit Mining costs were benchmarked against similar sized open pit mining operations in South America. These figures were provided by Mining Plus using data from their internal database, as well as from Challenger's external consultants databases. Benchmark costs include drill, blast, load, haul, auxiliary equipment required to maintain operation and mine owner's costs.

Corrections were applied for unit power and diesel costs at the benchmark operations. Benchmark costs were inflated to Q3 2023 prices using US CPI rates. Most of the benchmark operations are located at altitude, whereas Hualilan is not, however no further adjustments have been made to account for this difference in altitude.

These benchmark rates were then checked against a first principles mining cost model built around an internally constructed labour model, and with fuel and maintenance estimates provided by a number of mining equipment manufacturers, to test for reasonableness.

The open pit mine plan was designed around a constant mining rate of 200,000 t per month, such that the 9.6 Mt total material from the open pit is steadily mined at 2.4 Mt per annum over four years. Initial discussions with mining contractors have been initiated.

For reference, the company notes a $US3.00/t contractor quotation was sourced for the Mining Plus 2022 PEA for the Diablillos Project located in Salta Province, Argentina (Table 23-2).

An approximation of headcount for the open pit, which will have contractor labour for mine operations and maintenance but owner technical services and operations oversight/ supervision, has been included as Table 23-3.

Table 23-2: Open Pit Mining Unit Cost Breakdown for Similar Operation

Component	Unit Cost ($/t mined)	Inclusions
Drill	0.32	Equipment: 1x Epiroc D65 or equivalent Price Incl: Labour, Fuel/ lube, GET, Maintenance, contractor margin.
Blast	0.57	Bulk Explosive, Explosive Accessories, contractor PLTS service incl. equipment + labour, contractor margin Equipment: MPU, LVs, stemming loader
Load	0.45	Equipment: 1x Liebherr R9150 or equivalent

Component	Unit Cost ($/t mined)	Inclusions
		Price Incl: Labour, Fuel/ lube, GET, Maintenance, contractor margin.
Haul	0.57	Equipment: 3x CAT773G Trucks or equivalent Price Incl: Labour, Fuel/ lube, GET, Maintenance, contractor margin.
Auxiliary – Pit + ROM	0.74	Equipment: 1x D8 Dozer, 1x 16M Grader, 1x 988 Loader, 2x 40,000L Water cart, 1x Service Cart or equivalents Price Incl: Labour, Fuel/ lube, GET, Maintenance, contractor margin.
Sub-Total	**2.65**	
Internal Technical + Supervision	0.35	Mine Planning, Survey, Geotechnical, Geology, OP Production Management
Mining Cost Total	**3.00**	

 # Costs include contractor margin

Table 23-3: Open Pit Headcount by Year

Headcount	Year 0	Year 1	Year 2	Year 3	Year 4 - 7
OP MTS HC					
OP Geotech	4	4	4	4	
OP Mine Planning	8	8	8	8	
OP Survey	6	6	6	6	
OP Geology	16	16	16	16	
OP OPERATIONS HC (Contractors)					
OP Auxiliary	20	20	20	20	
OP Blasting	6	6	6	6	
OP Haul	13	13	13	13	
OP Load	8	8	8	8	
OP Production Drilling	6	6	6	6	
OP Supervision	1	1	1	1	
OP MAINTENANCE HC (Contractors)					
Maintenance Supervision	1	1	1	1	
OP Auxiliary Maintenance	6	6	6	6	
OP Haul Maintenance	6	6	6	6	
OP Load + Drill Maintenance	6	6	6	6	

23.3 Underground Mining Operating Costs

Based on resource geometry, the scale of the deposit and grade distribution, longitudinal sub-level open stoping (SLOS) with paste backfill was selected as the mining method. Approximately 50% of tailings will be returned to the underground stope voids as part of the paste fill mix. The mine operating costs were developed by Challenger's underground mining consultants using a detailed, first-principles underground cost model. The costs have been scheduled by period and built up by type for each basic mining activity.

23.3.1 Development Costs

Development unit rate costs were calculated through a first principles cost model, developed by Challenger's underground mining consultants, using a preliminary mine development and production schedule built by Mining Plus.

This model used input costs, which predominantly originated through supplier quotations for Argentina, with typical industry productivity estimates provided by equipment vendors and adjusted to reflect local operating practices. The first principles development cost model assumes expatriate labour is used for jumbo, loader and underground mine operations supervision, as well as supporting the maintenance function, for the initial three years of operation and until training of local operators is complete.

This cost model has an internal, inbuilt 20% contingency. Table 23-4 shows the detail of the estimated unit prices for mine development.

Table 23-4: Underground Development Unit Costs

Development Type	Unit	Unit Cost
Inclined Development (5 m x 5 m)	US$/m	2,828
Horizontal development (5 m x 5 m)	US$/m	2,828
Vertical Development	US$/m	2,333
Slot Rises	US$/m	1,500

23.3.2 Production Costs

A summary of overall underground production operating costs is presented in Table 23-5. The mine operating cost estimate included the costs associated with stope preparation, drilling, blasting, ground support, backfill, underground loading and hauling and material transport to the primary crusher on surface, as well as support and ancillary equipment operations and maintenance, power, direct labour and mine operations supervision staff.

Table 23-5: Underground Production Operating Cost Summary

Category	US$/t Processed Incremental	US$/t Processed Total
Stoping	4.56	7.75
Slot Rises	3.19	
Production Bogging		2.89
Trucking		4.43
Mine Auxiliary – Pumping	0.05	14.66
Mine Auxiliary – Ventilation	0.51	
Mine Auxiliary – Backfill	3.13	
Mine Auxiliary – Power	2.67	
Mine Auxiliary – Labour	4.07	
Mine Auxiliary – General	4.22	
Mine Supervision		5.01
Total Underground Mining Cost		**34.74**

23.3.3 Equipment Costs

A diesel price of US$1.00/L has been used based on diesel supply quotes provided by YPF, the largest fuel distributor in Argentina. The equipment schedule was derived using the underground mine plan and physicals prepared by Mining Plus for the scoping study. The equipment schedule is provided in Table 23-6 with costs for the ownership of this equipment provided in Table 22-5.

The overall average underground mine operating cost was estimated at US$34.74/t of material mined excluding pre-development costs (capitalised mine development costs during the pre-operational period).

Table 23-6: Underground Fleet and Ancillary Equipment Schedule

Equipment Class	Yr 0	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7
Development								
Jumbo's	3.0	5.0	5.0	5.0	3.0	2.0	1.0	1.0
Charge Up	2.0	3.0	3.0	3.0	2.0	1.0	1.0	1.0
Cablebolter								
Production Rigs								
Production Drill Rigs	1.0	2.0	2.0	2.0	2.0	2.0	2.0	1.0
Load & Haul Fleet								
17t Loaders- Conventional	1.0	2.0	3.0	2.0	2.0	2.0	2.0	1.0
17t Loaders- Teleremote	1.0	1.0	2.0	1.0	2.0	2.0	2.0	1.0
17t Loaders Total	**2.0**	**3.0**	**5.0**	**3.0**	**4.0**	**4.0**	**4.0**	**2.0**
55t Trucks	2.0	4.0	6.0	6.0	6.0	6.0	5.0	2.0
Charge Up (Emulsion)	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0

Equipment Class	Yr 0	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7
Water Cart	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Grader	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Service Truck	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Auxiliary Fleet								
ITC's Service Crew	3.0	4.0	4.0	4.0	3.0	2.0	2.0	1.0
ITC's Paste Crew	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Light Truck	1.0	1.0	2.0	2.0	2.0	2.0	1.0	1.0
Light Vehicles - UG	10.0	15.0	16.0	15.0	12.0	10.0	9.0	5.0
Light Vehicle - Supervisors	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Light Vehicle Maint & Elect	3.0	5.0	5.0	5.0	4.0	3.0	3.0	2.0
Light Vehicle Surface	3.0	5.0	5.0	5.0	4.0	3.0	3.0	2.0
Site Bus	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0
Total Light Vehicles & Light Trucks	**21.0**	**30.0**	**32.0**	**31.0**	**26.0**	**22.0**	**20.0**	**14.0**
Shotcrete Rig	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Agitator	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0
Box Hole Borer	1.0	1.0	2.0	2.0	2.0	2.0	2.0	1.0

23.3.4 Labour

The underground operating/maintenance labour is listed in Table 24-5.

The portion of the underground owner's team costs prior to the mill operating (Year -0) is included in the pre-production capital cost.

Table 23-7: Underground Mining Non-Operating Labour (Mine Supervision)

	Year							
Headcount	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**
UG MTS HC								
UG Geotech	4	4	4	4	4	4	4	4
UG Mine Planning	14	14	14	14	14	14	14	13
UG Survey	6	12	12	12	12	12	12	12
UG Geology	10	14	14	14	13	13	13	15
UG OPERATIONS HC								
UG Auxiliary	36	48	48	48	44	44	44	32
UG Blasting	16	32	32	32	20	20	8	8
UG Development Drilling	28	32	32	28	16	16	0	0
UG Haul	8	20	20	24	24	24	20	12
UG Load	12	16	20	20	16	12	12	12
UG Paste Crew	0	32	32	32	32	32	32	32
UG Production Drilling	8	8	12	12	8	8	8	8

	Year							
Headcount	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**
UG Dispatch	4	4	4	4	4	4	4	4
UG Supervision	11	11	11	7	7	7	7	7
UG MAINTENANCE HC								
Maintenance Supervision	1	1	1	1	1	1	1	1
Reliability & Planning	7	7	7	7	7	7	7	7
Maintenance Supervision	1	1	1	1	1	1	1	1
UG Auxiliary Maintenance	8	8	8	8	8	8	8	8
UG Development Drill Maintenance	8	8	8	8	8	8	8	8
UG Electrical Services	6	6	6	6	6	6	6	6
UG Haul Maintenance	8	8	8	8	8	8	8	8
UG Load Maintenance	8	8	8	8	8	8	8	8
UG Production Drill Maintenance	4	4	4	4	4	4	4	4
Generic MFM Services	24	24	24	24	24	24	24	24

23.3.5 Power

The total power demand for the UG operations has been calculated using the underground equipment schedule, adjusted for utilisation (operating hours) and demand factor (% of nameplate draw). This is detailed in Table 23-8. A contingency of 10% was applied with power costs of US$0.07/kWh applied which is the target power price to trigger the Memorandum of Understanding for renewable power with YPF as outlined in section 18.2. Current grid power is US$0.06/kWh.

Table 23-8: Underground Power Usage

	Total	Yr 0	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7
Jumbos									
Jumbo Percussion hours (Total)		6,367	10,928	11,889	11,663	6,685	4,290	1,401	346
Power Pack Hour Factor		1	1	1	1	1	1	1	1
kw		165	165	165	165	165	165	165	165
MWh		**1,313**	**2,254**	**2,452**	**2,406**	**1,379**	**885**	**289**	**71**
Raisebore									
Raisebore powerpack hours		2,598	2,504	2,151	1,700	960	823	0	0
kw		150	150	150	150	150	150	150	150
MWh		**390**	**376**	**323**	**255**	**144**	**123**	**0**	**0**
Longhole Rigs									
included in rates									
Pumps									
Mono Pump Operating Hrs		122,007	19,768	21,991	20,604	17,064	16,206	13,706	7,952
KW Rating		75	75	75	75	75	75	75	76
MWh		**9,151**	**1,483**	**1,649**	**1,545**	**1,280**	**1,215**	**1,028**	**604**

	Total	Yr 0	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7
Flygt Pumps OP Hrs		152	152	152	152	152	152	152	152
kw		8	8	8	8	8	8	8	8
MWh		**1**	**1**	**1**	**1**	**1**	**1**	**1**	**1**
Ventilation - Secondary									
# Fans		11	17	20	18	17	15	14	8
kw		150	150	150	150	150	150	150	151
Operating hours		5,694	5,694	5,694	5,694	5,694	5,694	5,694	5,694
MWh		**9,395**	**14,520**	**17,082**	**15,374**	**14,520**	**12,812**	**11,957**	**6,878**
Ventilation - Primary Fans									
Prim Fan count		2	3	3	3	3	3	3	3
kw		500	750	750	750	750	750	750	750
Operating hours		8,760	8,760	8,760	8,760	8,760	8,760	8,760	8,760
MWh		**4,380**	**6,570**	**6,570**	**6,570**	**6,570**	**6,570**	**6,570**	**6,570**
Pasteplant									
kw			100	100	100	100	100	100	100
Operating hours	m^3/hr	0	3	1,900	2,947	2,243	3,192	3,301	3,532
MWh		**0**	**0**	**190**	**295**	**224**	**319**	**330**	**353**
Total MWh		**24,630**	**25,203**	**28,267**	**26,446**	**24,118**	**21,926**	**20,176**	**14,478**
Contingency		10%	10%	10%	10%	10%	10%	10%	10%
Total Adjusted MWh	220,990	31,874	32,616	36,581	34,224	31,211	28,374	26,110	18,737
Power Costs (US$M)	**15.47**	**2.23**	**2.28**	**2.56**	**2.40**	**2.18**	**1.99**	**1.83**	**1.31**

23.3.6 Underground Grade Control

It has been assumed underground grade control will involve the sampling and assaying of all lateral development at 3 m intervals. Assay rates of US$40/assay have been taken from current supplier assay rates with this cost included as a standalone cost in the financial model.

23.4 Process Plant Operating Costs

23.4.1 Flotation + FTL (Flotation Tails Leach)

The process operating cost estimate accounts for the operating and maintenance costs associated with the 3,000 t/day process plant operation, supporting services infrastructure and tailings filtering. Operating costs associated with the backfill plant are included in the mine operating cost estimate.

Process plant operating costs have been estimated by Challenger's consulting metallurgists from first principles, using mechanical equipment specifications for estimation of power consumption, metallurgical test-work for reagent and grinding media consumption estimates, preliminary labour schedules and salary build-ups for process labour and maintenance labour.

The cost of spares was estimated as a fixed percentage of 5% of the mechanical equipment supply cost.

Quotations for consumables such as reagents, lime, binder and grinding media were obtained from suppliers inclusive of transportation to site. A unit power cost of US$0.07/kWh was assumed with power consumption based on the results of comminution testing and desired grind size. An allowance equal to the power usage of the comminution circuit was applied to the rest of the process plant.

The ore has been divided into three separate ore types based on gold and zinc grades. The ore types have slightly different flow sheets with the lower grade ore containing <1.5 g/t Au and <1.5% Zn (Ore Type A) processed via bulk flotation with cleaning stages.

The higher grade material ≥1.5 g/t Au with <1.5% Zn (Ore Type B) follows the same flow sheet with the additional of flotation tails leach (FTL).

For the material containing ≥1.5% Zn (Ore Type C) a stage of Pb-Cu rougher flotation and Zn flotation is added. The plant has been designed to batch all three ore types by bypassing the Cu-Pb and Zn flotation and/or the FTL circuit. For this reason, an availability of 72% has been assumed for the flotation circuit.

Given the slightly different flowsheets the three ore types have different reagent consumption which drives process costs. Annual and life-of-mine operating costs for the process and surface infrastructure for the three ore types are shown in Table 23-9, Table 23-10 and Table 23-11.

Table 23-9: Process Operating Cost for Ore Type C

Ore Type C (Au≥ 1.5 g/t Au, Zn≥ 1.5%) Sequential Flotation + FTL		
Category	Cost	
	Annual US$	Unit US$/t Ore
Operating Labor	1,915,984	1.82
Maintenance Labor	1,084,274	1.08
Power	2,100,000	2.10
Reagents and Consumables	9,499,572	9.50
Spares	1,712,329	1.71
Assays	100,000	0.10
Totals	**16,312,159**	**16.31**

Table 23-10: Process Operating Cost for Ore Type B

Ore Type B (Au≥ 1.5 g/t Au, Zn<1.5%) Bulk Flotation + FTL		
Category	Cost	
	Annual US$	Unit US$/t Ore
Operating Labor	1,915,984	1.82
Maintenance Labor	1,084,274	1.08
Power	2,100,000	2.10
Reagents and Consumables	5,306,407	5.31
Spares	1,712,329	1.71
Assays	100,000	0.10
Totals	**12,118,994**	**12.12**

Table 23-11: Process Operating Cost for Ore Type A

Ore Type A (Au<1.5 g/t Au, Zn<1.5%) Bulk Flotation no FTL		
Category	Cost	
	Annual US$	Unit US$/t Ore
Operating Labor	1,915,984	1.82
Maintenance Labor	1,084,274	1.08
Power	2,100,000	2.10
Reagents and Consumables	2,443,723	2.44
Spares	1,712,329	1.71
Assays	100,000	0.10
Totals	**9,256,310**	**9.26**

Reagent consumption and the cost of reagents in detail is contained in Table 23-12 to Table 23-14. It has been assumed that SIPX, R208, Lime, LPG and HCL will be sourced locally with unit cost based on reputable supplier rates. All other consumables will be imported with costs used inclusive of all import duties and taxes, seaborn freight and port charges, transport from Buenos Aires port to site based on transportation via 20 foot containers.

Table 23-12: Reagent Consumption for Sequential Flotation + FTL (Ore Type C)

Item	Consumption Rate	Annual Usage	Unit Cost	Annual Cost	Unit Cost
	Kg/t Ore	Kg	US$/Kg	US$	US$/t Ore
Crusher Liners	0.07	70,000	1.12	78,229	0.08
Grinding balls - SAG	-	537,000	1.32	708,602	0.71
Grinding balls - Ball	1.125	358,000	1.62	580,875	0.58
Regrinding balls (Ceramic) *	0.08	80,000	2.13	170,205	0.17
PAX	0	0	0.00	0	0.00

Item	Consumption Rate	Annual Usage	Unit Cost	Annual Cost	Unit Cost
	Kg/t Ore	Kg	US$/Kg	US$	US$/t Ore
SIPX	0.084	84,000	1.795	150,780	0.15
R208	0.0035	3,500	3.325	11,638	0.01
MIBC	0.0735	80,500	1.77	142,812	0.14
ZnSO4	0.49	490,000	3.07	1,504,476	1.50
CuSO4	0.7196	717,600	2.88	2,065,583	2.07
Flocculant	0.043	41,300	1.62	66,843	0.07
NaCN	0.595	544,900	2.88	1,568,473	1.57
Lime	1.566	2,782,600	0.112	311,651	0.31
SMBS	0.49	439,000	2.16	948,195	0.95
Oxygen/Air	-	-	-	-	-
Activated Carbon	0.088	79,200	2.14	169,580	0.17
LPG	0.098	88,000	1.074	94,512	0.09
NaOH	0.098	88,000	1.62	142,785	0.14
HCL	0.098	88,000	0.410	36,080	0.04
Fluxes	-	-	-	10,000	0.01
Soda Ash	2.185	2,184,700	0.34	**738,254**	0.74
Totals				**9,499,572**	**9.50**

Table 23-13: Reagent Consumption for Bulk Flotation + FTL (Ore Type B)

Item	Consumption Rate	Annual Usage	Unit Cost	Annual Cost	Unit Cost
	Kg/t Ore	Kg	US$/Kg	US$	US$/t Ore
Crusher Liners	0.07	70,000	1.12	78,229	0.08
Grinding balls - SAG	0	617,000	1.32	814,167	0.81
Grinding balls - Ball	1.125	411,000	1.62	666,871	0.67
Regrinding balls (Ceramic) *	0.08	33,500	2.13	71,273	0.07
PAX	0	63,000	2.16	136,073	0.14
SIPX	0.084	112,000	1.795	201,040	0.20
R208	0.0035	77,000	3.325	256,025	0.26
MIBC	0.0735	49,000	1.77	86,929	0.09
ZnSO4	0.49		3.07	0	0.00
CuSO4	0.7196	35,000	2.88	100,746	0.10
Flocculant	0.043	38,200	1.62	61,826	0.06
NaCN	0.595	462,000	2.88	1,329,848	1.33
Lime	1.566	1,478,300	0.112	165,570	0.17
SMBS	0.49	462,000	2.16	997,872	1.00

Item	Consumption Rate	Annual Usage	Unit Cost	Annual Cost	Unit Cost
	Kg/t Ore	Kg	US$/Kg	US$	US$/t Ore
Oxygen/Air	-	-	-	-	-
Activated Carbon	0.088	83,000	2.14	177,716	0.18
LPG	0.098	49,000	1.074	52,626	0.05
NaOH	0.098	49,000	1.62	79,505	0.08
HCL	0.098	49,000	0.410	20,090	0.02
Fluxes				10,000	0.01
Totals				**5,306,407**	**5.31**

Table 23-14: Reagent Consumption for Bulk Flotation + FTL (Ore Type A)

Item	Consumption Rate	Annual Usage	Unit Cost	Annual Cost	Unit Cost
	Kg/t Ore	Kg	US$/Kg	US$	US$/t Ore
Crusher Liners	0.07	70,000	1.12	78,229	0.08
Grinding balls - SAG	0.617	617,000	1.32	814,167	0.81
Grinding balls - Ball	0.411	411,000	1.62	666,871	0.67
Regrinding balls (Ceramic) *	0.0335	33,500	2.13	71,273	0.07
PAX	0.063	63,000	2.16	136,073	0.14
SIPX	0.112	112,000	1.795	201,040	0.20
R208	0.077	77,000	3.325	256,025	0.26
MIBC	0.049	49,000	1.77	86,929	0.09
ZnSO4	0		3.07	0	0.00
CuSO4	0.035	35,000	2.88	100,746	0.10
Flocculant	0.0382	38,200	1.62	61,826	0.06
Totals				**2,473,179**	**2.47**

23.4.2 Assay Lab

Assays will be undertaken at the existing commercial assay lab in San Juan which offers rapid turnaround and is located 1.5 hours by road from site. The budget covers the cost of 2,500 samples per annum including transport to the lab based on current rates.

23.4.3 Process Plant Labour

The labour schedule for the process plant has been estimated as detailed in Table 23-15.

Table 23-15: Process Plant Labour

Headcount	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
PROCESS METALLURGY HC								
Metallurgy	3	3	3	3	3	3	3	3
Process Laboratory	1	6	6	6	6	6	6	6
PROCESS OPERATIONS HC								
Comminution		8	8	8	8	8	8	8
Flotation		8	8	8	8	8	8	8
Process Auxiliary		6	6	6	6	6	6	6
Process Supervision		5	5	5	5	5	5	5
Process Dispatch		4	4	4	4	4	4	4
Refinery		4	4	4	4	4	4	4
FIXED PLANT MAINTENANCE HC								
Fixed Plant Maintenance		26	26	26	26	26	26	26
Maintenance Supervision	1	1	1	1	1	1	1	1
Reliability & Planning	5	5	5	5	5	5	5	5

23.4.4 CIL (Carbon in Leach) Operating Cost Estimate

A process operating cost estimate for operating and maintenance costs associated with the 3,000 t/day process CIL only plant operation, supporting services infrastructure and tailings filtering was prepared. This involves the existing comminution and gravity, with the flotation circuit bypassed and the gravity tails direct to CIL. In practice, should CIL be pursued as the process route rather than flotation, the plant would be installed without the flotation circuit.

Like the processing costs estimated for flotation, the CIL process plant operating costs have been estimated by Challenger's consulting metallurgists from first principles, using mechanical equipment specifications for estimation of power consumption, metallurgical test-work for reagent and grinding media consumption estimates, preliminary labour schedules, and salary build-ups for process labour and maintenance labour. The cost of spares were estimated as a fixed percentage of 5% of the mechanical equipment supply cost.

Operating costs for CIL only are summarised in Table 23-16 with reagent and consumable consumption outlined in Table 23-17.

Table 23-16: Process Operating Costs Gravity plus CIL

Process Costs Gravity plus CIL		
Category	Cost	
	Annual US$	Unit US$/t Ore
Operating Labor	1,676,320	1.68
Maintenance Labor	964,442	0.96
Power	2,100,000	2.10
Reagents and Consumables	5,460,322	5.46
Spares	1,092,538	1.09
Assays	100,000	0.10
Totals	**11,393,622**	**11.39**

Table 23-17: Reagent and Consumable Consumption Gravity plus CIL

Item	Consumption Rate	Annual Usage	Unit Cost	Annual Cost	Unit Cost
	Kg/t Ore	Kg	US$/Kg	US$	US$/t Ore
Crusher Liners	0.07	70,000	1.12	78,229	0.08
Grinding balls - SAG	0	0	1.32	0	0.00
Grinding balls - Ball	1.125	1,125,000	1.62	1,825,377	1.83
Regrinding balls (Ceramic) *	0.08	0	2.13	0	0.00
NaCN	1	1,000,000	2.88	2,878,460	2.88
Lime	1.566	1,566,000	0.11	175,392	0.18
Activated Carbon	0.088	88,000	2.14	188,422	0.19
LPG	0.098	98,000	1.07	105,252	0.11
NaOH	0.098	98,000	1.62	159,011	0.16
HCL	0.098	98,000	0.41	40,180	0.04
Fluxes				10,000	0.01
Totals				**5,460,322**	**5.46**

23.5 General and Administrative Operating Costs (G&A)

G&A costs predominantly include labour, administrative and miscellaneous costs associated with the Finance, IT, Supply Chain, Warehouse, Human Resources, Camp Administration/ Maintenance, Health, Safety, Training, Security, Environment, Permitting, Government and Community Affairs, Communications and Executive (General Management) functions.

An allowance has been made for insurance and local compliance costs, as well as for community development grants.

Camp accommodation, catering, laundry, cleaning and the cost of transporting personnel from San Juan to Hualilan and vice-versa has been incorporated into G&A. This is based on existing unit rates from the temporary camp established at Hualilan. Average camp occupancy over the key production period is 210 beds.

Power draw has been estimated by benchmarking with power consumption figures benchmarked using similar operations in South America.

G&A costs that occur prior to the process plant operating are included in the pre-production capital costs (refer to section 22.11). The summary of operational G&A costs is in Table 23-18.

An approximation of headcount for G&A has been included as Table 23-19.

Table 23-18: General and Administrative (G&A) Operating Costs

Annual G&A Costs	Annual Cost US$
Accommodation, Catering, Laundry, Cleaning & Personnel Transport	1,406,053
Commercial (Finance, IT, Supply Chain/ Warehouse)	1,025,516
Human Resources, Recruitment, Camp Admin + Maintenance	631,822
Health, Safety, Security, Environment & Training	799,160
Executive, Community Affairs, Government Affairs, Permitting & Communications	696,999
Insurance	600,000
Community Grants	50,000
Local compliance costs	100,000
Average Power Draw for facilities: Truckshop, Warehouse, Camp and Admin building	281,459
TOTAL G&A Costs	**5,591,010**

Table 23-19: General and Administrative (G&A) Labour Model

Headcount	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
GM & LTO								
General Management	3	3	3	3	3	3	3	3
Permits	1	1	1	1	1	1	1	1
Communications	1	1	1	1	1	1	1	
Community & Government Affairs	1	1	1	1	1	1	1	1
COMMERCIAL								
Finance	8	10	10	10	10	10	9	7
IT	2	3	3	3	3	3	3	2
Supply Chain	4	4	4	4	4	4	4	4
Warehouse	10	10	10	10	10	10	10	10
HR & CAMP								

Headcount	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7
Human Resources & Recruitment	6	5	5	5	5	5	5	4
Camp Administration & Maintenance	16	16	16	16	16	16	16	16
Catering/ Cleaning (Contractors)	40	40	40	40	40	40	40	40
HSSET								
Environment	3	3	3	3	3	3	3	3
G&A Safety	2	2	2	2	2	2	2	2
Process Safety		2	2	2	2	2	2	2
OP Safety	2	2	2					
UG Safety	3	3	3	3	3	3	3	3
Training	3	3	3	3	3	3	3	1
Emergency Services	6	6	6	6	6	6	6	6
Site Doctor/ Nurses (Contractor)	6	6	6	6	6	6	6	6
Security	6	6	6	6	6	6	6	6
Security Guards (Contractor)	32	32	32	32	32	32	32	32

23.6 Tailings Operating Costs

The cost for construction of the TSF has been included in the CAPEX and includes the cost for quarry and transport of materials from the open pit; the labour and equipment cost associated with stacking and general construction of the embankment, including a premium for this to be executed by a contractor; and, for materials including geotextiles, HDPE liner and the leakage collection and pumping system (LCRS).

There are no explicit operating costs associated with the TSF. Transfer of tails slurry from the processing facility to the TSF has been included in the processing cost (as has the assumption of light power draw if pumping is required from the LCRS). For simplicity, basic labour costs associated with controlling TSF deposition patterns have been assumed to be absorbed in process labour and supervision. The TSF civil cost estimate is detailed in Table 23-20.

Table 23-20: TSF Civil Cost Estimate

Component	Unit Cost ($/t placed)	Inclusions
TSF/ Civils	0.60/t placed	Equipment: 1x D8 Dozer, 1x 16M Grader, 1x CS54 Roller, 1x Excavator 320F, 1x 40t Truck Price Incl: Labour, Fuel/ lube, GET, Maintenance, contractor margin.

24 ECONOMIC ANALYSIS

24.1 Introduction

All financial numbers referenced are in United States Dollars (USD$) unless otherwise stated. No escalation of revenue and costs has been incorporated. Income tax is assumed at the Argentinian Taxation Office prescribed corporate income tax rate and is treated in this study as a flat rate of (35%), with previous exploration offset as carried forward and as tax losses that may be available and realised by Challenger in accordance with the Argentinian tax laws. Totals in tables may not reflect summed components precisely due to rounding.

The results of the analysis show the Hualilan Gold project to be economically robust. The net present value of the net cashflow with an 5% discount rate (NPV5) is $409.30 million on a pre-tax project basis, and US$294.78 million post-tax, using a base gold price of US$1,750/oz. Project internal rates of return (IRR) are 75.23% pre-tax and 66.04% post-tax.

The project payback period (i.e. process plant start-up until all initial expenditures are recovered) at a gold price of US$1,750/oz is expected to be 1 year 3 months from production start date and 2 years 3 months from construction start.

Like most gold mining projects, the key economic indicators (NPV and IRR) are most sensitive to changes in gold price. As such, a 10% decrease in gold price would reduce the post-tax NPV5 to $315.30 million pre-tax and $234.17 million post tax and the IRR to 59.50% pre-tax and 53.59% post tax.

24.2 Cash Flow Model

A financial model in Microsoft Excel was developed for Challenger by an independent financial model expert, to allow economic evaluation of the Hualilan Gold project. The model has sufficient capability to assess the capital structure for the development of the project, including the project's debt carrying capacity.

Challenger provided certain capital and operating inputs, including mining physicals, some mining costs, recoveries and payabilities. The financial model imports the inputs into the cash flow model which allows evaluation and due diligence of a number of scenarios. The model includes production from Indicated Resources (81%) and Inferred Resources (19%) .

The model assumes the following broad project scope:

- Total project duration is 7.75 years.
- The schedule commences with the mining capital development phase, plant establishment and preparation of infrastructure occurring for months 1 to 12.
- The operating phase follows with mining commencing in month 1 (open pit), month 7 (underground) and ore processing in month 13.

- The open pit mining proceeds for 4.25 years with a total movement of 9,663,110 tonnes at a strip ratio of 6.37:1, producing 1,310,750 tonnes of PMI at 3.41 g/t for 143,746 ounces of gold, 22.31 g/t silver for 940,037 ounces of silver, 1.52% for 19,892 tonnes of zinc and 0.33% for 4,307 tonnes of lead delivered to the ROM pad.
- Underground mining movement is 5,797,590 tonnes of PMI at 3.61 g/t for 672,283 ounces of gold. 12.09 g/t silver for 2,254,004 ounces of silver, 1.35% for 78,484 tonnes of zinc and 0.08% for 4,656 tonnes of lead, delivered to the ROM pad.
- Milling commences in month 13, with the mill operating full time at around 1.0 Mtpa. Ore processing continues for 6.75 years.
- Recovered gold totals 782,223 ounces, silver 2,971,826 ounces, zinc 61,986 tonnes and lead 3,206 tonnes over the project life.
- Payable gold totals 737,922 ounces, payable silver 1,776, 948 ounces, payable zinc 52,034 tonnes and payable lead 3,014 tonnes over the project life.

Financial parameters used in the project financial analysis are:

- No financing.
- Income tax is assumed at the Argentinian Taxation Office prescribed corporate income tax rate and is treated in this study as a flat rate (35%), with an offset for carried forward tax losses and a double deduction for exploration associated expenses allowed under the Argentinian tax laws.
- Nil allowance for cost escalation.
- No hedging
- Production royalties of 3.0% for all metals, 7.69% export tax and a community tax of 1.5%.
- Average recovery for LOM of 95.9% gold, 93.0% silver, 90.9% zinc and 89.2% lead.
- Treatment, refining and penalty costs average (per ounce of gold produced) $80, transport and freight costs $113, mine operating costs $393, process operating costs $126 and general and administration costs $51.
- NSR Payabilities (payability after TC and RC's, freight and penalties) average 88.4% for gold, 54.8% for silver, 73.1% for zinc and 93.6% for lead.
- Capital and operating costs are detailed in Sections 23 and 24 and include all site costs.
- Gold sales were priced at $1750 per ounce and were fixed for the project.

The mining and processing schedules used as the basis for the economic evaluation are based on the LOM production schedule and the detailed capital and operating cost estimates in sections 22 and 23 of this scoping study. LOM production and cost estimate templates for input into the financial model are included in section 15.

24.3 Mine Production and Mill Feed

The mine production schedule presented in section 15 is summarised into an annual mine production and mill feed schedule in

Table 24-1. Life of mine mill feed totals 7.1 Mt at an average grade of 3.6 g/t Au, 14.0 g/t Ag, 1,4% Zn, 0.13% Pb with an assumed processing rate of 1.05 Mtpa.

Mining commences 12 months prior to the commencement of mill operations and the ore is milled over a 6.75 year period. It is assumed that ore will be either directly fed to the crusher or stockpiled adjacent to the crusher and rehandled to meet the processing schedule. The ore processing rate is a function of the testwork results summarised in section 11.

Table 24-1: Mine Production and Mill Feed Schedule

	Units	LOM	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8
Mining										
Total PMI	Mt	7,108	514	1,020	1,144	1,121	996	1,018	1,005	290
Grade Mined	Au g/t	3.57	3.78	3.69	3.39	3.44	3.69	3.43	3.67	3.62
Grade Mined	Ag g/t	13.98	22.51	14.92	15.96	14.38	10.25	13.01	12.78	8.61
Grade Mined	Zn %	1.38	1.31	1.87	1.18	0.99	1.28	1.75	1.59	0.73
Grade Mined	Lb %	0.13	0.32	0.18	0.10	0.12	0.06	0.10	0.12	0.03
Processing										
Contained Gold	oz	816,028	-	157,696	124,697	123,561	130,109	115,864	119,105	44,966
Contained Silver	oz	3,194,040	-	683,732	518,460	523,687	346,393	435,949	423,430	262,390
Contained Zinc	klb	98,376	-	22,964	13,158	11,297	12,891	17,984	16,109	3,973
Contained Lead	klb	7,658	-	2,074	1,052	967	664	1,069	1,233	599
Production										
Gold Production	oz	782,223	-	150,520	119,653	118,732	125,004	111,002	114,137	43,175
Silver Production	kt	2,971,826	-	634,857	489,201	494,428	324,191	403,600	395,327	230,222
Zinc Production	t	89,389	-	21,712	11,869	10,109	11,577	16,563	14,601	2,959
Lead Production	t	6,834	-	1,844	893	832	583	991	1,150	541
Production Costs (payable gold)										
C1 Cost	$/oz	527	-	335	676	653	557	512	495	648
AISC	$/oz	830	-	585	942	941	863	867	835	1,102
Cashflows										
Assumed Gold Price	$/oz	1,750	-	1,750	1,750	1,750	1,750	1,750	1,750	1,750
Assumed Silver Price	$/oz	20	-	20	20	20	20	20	20	20
Assumed Zinc Price	$lb	1.15	-	1.15	1.15	1.15	1.15	1.15	1.15	1.15
Assumed Lead Price	$lb	0.94	-	0.94	0.94	0.94	0.94	0.94	0.94	0.94
Revenue	$M	1,465.03	-	293.77	220.99	217.07	227.29	214.58	215.51	75.81
Net Operating Cashflow	**$M**	**384.49**	**(141.05)**	**119.71**	**76.29**	**72.80**	**74.19**	**63.46**	**79.17**	**39.92**
Cumulative Operating Cashflow	**$M**	**384.49**	**(141.05)**	**(21.34)**	**54.95**	**127.75**	**201.94**	**265.4**	**344.57**	**384.49**

Note: Figures are presented on an annualised basis from the commencement of project development, rounding errors may occur

C1 = operating costs include all mining and processing costs, site administration costs, transport, refining charges.
AISC = C1 costs plus royalties and sustaining capital however excludes corporate head office costs

24.4 Metal Price Assumptions

A base case gold price of $1,750/oz was utilised to evaluate the project.

This price was approximately $230/oz lower than the prevailing gold price during the completion of the study.

The gold price has been relatively stable in recent years (Figure 24-1) however the impact of higher or lower gold prices have been addressed as part of the sensitivity analysis completed within this section.

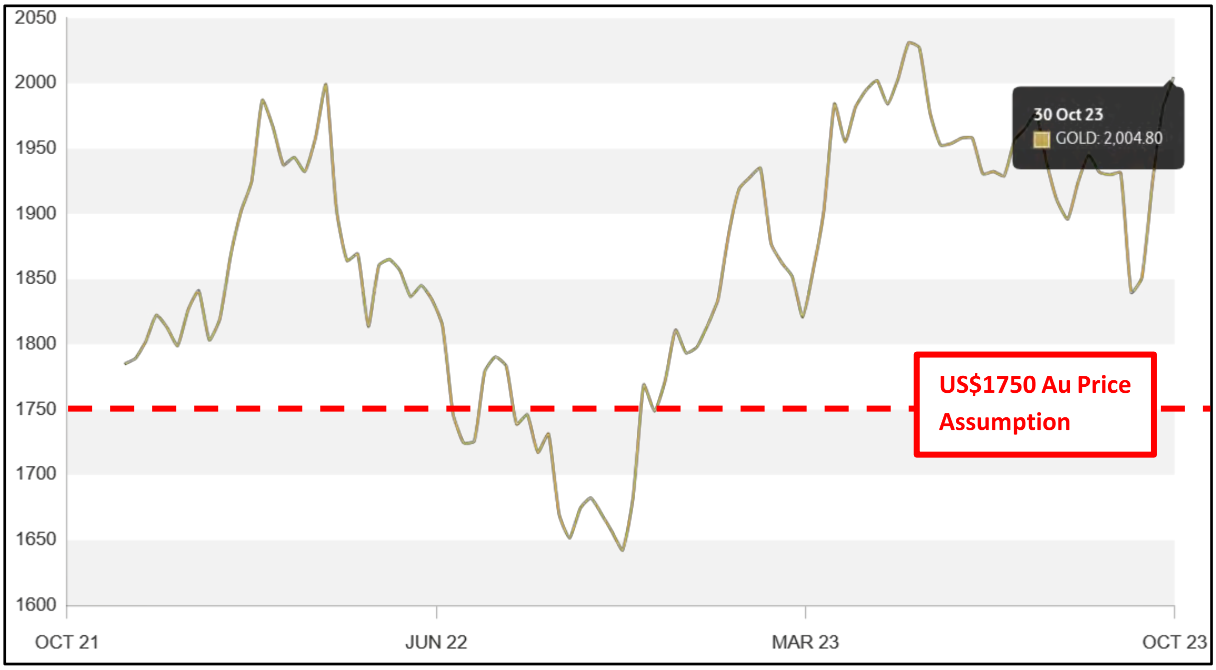

Figure 24-1: Historical Gold Price (source Perth Mint)

24.5 Cost Estimates

All cost estimates are in United States dollars as at the first half 2023.

Capital, including pre-production cost estimates, are presented in section 22 of this report. Initial capital is estimated at US$133.7 million and sustaining capital is estimated at US$59.0 million over LOM, including development. Pre-production operating costs are an additional US$18.4 million.

Operating cost estimates are presented in section 23 of this report. The estimated pre-production mining cost is calculated as the total cost of pre-production mining, processing and operating activities on an accrual's basis. 30-day allowances have been made for creditor-days or deferred payment arrangements.

24.6 Economic Results

Project economics for are presented in the table below. These base case economic results for the Hualilan Gold project are favourable, however further work to improve the economics is ongoing.

The cash flow financial model for the Scoping Study is summarised by year in **Error! Reference source not found.**.

Table 24-2: Staged Project Economics Summary

	Units	
Project Life	Years	7.75
Total Potential Mining Inventory (contained)[1]	7.1Mt @ 3.57 g/t Au for 816 koz Au, 13.98 g/t Ag for 3,194 koz Ag, 1.38% Zn for 197,066 klbs Zn and 0.11% Pb for 15,066 klbs Pb.	
Gold Sales	Oz	737,922
Silver Sales	Oz	1,776,948
Zinc Sales	klbs	114,683
Lead Sales	klbs	6,645
Revenue	**$M**	**1,465.03**
Treatment and refining costs	$M	(58.7)
Transport and freight costs	$M	(83.38)
Net Revenue before Royalties (NSR)	**$M**	**1,322.95**
Royalties and state taxes	$M	(166.27)
Net Revenue after Royalties	$M	1,156,.68
Mining Operating expenses	$M	(287.69)
Process Operating expenses	$M	(93.13)
G&A Operating expenses (during production)	$M	(37.74)
Operating Margin (EBITDA)	**$M**	**738.1**
Initial Plant and Infrastructure Capex	$M	(133.7)
Initial Mine Development and Mobilisation	$M	(18.42)
Initial Capex (incl. Pre-production Operating costs)	**$M**	**(152.17)**
Underground Mine Sustaining capex	$M	(45.0)
Mine infrastructure and plant sustaining capex	$M	(13.97)
Sustaining Capex: Total	**$M**	**(59.87)**
Total Capital and Sustaining Capital	**$M**	**(211.09)**
Undiscounted Cashflow Pre-tax	**$M**	**527.03**
Tax Payable	$M	(142.54)
Undiscounted Cashflow after tax	**$M**	**384.49**

1: The total PMI contained underpinning the above production target has been prepared by a Competent Person or Persons in accordance with the requirements of the JORC (2012) Code. Refer to JORC tables, Qualifications and Competent Persons Statements. Based on assumed throughput of 1.0Mtpa.
2. Refer Chapters 23 and 24 for a detailed breakdown of costs, pre-production costs are net of commissioning revenue
3. All figures are presented in nominal United States dollars, tax is applied at a flat corporate rate of 35%, unadjusted for inflation

4. Rounding errors may occur

24.7 Argentinian Inflation Forecasts

Argentina is currently experiencing hyperinflation climbing above an annualised rate of 100% in 2023. This makes forecasting of inflation rates and foreign exchange rates extremely difficult, and such forecasts are subject to considerable uncertainty.

The country has experienced hyperinflation in the past including a period between 1975 and 1990 where it had an average annual inflation rate of 300%. While the government is taking steps to reduce inflation, it is not easy to forecast how successful the strategies will be. From January 2023 to September 2023, annualised inflation has progressively increased to over 120% on an annualised month-to-month basis.

For the purposes of this study, costs were estimated and modelled in real 2023 US dollars to coincide with the commodity pricing, also modelled in real 2023 US dollars. Local contributions to costs, such as local labour rates, were modelled at current US dollar productivity rates and converted to US dollars at foreign exchange rates that reflected actual US dollars and not reduced over time for the depreciation of the Argentinian Peso in a Hyper-inflation environment. No further adjustments to foreign exchange assumptions were made to cost estimates within the economic model.

An adjustment to foreign exchange was made to exploration losses and carry forward tax losses balances to account for the effect of the high inflation rate assumed for the Argentinian Peso. This adjustment reduced the erosion of the exploration losses and carry forward tax losses balances in real terms and ensured that the full expenditure in US dollars was ultimately deducted for the determination of taxable income.

24.8 Sensitivity Analysis

The project is most sensitive to changes in the gold price. The NPV5 and IRR sensitivity to changes in gold price are shown in Table 24-3 on a LOM basis.

Table 24-3: Gold Price Project Sensitivity

Gold Price Sensitivity - USD	$/oz	1,650	1,750	1,850	Spot (1,980)
Project Net Cashflow, Pre-Tax (USD)					
Project Net Cashflow, Pre-Tax	$M	462.43	527.03	591.63	675.60
Pre-tax NPV5%	$M	355.70	409.03	462.37	531.70
Pre-tax IRR	% p.a.	66.23	75.23	84.29	96.16
Payback period from production start	Years	1.91	1.3	1.3	1.3
Project Net Cashflow, Ungeared, Post-Tax (USD)					
Project Net Cashflow, Post-Tax	$M	342.50	384.49	426.47	481.06
Post-tax NPV5%	$M	260.40	294.78	329.05	373.54
Post-tax IRR	% p.a.	59.11	66.04	72.66	80.95

The project sensitivity to ±10% changes in key operating parameters are also shown below. These include changes to gold price, plant and G&A opex, mining opex, all metals prices, capex and overall opex with their sensitivity to the staged post-tax, ungeared NPV5 presented in Figure 24-2.

A review of the sensitivity figures indicates that the post-tax, ungeared NPV5 and IRR are most sensitive to changes in the gold price, and all metals prices.

The project is more sensitive to changes in operating costs (mining, processing, site G&A) than capital costs, a result of the low base case capital costs for the project.

From the sensitivity analysis it is apparent that the project is most sensitive to changes in the realised gold price and metal prices. Due to the low level of capital expenditure required to go into production, the project economics are not overly sensitive to capex expenditure within the scoping study estimation accuracy. To illustrate, consider the following:

- A 20% reduction in gold price would reduce the LOM project NPV5 (pre-tax) to around US$221.56 M whilst a 20% increase would deliver a LOM project NPV5 of US$596.5 M.
- A 20% increase in operating cost reduces the LOM project NPV5 to around US$302.02 M, whilst a cost reduction of 20% results in US$516.04 M NPV5.
- A 20% increase in all metals prices to a LOM NPV5 of US$622.01 M, whilst a 20% reduction delivers a NPV5 of US$196.06 M.

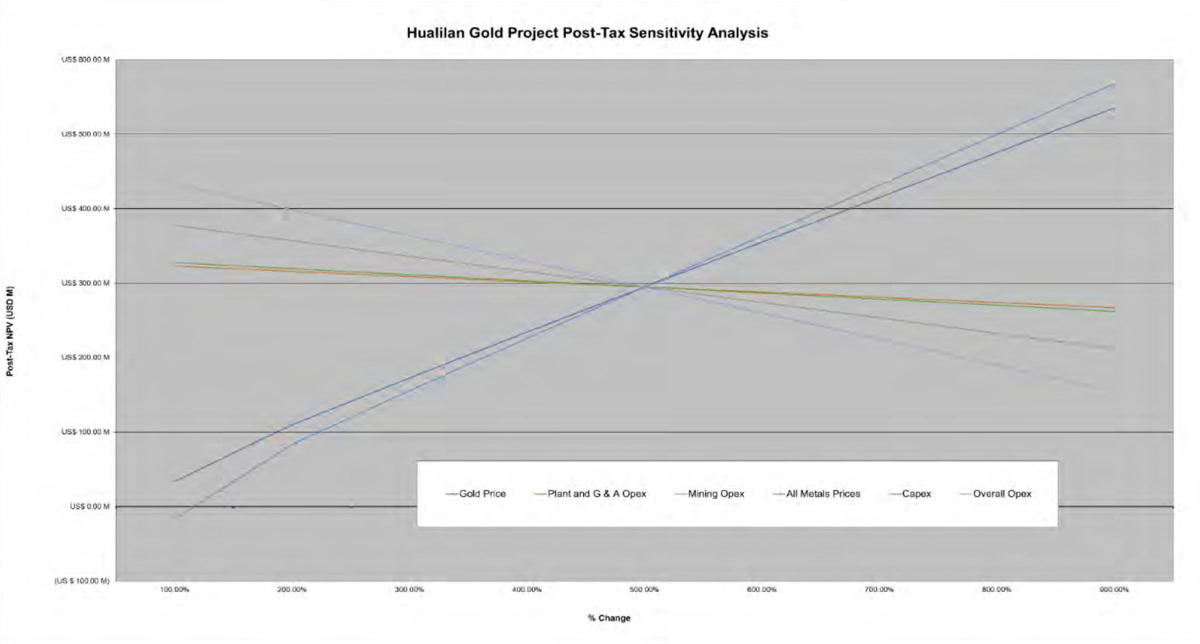

Figure 24-2: NPV 5% Sensitivity Chart

The sensitivity analysis demonstrates the project would require a grade reduction or gold price reduction in the order of 44% to reach the breakeven gold price from an NPV perspective.

24.9 Financing

Challenger has 100% ownership of the Hualilan Gold Project, with US$15 M unsecured debt and no other covenants and no security held over the project. This clean ownership structure enhances opportunities and provides maximum flexibility for potential funding structures for the project development.

The study has provided positive economic metrics and the planned timetable of activities to deliver key development milestones that directors and management believe is conducive to the funding of the project. The positive technical and economic fundamentals provide a platform for discussions on debt, equity financiers and forward sales arrangements.

The Company has held early-stage fruitful discussions with potential project finance providers in Argentina, the US and Europe, and various international royalty and stream providers. These initial discussions have been positive and indicate that the Company will likely have a range of non-dilutive finance options available for consideration.

The Company’s board has experience in financing and in developing projects internationally, and several board members and senior executives have been involved with Challenger since the RTO in 2019.

Current management has international experience in bringing companies into production, including Dominion Minerals Corp, which acquired and developed the Cerro Corcha Gold/Copper project in Panama, with a project NPV in excess of US$500 million and the Antas Copper Mine in Brazil, which was acquired by Oz Minerals Limited in 2018 for approximately US$400 million.

The Company's major shareholders comprise high quality investment funds including the BlackRock Group (BlackRock Inc. and its subsidiaries).

The Company's aim will be to avoid dilution to existing shareholders as much as possible.

All the material assumptions on which the forecast financial information is based has been included in this scoping study.

For the reasons outlined above, the board believes that there is a 'reasonable basis' to assume that future funding will be available and securable.

25 ADJACENT PROPERTIES

There are no advanced exploration projects immediately adjacent to the Hualilan project. Within San Juan Province there is one operating mine and several advanced projects, none of which are of similar geological style to Hualilan. Most of these are located in the frontal part of the Argentinian cordillera. More common in this zone are porphyry style Cu, Au, Ag, Mo or epithermal (low and high sulphidation) Au, Ag. Figure 25-1 shows all of the projects.

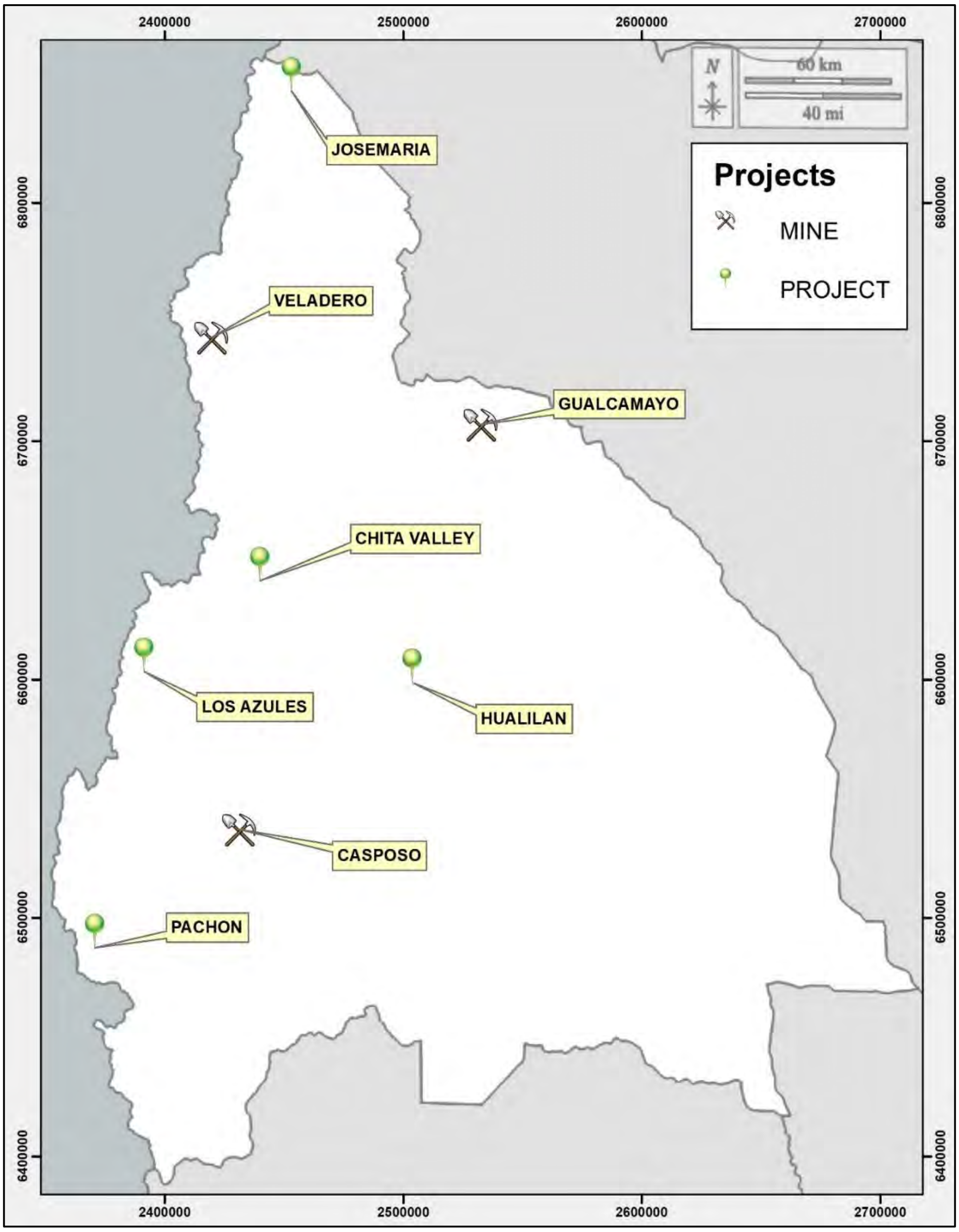

Figure 25-1: Location of some mines and mineral deposits in San Juan.

25.1 Veladero

Veladero is an operating mine which is a Joint Venture between Barrick Gold Corporation (50%) and Shandong Gold Group Co. Ltd. (50%) with Barrick as the operator. The mine is 370 kilometres northwest of San Juan City. The deposit is a high sulphidation gold-silver hosted by volcaniclastic sediments, tuffs, and volcanic breccias related to a Miocene dome complex. Mining is an open pit operation at 4,000 to 4,850 metres above sea level that has been operating since 2005. Gold is recovered using cyanide-heap leaching and a Merrill-Crowe circuit for gold and silver recovery. Average gold recovery is approximately 77% and average silver recovery is approximately 11%.

25.2 Gualcamayo

Gualcamayo is a gold mine that was operated by Minas Argentinas S.A., that has been on care and maintenance since 2022. The mine is located at altitude of 1,940 to 2,670 metres above sea level, approximately 270 km north of San Juan City and 110 km from the city of San Jose de Jachal. Gold mineralisation occurs in limestone of the San Juan Formation, within breccia and fracture zones. Mineralisation is related to a hydrothermal event extending into the surrounding marble and limestone. A Mineral Resource is estimated at cut-off grades between 0.2 to 0.4 g/t Au for mineralisation recoverable by open pit and 1.85 g/t Au for underground recoverable mineralisation.

25.3 Casposo

The Casposo Mine (Austral Gold Limited 100%) has been on care and maintenance since the mine ceased production in 2016. The mine is located approximately 150 km west of the city of San Juan, in the department of Calingasta, San Juan Province, and is 166 kilometres via sealed road from the Hualilan project. It was an open pit and underground operation, processing 400 ktpa. The open pit started in 2010, and underground production started in 2013. Silver grades improved and gold production declined as the mine got deeper. Gold recovery was 90% and silver recovery was 79%. Processing was by whole mineralised material cyanide leaching, extraction by-current decantation and filtration for liquid-solid separation and Merrill-Crowe for recovery of the metal from the leach solution.

25.4 Josemaria

Josemaria is a porphyry copper-gold deposit, 100% owned by Josemaria Resources Inc which completed a feasibility study in 2020. The project is located approximately 7 hours drive from San Juan in the frontal Cordillera at elevations ranging from 4,000 to 4,900 metres above sea level. Mineral zones of the deposit are defined by the relative abundance of chalcopyrite, pyrite and chalcocite. Proposed mining is by open pit with throughput of 150 ktpd. Processing involves flotation. Metallurgical recovery is expected to be 85% for Cu, 63% for Au and 72%

for Ag. Arsenic is present in the mineralisation and arsenic concentrations will have to be managed to reduce penalties.

25.5 Pachon

Pachon is a porphyry copper and molybdenum porphyry 100% owned by Glencore plc. The deposit is in the southwest of San Juan Province, 159 km from the town of Barreal, 363 km from San Juan at between 3,600 and 4,200 metres above sea level. The bulk of the mineralisation takes the form of primary disseminated chalcopyrite and molybdenite with a relatively small secondary copper enrichment weathering zone which contains chalcocite and minor covellite.

25.6 Los Azules

Los Azules is a copper, gold, silver molybdenum porphyry located 80 kilometres WNW of Calingasta, at between 3,300 and up to 4,500 metres above sea level. Porphyry copper mineralisation and hydrothermal alteration are spatially, temporally, and genetically related to the intrusions. A proposed open pit mine and copper concentrator plant would export via ports in Chile. Copper recovery to concentrate is expected to be 91% at a concentrate grade of 30% Cu.

26 OTHER RELEVANT DATA AND INFORMATION

26.1 Risk Analysis Approach

A risk assessment was undertaken by the parties involved in this report. Identifying the major risks associated with the project at a high level, noting that a more detailed, technical/operational risk assessment will need to be undertaken at future study phases, utilising this risk assessment as a baseline and reassessing these risks.

The below is the risk matrix used with details as to the consequence being identified.

RISK MATRIX & CRITERIA

	Consequence Descriptors				Likelihood Descriptors				
Category	People	Environment (reflects change from conditions existing before incident occurred)	Economic/ Opportunity Loss (In AUD)	Reputation	Rare Is expected to occur on rare (<10%) occasions	Unlikely Is expected to occur on infrequent (10% to 25%) occasions	Possible Is expected to occur on some (25% to 75%) occasions	Likely Is expected to occur on many (75% to 90%) occasions	Almost Certain Is expected to occur in most (>90%) occasions
Catastrophic	Fatality or Permanent Disabling Injury/Illness	Disastrous and/or widespread environmental impact	Huge financial loss (>$500,000)	Damage to corporate reputation at international level/significant negative impact on international workbook	H (15)	H (10)	E (6)	E (3)	E (1)
Major	Lost Time Injury/Illness	Serious environmental impact	Major Financial Loss ($50,000 to $500,000)	Damage to corporate reputation at national level/significant negative impact on national workbook	M (19)	H (14)	H (9)	E (5)	E (2)
Moderate	Alternate Duties Injury/Illness	Substantial environmental impact	High Financial Loss ($20,000 to $50,000)	Ongoing adverse impact to relationship with Client	L (22)	M (18)	H (13)	E (8)	E (4)
Minor	Medical Treatment Injury/Illness	Small and/or localised environmental impact	Medium Financial Loss ($5,000 to $20,000)	Minor medium term damages to relationship with Client	L (24)	L (21)	M (17)	H (12)	E (7)
Insignificant	Minor Injury/Illness	Little or no environmental impact	Low financial loss (<$5,000)	Low level repairable damage to relationship with Client	L (25)	L (23)	M (20)	H (16)	H (11)

The below table identifies the level of risk ranking and their overall descriptors:

Extreme	Intolerable; *immediate* action required to eliminate, reduce or transfer risk; monitor & review to confirm at ALARP [as low as reasonably practicable]
High	Intolerable; action required to eliminate, reduce or transfer risk; monitor & review to confirm at ALARP
Medium	Tolerable with effective safeguards; monitor and review to confirm at ALARP
Low	Acceptable; manage with safeguards

26.2 Key Risks Identified and Proposed Mitigation Measures

Segment	Area	What could happen?	What could cause it?	What could it lead to?	What are the existing or potential controls?		Highest Consequence	Likelihood of Occurring	Risk Ranking
Operational	**Geological Model and Resource Estimation**	Overestimation or underestimation of the tonnage and/or grade of the Resource Estimate	**External Factors**: - Orebody variations **Internal Factors**: - Inadequate sampling. - Technical/ modelling errors.	**Financial**: Project is or becomes financially unviable. Over or under-investment in (or investment in the wrong type of) property, plant and equipment. Reduction in operating margin. **LTO**: Government, personnel, community and investor expectations not met	- Resource estimation completed by competent person. Peer reviewed. - Use of appropriately certified labs for test work.		Major	Rare	L (19)
Operational	**OP Geotechnical**	Open pit slope geotechnical failure (multi-bench, single bench, rockfall and interactions with voids and underground workings). Also includes natural slopes.	**External Factors**: - Climatic events and natural disasters. **Internal Factors**: - Poor design parameters, potentially due to insufficient rock mass and structural model understanding. - Mining, blasting and general operating practices (incl. drainage and water management). - Quality of geotechnical monitoring and assessment. - Improper Void management	**HSP**: Safety incidents as a result of slope failure. **Environment**: Local ecosystem may become geotechnically unstable. **Financial**: Loss of asset/s. Operational disruptions. High remediation or recovery costs. **LTO**: Increased regulatory scrutiny. Community/ stakeholder engagements around perceived or real unsafe conditions. Reputational loss.	- Slope and water monitoring. Regular geotechnical assessments. Geological Reconciliations. - Daily inspections. - TARP/ management system around rockfall incl. exclusion zones and hazard maps. - Training for employees. - Effective drainage system. - Perimeter control blasting incl. pre-split. - Geotechnical drilling and rock testing campaign. - Safety Assessments (Take 5s, JSAs etc.) - Standard Operating Procedures (working under highwalls, traffic , blasting, loading, communications, weather management plans etc.) - Expert design reviews and auditing. - Appropriate equipment for job		Moderate	Rare	L (22)

			interaction between open pit and underground workings		(digger height for bench height, FOPS etc.)				
Operational	**Geotechnical**	Dump Failure	**External Factors:** - Seismic activity. - Climatic events and natural disasters. **Internal Factors:** - Poor design parameters, potentially due to insufficient material understanding. - Poor general operating practices (incl. drainage and water management).	**HSP:** Safety incidents as a result of dump failure. **Environment:** Local ecosystem may be covered. **Financial:** Loss of asset/s. Operational disruptions. High remediation or recovery costs. **LTO:** Increased regulatory scrutiny. Community/ stakeholder engagements around perceived or real unsafe conditions. Reputational loss.	- Slope and seismic monitoring. Regular geotechnical assessments. - Daily inspections. - TARP/management system incl. exclusion zones and hazard maps. - Training for employees. - Effective drainage system. - Geotechnical material testing. - Safety Assessments (Take 5s, JSAs etc.) - Standard Operating Procedures (working on tip heads, traffic, bunding, unloading communications, weather management plans etc.) - Expert design reviews and regular auditing.		Minor	Rare	L (24)

Operational	**UG Geotechnical**	Underground stope or workings geotechnical failure.	**External Factors:** - Seismic activity. - Climatic events and natural disasters. **Internal Factors:** - Poor design parameters, potentially due to insufficient rock mass, structural, void and stress/deformation model understanding. - Mining, blasting, ground support and general operating practices (incl. water management). - Quality of geotechnical monitoring and assessment.	**HSP**: Safety incidents as a result of underground failure. **Financial:** Loss of asset/s. Operational disruptions. High remediation or recovery costs. **LTO:** Increased regulatory scrutiny. Community/ stakeholder engagements around perceived or real unsafe conditions. Reputational loss.	- Geotechnical and seismic monitoring. Regular geotechnical assessments. - Daily inspections. - TARP/management system incl. exclusion zones and hazard maps. - Training for employees. - Effective ground support and ground control systems including QA/QC and bolt/material testing etc. - Effective water management system. - Controlled Blasting - Geotechnical drilling and strength testing campaign. - Safety Assessments (Take 5s, JSAs etc.) - Standard Operating Procedures (drilling, blasting, traffic, ground support, mapping, communications etc.) - Expert design reviews and regular auditing. - Refuge chambers		Moderate	Unlikely	M (18)
Operational	**Hydrology**	Flooding of pit and/ or surrounding areas from surface water run-off.	**External Factors**: - Rainfall, natural disasters, change in weather patterns. - Upstream land use and activities. **Internal Factors**: - Poorly designed or maintained surface water drainage systems . - Inefficient water management plans and/ or warning systems.	**HSP**: Flash flooding, pit in-rush. Incidents arising from concealed (submerged/ underwater) hazards, softened ground or slippery surfaces. **Environment**: Transport of pollutive materials by water to other parts of the local ecosystem. Creation of geotechnical or landform risks through saturation and/ or erosion. **Financial**: Operational disruption during flooding event and cleanup leading to lost revenue. Damage to infrastructure and equipment, causing	- Construction of rainwater/ flood diversion channels in appropriate locations. - Design consideration for impoundments/ TSF to include sufficient freeboard to account for significant storm event based on historical data. - Bund-walls/ drainage established around excavations. - Engineered mine roads to include appropriate crossfall, drainage and culverts. - Environmental and meteorological studies to ensure that expected rainfall patterns are reasonably well understood. - Early warning systems and seasonal cleanup/ preparation activities.		Minor	Rare	L (24)

				production disruption and/ or leading to a requirement for replacements. **LTO**: Displacement of community or disruption to transport systems, potentially aggravated by water management/ flood diversion systems.					
Operational	**Hydrogeology**	Intersecting ground water table unexpectedly presents unmanageable quantity of water in OP or UG	**External Factors**: - Groundwater table. - Climate and natural disasters. **Internal Factors**: - Lack of understanding of hydrogeology. - Lack of contingency plan for water.	**HSP**: Incidents arising from concealed (submerged/ underwater) hazards, softened ground or slippery surfaces. **Environment**: Water gathering in pit or underground sumps or workings is susceptible to contamination (e.g. hydrocarbons) and improperly managed disposal of this water might lead to local ecosystem contamination. **Financial**: Damage to equipment. Operational delays as a result of encountering and needing to mange water. **LTO**: Potential additional reporting requirements.	- Groundwater table surveys/ pump tests and ongoing monitoring. - Development of water management and contingency plans.		Minor	Unlikely	M (21)

Operational	**Open Pit Mining**	Planned open pit production throughput rates not achieved	**External Factors**: - Geopolitical, regulatory or stakeholder issues, incl. industrial action. - Climatic events and/ or natural disasters. **Internal Factors**: - Workforce capabilities. - Availability, Reliability and Utilisation of fleet. - Quality of geological model and planning.	**HSP:** Increase in safety incidents from workers become stressed and/or complacent while trying to meet perceived production demands. **Environment:** Increase in environmental incidents from workers become stressed and cutting corners while trying to meet production demands. **Financial:** Failure to meet production schedule will decrease profitability and asset value. Potential consequences of trying to close later production gaps could lead to financial loss through accident damage. **LTO:** Failure to meet production schedule might impact government revenue, community commitments - potentially leading to grievances.	- Comprehensive training program for operators and multi-skilling to ensure depth of coverage. - Preventative maintenance, condition monitoring and inspection program aligned with industry best practice standards to maximise availability, incl. OEM maintenance support. - Supply chain diversification. OEM Vendor Managed Inventory contracts with appropriate bonuses and penalties built in to ensure continuity of supply. - Advanced grade control drilling program to ensure sufficient understanding of short to medium-term geology, in order to avoid short term variations to plan to meet gold call. - Scenario analysis and calibrated productivity assumptions, combined with advanced scheduling tools and a strong technical services team, to ensure the development of realistic and deliverable short- to medium-term plans. - Robust Permit to Work and lead-indicator focussed safety systems.		Minor	Unlikely	M (21)

Operational	**Open Pit Mining**	Production quality not achieved: spatial compliance to plan; dilution/ ore loss.	**External Factors**: - Geological variability. **Internal Factors**: - Workforce capabilities and performance pressure. - Fleet selection and mining style. - Plan quality. - Operational controls and monitoring technology.	**HSP**: Increase in safety incidents from workers become stressed and complacent while trying to meet production demands/ catch up on plans. **Environment**: Increase in environmental incidents from workers become stressed and cutting corners, particularly regarding housekeeping, while trying to meet production demands. Additional environmental disturbance as a result of plan changes to accommodate unforeseen issues. **Financial**: Reduced profitability as ore loss results in less gold sent to process plan, or higher demand on plant throughput at lower grades due to dilution. **LTO**: Failure to meet production schedule might impact government revenue, community commitments - potentially leading to grievances.	- Experienced technical expertise responsible for mine planning. - Utilisation of technology to improve plan quality, including: high quality equipment utilisation data; drone and other surveys; mine planning and surveying packages; blast movement monitoring; and, fragmentation monitoring systems. - Advanced grade control drilling program to ensure sufficient understanding of short- to medium-term geology, in order to avoid short term variations to plan to meet gold call. - Scenario analysis and calibrated productivity assumptions, combined with advanced scheduling tools and a strong technical services team, to ensure the development of realistic and deliverable short- to medium-term plans.		Minor	Unlikely	M (21)

Operational	**Underground Mining**	Planned underground production throughput rates not achieved	**External Factors**: - Geopolitical, regulatory or stakeholder issues, incl. industrial action. - Climatic events and/ or natural disasters. - Delays to equipment delivery. **Internal Factors**: - Workforce capabilities. - Availability, Reliability and Utilisation of fleet. - Quality of geological model and planning.	**HSP**: Increase in safety incidents from workers become stressed and complacent while trying to meet production demands. **Environment**: Increase in environmental incidents from workers become stressed and cutting corners, particularly regarding housekeeping, while trying to meet production demands. **Financial**: Failure to meet production schedule will decrease profitability and asset value. Potential consequences of trying to close later production gaps could lead to financial loss through accident damage. **LTO**: Failure to meet production schedule might impact government revenue, community commitments - potentially leading to grievances.	- Comprehensive training program for operators and multi-skilling to ensure depth of coverage, including expatriate operator/ trainers during early production phases. - Preventative maintenance, condition monitoring and inspection program aligned with industry best practice standards to maximise availability, incl. OEM maintenance support. - Supply chain diversification. OEM Vendor Managed Inventory contracts with appropriate bonuses and penalties built in to ensure continuity of supply. - Advanced grade control drilling program to ensure sufficient understanding of short- to medium-term geology, in order to avoid short term variations to plan to meet gold call. - Scenario analysis and calibrated productivity assumptions, combined with advanced scheduling tools and a strong technical services team, to ensure the development of realistic and deliverable short- to medium-term plans. - Robust Permit to Work and lead-indicator focussed safety systems.		Minor	Unlikely	M (21)

Operational	**Underground Mining**	Production quality not achieved: spatial compliance to plan; dilution/ ore loss.	**External Factors**: - Geological variability **Internal Factors**: - Drill and blast practices. - Material handling practices. - Paste failure. - Plan quality.	**HSP**: Increase in safety incidents from workers become stressed and complacent while trying to meet production demands/ catch up on plans. **Environment**: Increase in environmental incidents from workers become stressed and cutting corners, particularly regarding housekeeping, while trying to meet production demands. **Financial**: Reduced profitability as ore loss results in less gold sent to process plan, or higher demand on plant throughput at lower grades due to dilution. **LTO**: Failure to meet production schedule might impact government revenue, community commitments - potentially leading to grievances.	- Engage with experts to optimise paste recipe. - Experienced technical expertise responsible for mine planning. - Advanced grade control drilling program to ensure sufficient understanding of short- to medium-term geology, in order to avoid short term variations to plan to meet gold call. - Scenario analysis and calibrated productivity assumptions, combined with advanced scheduling tools and a strong technical services team, to ensure the development of realistic and deliverable short- to medium-term plans.		Minor	Unlikely	M (21)

Operational	**Process**	Production throughput rates not achieved	**External Factors**: - Geological/ geometallurgical variability. - Disruption to supply of power or water. - Natural disasters. **Internal Factors**: - Workforce capabilities. - Availability, reliability and utilisation of process plant. - Quality of planning for material inputs. - Design faults incl. insufficient surge capacity.	**HSP**: Inexperienced operations personnel working around the process plant are typically more likely to be involved in accidents and safety incidents causing LTI and production delays **Environment**: Increase in environmental incidents from workers become stressed and cutting corners, particularly regarding housekeeping, while trying to meet production demands. **Financial**: Failure to meet production schedule will decrease profitability and asset value. **LTO**: Failure to meet production schedule might impact government revenue, community commitments - potentially leading to grievances.	- Adequate metallurgical and comminution testwork to ensure geochemistry and physical/ chemical properties of ore are properly understood, and that the plant is appropriately engineered and designed to manage that various types of ore feed. - Robust operational readiness plan developed during construction phase. Leadership team onboarded before commissioning. - Agreement with power supplier and infrastructure engineered to allow for surge capacity or increased demand. - Comprehensive training program for operators and multi-skilling to ensure depth of coverage, including expatriate operator/ trainers during early production phases. - Preventative maintenance, condition monitoring and inspection program aligned with industry best practice standards to maximise availability, incl. OEM maintenance support. - Supply chain diversification. OEM Vendor Managed Inventory contracts with appropriate bonuses and penalties built in to ensure continuity of supply.		Minor	Unlikely	M (21)

Operational	**Process**	Metallurgical recoveries not achieved.	**External Factors**: - weathering/oxidation of ore on ROM pad due to weather events or delayed operations start - poor quality water affects flotation performance **Internal Factors**: - feed quality variations cause inconsistent plant feed - ROM pad too small to provide sufficient space for blending and campaign treatment - samples for test work not representative of plant feed - inexperienced operations team affects recovery	**HSP**: Inexperienced operations personnel working around the mine are typically more likely to be involved in accidents and safety incidents causing LTI and production delays. **Environment**: Ore weathering on ROM causes acid mine drainage **Financial**: Reduced revenue will affect the profitability of operation **LTO**: Failure to meet concentrate production schedule might impact government revenue, community commitments - potentially leading to grievances.	- Test work program for comminution/flotation/leaching is underway - Perform locked cycle tests to better estimate recovery and reagent consumptions - Generate a geometallurgical model to better estimate recovery in the block model and provide opportunity to maximise recovery in the mining/processing schedule - Perform vendor testwork for thickener and filtration design - Research similar processes including site visits - Conduct testwork with site water - Ensure mine plan takes potential broken ore oxidation into account - Develop a ROM management and blending strategy		Minor	Unlikely	M (21)

Operational	**Workforce Capabilities**	Shortage of skilled labour available/ willing to join the operation.	**External Factors**: - Lack of experienced in-country work force for flotation processing operations. - Competition for small labour pool. **Internal Factors**: - Unattractive workplace due to poor workplace culture/ reputation and/ or limited visibility around career progression. - Uncompetitive remuneration and benefits program. - Ineffective recruitment personnel and policies.	**HSP**: Inexperienced operations personnel working around the mine are typically more likely to be involved in accidents and safety incidents. **Environment**: Inexperienced operations personnel working around the mine are typically more likely to be involved in environmental incidents, leading to potentially higher environmental inspection and compliance burden. **Financial**: Increased cost associated with: upskilling workforce to meet standards; inefficient operational performance, or delays to deliverables, due to inexperience; requirement for additional safety and environmental oversight and compliance; requirement to pay higher salaries to attract appropriately skilled personnel from outside of province, or internationally. **LTO**: Part of the mitigation strategy for this issue will involve onboarding labour from other provinces and countries for training purposes. This could potentially create friction in the community around whether the Company is making sufficient efforts	- Invest in and develop a robust employee training framework, utilising appropriately skilled expert trainers to build workforce capability. - Develop strategy of how to identify the critical basic attributes in inexperienced/ unskilled prospective employees that are indicative of trainability and success, and partner with local universities and communities to identify suitable candidates to onboard and develop. Include graduate and traineeship programs. - Examine opportunities for semi-automation or utilisation of technology to reduce workforce size and complexity. - Ensure labour market is well understood and structure competitive monetary and non-monetary remuneration packages focussed on rewarding performance and reliability of employees. - Early engagement with labour unions to build positive and collaborative relationships to ensure mutually beneficial outcomes for both parties. - ensure labour external to the province is utilised for training and upskilling and communicate this to all stakeholders		Minor	Unlikely	M (21)

				to develop the local workforce if not communicated well..					
Operational	**TSF**	Catastrophic structural failure of tailings impoundment	**External Factors**: - Seismicity. - Weather events, incl. rainfall and natural disasters. **Internal Factors**: - Design flaws. - Quality of design execution, incl. materials and construction. - Monitoring system failure. - Incorrect or inadequate maintenance. - Management of tailings deposition and/ or freeboard.	**HSP**: Potential multiple fatalities through engulfment by liquefied tailings. **Environment**: Habitat destruction, contamination of soil and/ or bodies of water. **Financial**: Substantial fines. Major cost associated with cleaning up spilt tailings, incl. potential requirement to rebuild impoundment. Compensation related to loss of life, environmental damage. Operational delays. **LTO**: Loss of licence to operate. Reputational damage.	- ANCOLD standards (or better) incorporated into design. Best practice FoS with local climatic and seismicity factors accounted for in design. Downstream construction where possible. - Construction QAQC expertise employed to supervise TSF build. Signed off by external EOR. - Ongoing geotechnical and hydrological monitoring. - Robust emergency response plan and training. - As much as is possible, isolate TSF downstream from all other work areas.		Major	Rare	L (19)

Operational	**TSF**	Seepage of contaminants from TSF into groundwater table or local hydrological catchment.	**External Factors**: - Seismicity. - Weather events, incl. rainfall and natural disasters. **Internal Factors**: - Design flaws. - Quality of design execution, incl. materials and construction. - Monitoring system failure. - Incorrect or inadequate maintenance. - Management of tailings deposition and/ or freeboard.	**Environment**: Habitat destruction, contamination of soil and/ or bodies of water including water table. **Financial**: Substantial fines. Major cost associated with cleaning up contamination. or fines for environmental damage. **LTO**: Loss of licence to operate. Reputational damage.	- ANCOLD standards (or better) incorporated into design. Best practice FoS with local climatic factors accounted for in design. Material selection designed to reduce risk of seepage in compliance with relevant Argentine standards. - QAQC expertise employed to supervise TSF build. Signed off by external EOR. - Ongoing hydrological monitoring. - Robust emergency response plan and training.		Major	Rare	L (19)
Environmental	**Biodiversity**	Local biodiversity loss, including vulnerable or endangered species	**External Factors**: - Existing use of land including for community and agricultural use. - Proliferation of invasive species. - Climatic conditions, including change in weather patterns and natural disasters. **Internal Factors**: - Quantum of ground (habitat) disturbance as a result of mine, TSF and waste dump construction. - Noise, light and vibration pollution. - Management of chemicals and the use and disposal of contaminated or untreated water.	**Environment**: Imbalance in the ecosystem, which may or may not be related to loss of key species. **Financial**: Future Provincial or Federal regulation around this topic could result in fines being applied for non-compliance. **LTO**: Reputational damage or community grievance arising from the perception that the operation is harmful to local flora and fauna.	- Biodiversity assessment to baseline local flora and fauna, with continued monitoring throughout operations. - Closure plan to include rehabilitation targets in line with global best practice standards which address habitat management and biodiversity, and are developed in collaboration with local authorities and community groups.		Insignificant	Rare	L (25)

Environmental	**CO_2 Emission Intensity**	Operation becomes a major contributor to CO_2 emissions.	**External Factors**: - Availability and price of low emissions energy sources. - Regulatory environment in Argentina. - Stakeholder expectations. **Internal Factors**: - Primary energy sources. - Efficiency of energy consumption in operations.	**Environment**: CO_2 Emissions have the potential to exacerbate effects of global warming. **Financial**: Availability of credit, or unfavourably adjusted lending criteria, if operation is a perceived or real major CO_2 emitter. **LTO**: Community grievance arising from perception that Hualilan is a major contributor to global emissions. Possible investor backlash if Hualilan not shown to be making an effort to abate emissions.	- Identify and potentially implement technologies or strategies which make relative emission reductions from heavy mining equipment. - Identify and potentially implement green power solutions where possible. - Education of stakeholders of Hualilan's likely position as one of the lowest carbon footprint gold mines worldwide.		Insignificant	Unlikely	L (23)
Environmental	**Water Management**	Consumption of water in operations reduces local supply available for other users (incl. wildlife).	**External Factors**: - Climatic conditions, including rainfall. - Local agricultural or industrial demand for water. - Availability of local water source (ground water or surface water). - Law and Regulation around water use and consumption. **Internal Factors**: - Water consumption volumes and efficiency. - Water recycling and storage practices. - Water source selection	**Environment**: Overuse could deplete groundwater aquifer. Reduction of surface water availability could harm localised flora and fauna. **Financial**: Restrictions on water consumption as a result of a later community grievance could disrupt operations, leading to financial losses. **LTO**: Community grievance arising from a perception that the mine is using excessive water, or is contributing to a unsafe environmental condition due to said consumption, might lead to operational disruptions through protest, or additional regulatory burden that translates to	- Regular engagement with local stakeholders, including communities and regulators, to understand local water needs. - Implementation of water monitoring and reporting systems. - Evaluate new technologies or ways of work which might reduce water consumption from traditional mining and processing activities. - Evaluate options to treat and recover wastewater or tailings water.		Minor	Rare	L (24)

			(groundwater or surface water).	operational disruption or cost increases.					
Environmental	**Air Quality**	Mining operations negatively impact local air quality.	**External Factors**: - Weather and natural disasters. **Internal Factors**: - Land disturbance. - Blasting and mining practices.	**HSP**: High levels of respirable dust present potential health impacts to employees. **Environment**: Reduced visibility. Poor air quality may impact local flora and fauna. **Financial**: Operational disruption due to visibility issues.	- Dust suppression - spray roads, ventilation, spray crusher area, transfer chutes, conveyors etc.		Minor	Unlikely	M (21)
Environmental	**Water Quality**	Mining operations negatively impact local water quality.	**External Factors**: - Weather and natural disasters. **Internal Factors**: - Land disturbance. - Water management incl. run-off. - Chemical usage.	**HSP**: Contaminated water has a potential health impact for employees. **Environment**: Habitat degradation, contamination of soil and/ or bodies of water including water table **Financial:** Substantial fines. Major cost associated with cleaning up contamination. or fines for environmental damage. **LTO:** Loss of licence to	- Hydrological monitoring. - Runoff management system and collection ponds. - Housekeeping of chemicals and contaminants.		Minor	Rare	L (24)

				operate. Reputational damage.					
Environmental	**Noise and Vibration**	Persistent community grievances lodged relating to noise generated by operations.	**External Factors**: - Community proximity. **Internal Factors**: - Timing of certain types of activities - such as road train departures/ arrivals. - Blasting activities.	**HSP:** Potential impact on employee hearing. **Environment:** Disturbance to local fauna. **LTO**: Community grievances arising from a perception that the operation is creating excessive noise outside of reasonable hours. Grievances arising from vibration damage to community structures or infrastructure.	- Minimise the frequency, where controllable, of large transport convoys arriving to/ leaving from site during the evening. - Blasting practices focussed on minimising ground vibrations. - Vibration and noise monitoring. - PPE for employees.		Minor	Rare	L (24)
LTO	**Cultural Heritage**	Archaeological sites of cultural significance prevent ground disturbance.	**External Factors**: - Existence of sites which are recognised as having historical or cultural significance. **Internal Factors**: - Footprint of proposed ground disturbance.	**Financial**: Restrictions on mining ground disturbance or the requirement to relocate sites of cultural significance. **LTO**: Community grievance arising from the disturbance or mismanagement of sites of cultural significance.	- Assessment of planned areas of disturbance using local experts (University of San Juan, et al.) conducted to determine if there are any sites of recognised cultural significance. - Engage with government, community and other relevant stakeholders to develop a management plan for any sites of significance identified. - If necessary, relocate sites and/ or construct facilities of equivalent amenity for community use. - Educate employees about the importance of cultural sites.		Minor	Rare	L (24)

LTO	**Health & Safety**	Asset unjustly develops reputation for being unsafe or not caring about the health and safety of workers.	**External Factors**: - Historical precedents with mining industry. - Media coverage. - Regulatory bodies and standards. **Internal Factors**: - Quality and currency of safety protocols/ management systems. - Workplace culture. - Communication & feedback processes. - Training.	**HSP**: Decrease in morale contributing to an increase in incidents. **Financial**: Higher operational costs associated with implementing additional safety controls. Higher recruitment/ labour costs if available talent pool reduces. Operational disruption through industrial action, government/ regulatory authority interventions. **LTO**: Grievances and industrial action. Suppliers and partners withdraw support.	- Strong, positive safety culture lead by executive and management. - Develop strong safety management system focussed on lead indicators and positive messaging. ISO certification. - Transparent reporting on safety statistics and engagement/ communication/ collaboration with key government and community stakeholders on safety performance. - Training for employees and permit to work system.		Minor	Rare	L (24)
LTO	**Local Employment and Training/ Development**	Community expectations not met in regards to local employment and labour force development.	**External Factors**: - Skilled workforce availability. - Market/ economic/ financial conditions. **Internal Factors**: - Miscommunication/ failure to manage expectations with stakeholders on labour matters. - Mismanagement of relationship with stakeholders. - Lack of a clear recruitment and labour strategy.	**HSP:** Disgruntled employees if jobs outsourced to non-locals **Financial:** Disruption to operations. Intervention of authorities leading to increased operating costs.	- Proactive partnership building with unions and government authorities. - Development and communication of hiring and training strategy. - Communication and dialogue with local communities and leaders. - Prioritisation of local employment.		Moderate	Unlikely	M (18)

Financial	**CAPEX Inflation**	CAPEX for mining equipment, processing facilities and general infrastructure is higher than budgeted/ assumed.	**External Factors**: - Materials cost/ market change. - Currency fluctuations. - Political/ regulatory instability. **Internal Factors**: - Inaccurate cost estimation. - Incorrect selection of equipment or plant. - Design changes, scope creep, lack of contingency.	**Financial**: Project becomes financially unviable. Profitability reduced. May not have sufficient cash to complete capital purchases. **LTO:** Inability to deliver on commitments to community or government due to lack of free cash and budget capacity. Reduced returns to shareholders.	- Development of strategic relationships with suppliers. - Detailed budgeting, using reasonable inputs and allowing for contingency. - Engage experienced project management and planning personnel. - Clear project scopes. - Engage with relevant authorities on import/ export matters.		Moderate	Unlikely	M (18)
Financial	**OPEX Inflation**	OPEX for mining or processing is higher than budgeted/ assumed.	**External Factors**: - Materials cost/ market change. - Currency fluctuations. - Political instability. **Internal Factors**: - Inaccurate budgeting. - Operational inefficiencies. - Supply chain inefficiencies.	**Financial**: Profitability reduced. Cashflow challenges. **LTO:** Inability to deliver on commitments to community or government due to lack of free cash and budget capacity. Reduced returns to shareholders.	- Development of strategic relationships with suppliers. - Detailed budgeting, using reasonable inputs and allowing for contingency. - Engage experienced operational management and technical personnel. - Development of continuous improvement and cost focussed operational culture. - Engage with relevant authorities on import/ export matters.		Moderate	Unlikely	M (18)
Financial	**Currency & Exchange**	Fluctuations in value of the Argentine Peso against other currencies outside of expectations.	**External Factors**: - Inflation. - Government legislation and monetary decisions. **Internal Factors**: - Ineffective use of hedging/ financial instruments	**Financial**: Increased costs, reduced profitability. Operation may become financially unviable. **LTO**: Inability to deliver on commitments to community or government due to lack of free cash and budget capacity. Reduced returns to shareholders.	- Target all costs in lowest quartile - Funds in USD - Use of BCS		Moderate	Possible	H (13)

Financial	**Access to Capital**	Unable to access capital due to economic uncertainty or investor concerns about the stability of the country's financial system.	**External Factors**: - Market conditions, Government legislation and monetary decisions **Internal Factors**: - Engagement of inappropriate and advisors and executives	**Financial**: Unable to fund construction or operation Project is cancelled. **LTO**: Cannot deliver returns for shareholders. Stakeholder commitments not met.	Hedging policies to guarantee costs are covered to reduce risk of having to access capital		Major	Unlikely	M (14)
Financial	**Commodity Prices**	Actual commodity sale prices are lower than those assumed for operational and financial models.	**External Factors**: - Shifts in supply and demand. - Major geopolitical and economic events. **Internal Factors**: - Inaccurate or aggressive forecasting, lack of market intelligence. - Hedging strategy. - Production delays or issues affecting timing of finished goods for sale.	**Financial**: Reduced profitability. Issues with liquidity and reduced cashflow. Requirement to repay debt at a different rate, or change in interest rates, as a result of commodity price reduction or subsequent flow-on effects. LTO: Inability to deliver on commitments to community or government due to lack of free cash and budget capacity. Reduced returns to shareholders.	- Develop mining and processing plan with conservative assumptions for commodity price, in line with industry best practice, without excessively restricting ability to access upside potential of commodity price. - Hedging strategy. - Debt structuring and conditions.		Minor	Possible	M (17)
Financial	**Concentrate Payabilities**	Payabilities for concentrates are lower that those assumed for operational and financial models.	**External Factors**: - Market conditions **Internal Factors**: - limited to one customer, no spot market sales	**Financial**: sub-optimal receipts, financial losses	- Multiple customer agreements - Quality production - Ability in agreements to maximise upturns in market conditions		Moderate	Possible	H (13)
Financial	**Financing Costs**	Interest rates and/ or borrowing costs increase relative to assumptions.	**External Factors**: - Market and economic conditions. - Monetary policy and banking regulations. **Internal Factors**: - Internal financing	**Financial**: Reduced profitability. Issues with liquidity and reduced cashflow. **LTO**: Inability to deliver on commitments to community or government due to lack of free cash and budget	- Structured project and general financing with consideration given to financing cost risks. - Hedging strategy. - Accurate and frequently revised budgeting and forecasts. - Development of strong organisational cost control culture.		Minor	Unlikely	M (21)

			structure. - Budgeting, cash and cost control,	capacity. Reduced returns to shareholders.					
Financial	**Repatriation of Dividends**	Cannot repatriate profits generated by asset to shareholders as dividends due to government fiscal controls.	**External Factors**: - Economic conditions. - Political climate/ public opinion **Internal Factors**: - Failure to negotiate appropriate fiscal stability agreement. - Quality of relationship and engagement with authorities and stakeholders.	**Financial**: Losses, reduced sales, inability to continue operations. **LTO**: Cannot deliver returns to shareholders.	- Fiscal stability agreement. - Partnership building with stakeholders and authorities. - temporary law allowing 20% of project revenue (~ 40% of 57% of cashflow) to be retained offshore		Major	Possible	H (9)
Financial	**Import Duties**	Unable to import critical goods for production, or there is an additional unplanned cost to import goods.	**External Factors**: - Economic conditions. - Political climate/ public opinion **Internal Factors**: - Failure to negotiate appropriate fiscal stability agreement. - Quality of relationship and engagement with authorities and stakeholders.	**Financial**: Increased costs, reduced profitability. Operation may become financially unviable. **LTO**: Inability to deliver on commitments to community or government due to lack of free cash and budget capacity. Reduced returns to shareholders.	- Fiscal stability agreement. - Partnership building with stakeholders and authorities.		Minor	Unlikely	M (21)

Financial	**Export Duties**	Cannot export finished goods, or there is an additional unplanned duty on exported goods.	**External Factors**: - Economic conditions. - Political climate/ public opinion **Internal Factors**: - Failure to negotiate appropriate fiscal stability agreement. - Quality of relationship and engagement with authorities and stakeholders.	**Financial**: Increased costs, reduced profitability. Operation may become financially unviable. **LTO**: Inability to deliver on commitments to community or government due to lack of free cash and budget capacity. Reduced returns to shareholders.	- Fiscal stability agreement. - Partnership building with stakeholders and authorities.		Minor	Unlikely	M (21)
Financial	**Royalties**	Increase in royalties paid to province due to increase in rate or change of calculation methodology.	**External Factors**: - Economic conditions. - Political climate/ public opinion **Internal Factors**: - Failure to negotiate appropriate fiscal stability agreement. - Quality of relationship and engagement with authorities and stakeholders.	**Financial**: Increased costs, reduced profitability. Operation may become financially unviable. **LTO**: Inability to deliver on commitments to community or government due to lack of free cash and budget capacity. Reduced returns to shareholders.	- Fiscal stability agreement. - Partnership building with stakeholders and authorities.		Minor	Unlikely	M (21)
Financial	**Corporate Tax and Other**	Increase in corporate or other (i.e. payroll) taxes.	**External Factors**: - Economic conditions. - Political climate/ public opinion **Internal Factors**: - Failure to negotiate appropriate fiscal stability agreement. - Quality of relationship and engagement with authorities and stakeholders.	**Financial**: Increased costs, reduced profitability. Operation may become financially unviable. **LTO**: Inability to deliver on commitments to community or government due to lack of free cash and budget capacity. Reduced returns to shareholders.	- Fiscal stability agreement. - Partnership building with stakeholders and authorities.		Minor	Unlikely	M (21)

Country	**Sovereign**	Argentina defaults on debt obligations.	**External Factors**: - Political instability. - Global economic/ market conditions. - Inflation. - Exchange rates and controls. **Internal Factors:** - Insurance coverage.	**Financial**: Capital controls limiting movement of money out of the country leading to supply chain issues and restrictions on shareholder dividends. Changes to taxation program to generate more revenue for government. Currency devaluation impacting value of assets. Credit crunch and liquidity crisis. Nationalisation.	- Government engagement and partnership building. - Careful management of working capital and finished goods to reduce impact of currency devaluation on cash. - Fiscal stability agreements with relevant authorities. - Insurance.		Moderate	Possible	H (13)
Country	**Infrastructure - Power**	Power grid/ infrastructure unable to support electrical power demand.	**External Factors**: - Supply constraints. - Infrastructure quality and maintenance. **Internal Factors**: - Failure to correctly forecast or manage power demand.	**HSP**: Power outages potentially expose workers to safety hazards. **Financial**: Operational downtime affecting productivity and revenue. **LTO**: Community grievances if mine power draw leads to brownouts/ blackouts in local communities.	- Detailed planning and assessment of power needs by competent and experienced engineering personnel. - Installation of backup energy sources. - Regular inspection and maintenance of infrastructure. - Utilisation of energy efficient technologies. - Engagement with government authorities and communities.		Minor	Unlikely	M (21)
Country	**Infrastructure - Transport**	National highway and roads servicing Hualilan degrade due to lack of maintenance and/ or excessive use/ overloading.	**External Factors**: - Government budget constraints and priorities. - Natural disasters. - Traffic volumes.	**HSP**: Increased vehicular accidents as roads deteriorate. **Environment**: Erosion. Noise or vibration contribute to biodiversity loss. **Financial**: Damage to equipment being transported. **LTO**: Community grievances arising due to noise and vibration, as well as heightened risk to general community safety as a result of deteriorating road quality.	- Program of controls on people and goods movement between San Juan and Hualilan e.g. employees must travel on bus, specified times for transports, fitness-for-work requirements for drivers.		Moderate	Unlikely	M (18)

Country	**Corruption.** The Company notes Argentina is ranked 94th of 180 on Transparency Internationals Corruptions Perception Index with the risk assessment completed in line with this ranking. Based on their collective in country experiences the Company/Directors/Officers believe that the risk posed by corruption in Argentina and San Juan is far below this ranking.	Management/ Executive are requested to pay bribes to ensure ongoing operations, and/ or employees engage in behaviour designed to elicit personal kick-backs.	**External Factors**: - Economic conditions and political climate. - Level of entrenchment of corrupt practices in the province/ nation. - Weak rule of law in relation to corrupt conduct. **Internal Factors**: - Corporate standards and culture around ethics and transparency. - Vetting and education of employees. - Protections for whistle-blowers and systems in place to facilitate reporting of unethical behaviour. - Quality of internal controls and audit.	**Financial**: - Operational impacts and delays as a result of not paying requested bribes. - Financial loss to company, in the form of acquiring inferior value goods and services, as a result of supply and procurement processes which are corrupted by the employees responsible for their execution. - Fines arising from regulators (incl. foreign regulators) for corrupt conduct. **LTO**: - Community grievance or protest as a result of the company being perceived to be engaged in corrupt practices. - Community grievance or protest as a result of corrupt decision making by the company on selection of suppliers of goods and services. - Damage to relations with community, government and/ or individual actors as a result of resisting or rejecting corruption. - Prosecution of Company, Directors and/ or Officers under the Foreign Corrupt Practices Act (US) or Criminal Code Act 1995 (Australia), - Resulting reputational damage to all.	- Robust anti-corruption policies and regular training for employees, potentially including ISO certification. - Implement whistleblower program. - Develop clear and transparent systems and processes around selection of suppliers for goods and services. - Develop clear systems and processes around community and government engagement and negotiation to ensure transparency and protection for employees and management. - Implement an internal audit function, and have systems in place to deal quickly and transparently with bribery or corruption allegations. - Carry out appropriate due diligence on new hires and ensure remuneration structures do not inadvertently encourage corruption.		Minor	Unlikely	M (21)

26.3 Opportunities

The company has identified several clear and material opportunities for improvement of the SS outcome which will be evaluated in the next stage of studies. These include:

- A low-grade zinc ore enrichment pathway, based on a recent flotation test on a composite grading 0.36% Zn which produced a saleable Zn concentrate grading 48% Zn. Based on prior flotation test work, an assumption was used in the Study that an economic zinc concentrate was only achievable from at a grade >1.5% Zn. The MRE contains approximately 267,000t of zinc of which only 70,000t is recovered in the Scoping Study mine plan. The ability to economically recover part of the additional 197,000t of zinc in the MRE could significantly enhance economics, given the recovered portion of the ~70,000t of zinc generates US$132m revenue based on the Study forecasts.
- Further improvement to the underground stope optimisation, development sequence and production scheduling. The underground stope optimisation was undertaken using an assumption of US$1700/oz gold. Additionally, some improvements in production and development unit costs in the order of 10-20% have already been identified in the intervening period since running the stope optimisation. These Improvements in production and development costs are yet to be incorporated into the optimisation, and are likely to result in additional stopes being included in the mine plan. Additionally, optimisation included a Pseudoflow analysis on the underground design to remove uneconomic areas that sit above the stope cut-off grade. Pseudoflow removed 832kt containing 72,000 oz AuEq[3] from the underground mine plan that may be profitable at current spot prices and revised operating costs.
- The improvement in underground optimisation includes reviewing the staging of development during the pre-production period to optimise CAPEX while trading off against ensuring access to the highest value stopes in early phases of the UG mine.
- Recovery of the 30-metre crown pillar which has been left between the base of the open pits and the underground workings. This crown pillar contains approximately 15,000 Oz AuEq[3]. The study currently assumes no recovery of this crown pillar, however additional geotechnical information may support the recovery of this crown pillar.
- Inclusion of a heap leaching option which provides a process path for a significant proportion (~50%) of the MRE that was excluded in the high-grade/ low-tonnage SS production model. Preliminary column testing on a low-grade composite yielded promising results. As a result of this, a panel of column tests were initiated to test the three material types separately at a range of different head grades. Results from this current panel of column tests will not be available until December 2023, but a positive outcome has potential to add significant value to the project.

- Reduction in open pit mining unit cost through owner-operator and bulk mining efficiencies. A unit cost of US$3.00/t was assumed for the Study, initially as a conservative estimate based on the predicted reduced scale of the open pit operation, and later to account for contractor premiums. However, preliminary first-principles cost modelling by the Company, and discussions with equipment vendors around collaboration and operating partnerships, indicates that an owner operated unit cost around US$2.00/t may be achievable at scale. This impact of a reduced mining unit cost is even more pronounced in a high-volume mining scenario that incorporates a low-grade heap leach. This cost estimate is supported by localised benchmarking at other owner-operator OP mines in Argentina.
- Potential processing of the Au-Ag concentrate on site to produce gold and silver dore. The project is forecast to produce 371kt of Au-Ag concentrate containing 634Koz Au and 1.9Moz Ag over the Life of Mine. The treatment of this concentrate on site to produce gold and silver dore rather than its transport and sale to off-takers as a concentrate could result in combined cost savings and additional revenue net to the project of over A$200 million based on the SS production forecast.

27 CONCLUSIONS AND NEXT STEPS

The study provides justification that the Hualilan project is commercially viable and accordingly, the Board of the Company has approved progression of the project to the next stage of studies.

Next steps to add to the robustness of the current project and provide a pathway for future development for the project include:

- Taking receipt of the final results for the suite of Column Leach tests currently underway, which will allow for an assessment of the viability of Heap Leach as a potential processing pathway for the low-grade mineralisation;
- Completion of additional flotation testing on the potential low-grade zinc enrichment pathway;
- Completion of additional flotation testing, including locked-cycle and variability test work, which will be required to provide sufficient data for the PFS;
- Testing to determine the liberation of the gold and silver in Au-Ag concentrate and evaluate options to produce dore on site from the Au-Ag concentrate;
- Development of a detailed first-principles open pit mining cost model, in collaboration with equipment vendors, to evaluate the potential owner operated bulk mining efficiencies;
- Completion of a suite of CIL test work (with dual-laboratory verification) to allow Au and Ag recoveries and NACN consumption to be modelled for both the high-grade and low-grade mineralisation, thereby allowing for a definitive evaluation of the CIL processing option;
- Update the first-principle cost models for the processing and general and administrative areas such that they can be utilised to assess the cost impact of variable process throughputs;
- Update the processing cost model to be inclusive of heap leaching, should the Column Test results be positive;
- Complete geotechnical data gathering, including: additional core logging; collection of Point Load Test data from existing drill core; gathering of televiewer data from existing drill holes; and, any drilling of additional geotechnical test holes;
- Updating the underground stope optimisation for final underground mining and development cost forecasts;
- Further optimisation of the open pit/ underground interface and which components of the orebody should be included in each; and,
- Additional drilling of some of the drill targets identified in the Hualilan regional exploration programme.

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APPENDICES

Appendix 1 – Price Waterhouse Coopers Report on Mining Tax

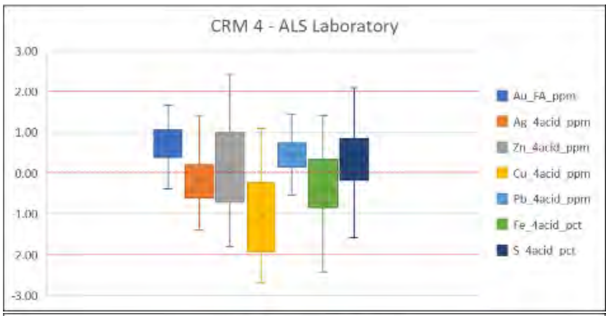
Appendix 1 - PWC Report on Mining Tax.pdf

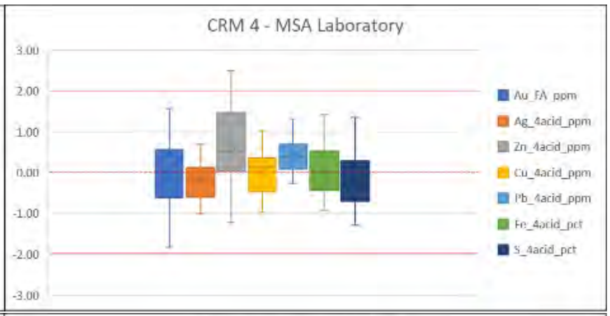
Appendix 1 - PWC
Report on Mining Tax

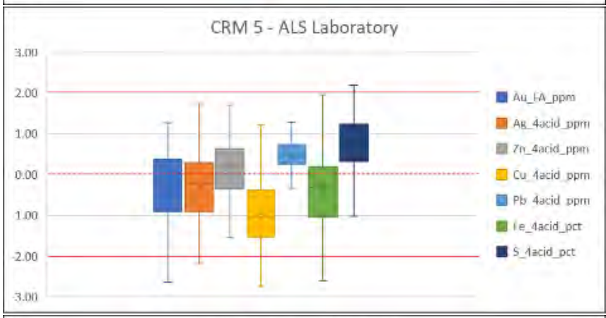
Appendix 2 – QAQC Charts

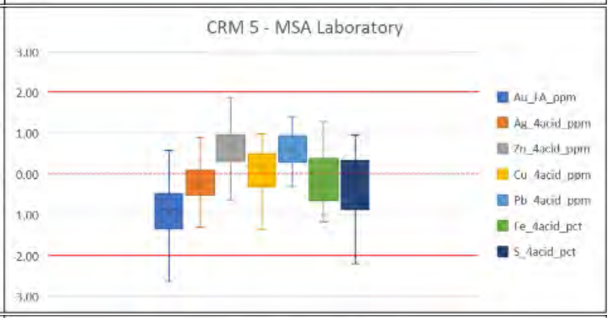
OREAS sourced CRM analysed at ALS, MSA and SGS Laboratories

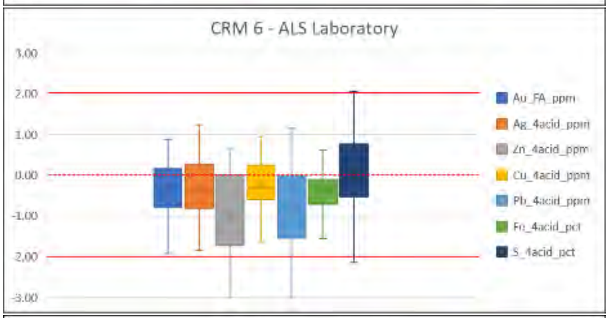
Box and whisker plots for CRM sourced from OREAS, inserted into sample sequences of drill core, RC and channel samples and analysed at ALS, MSA and SGS laboratories.

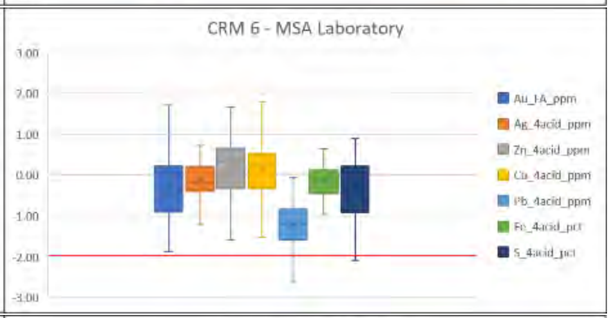


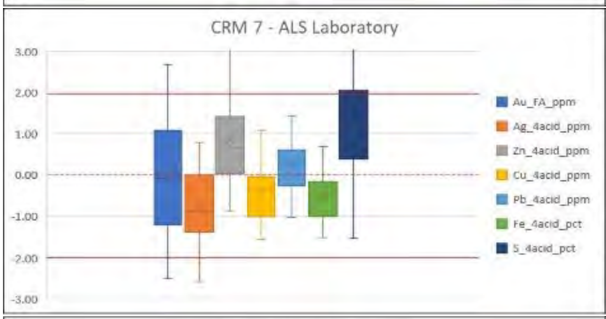


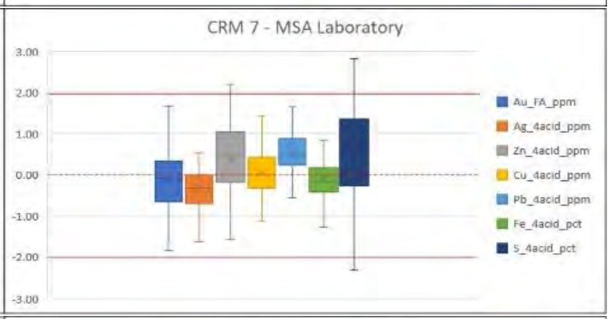


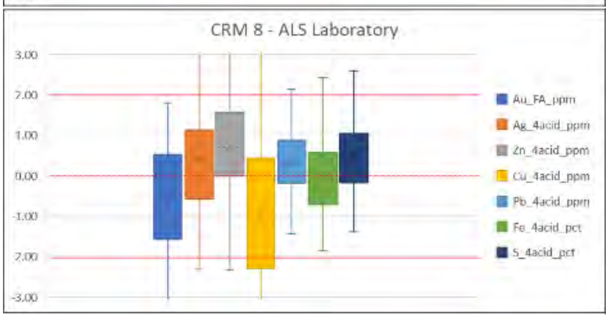


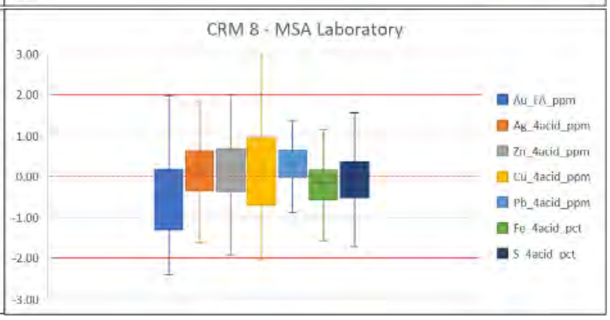


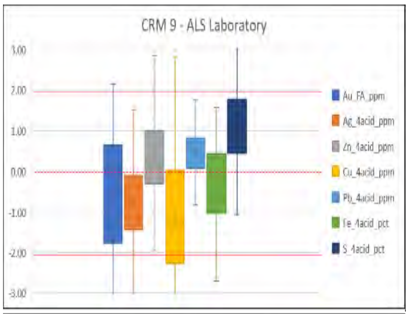








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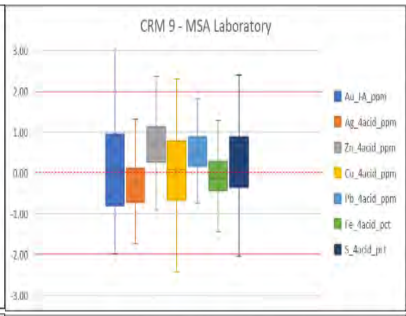
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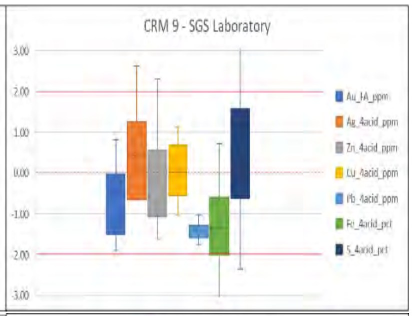
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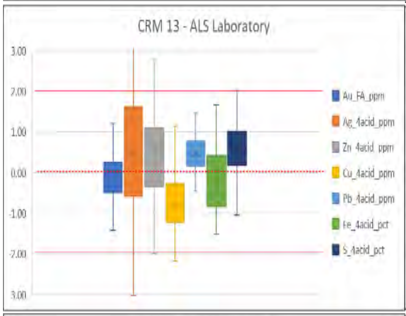
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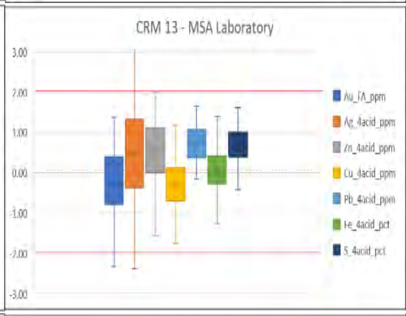
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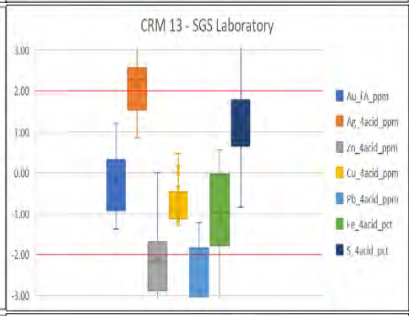
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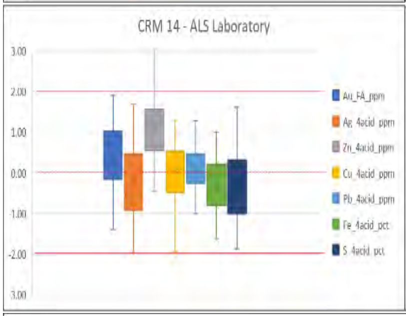
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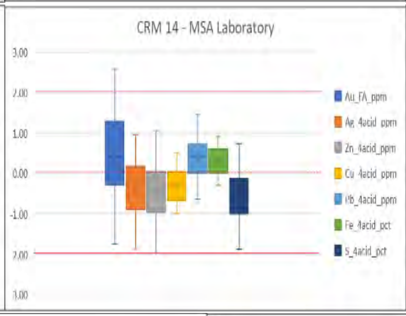
Fe_4acid_pct
S_4acid_pct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1.00
0.00
-1.00
-2.00
-3.00
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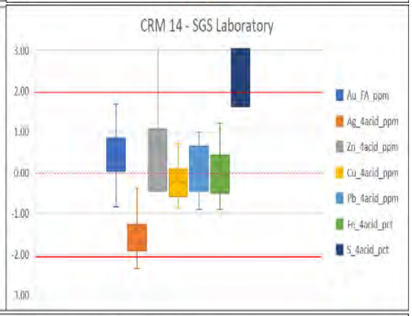
Zn_4acid_ppm
Cu_4acid_ppm
Pb_4acid_ppm
Fe_4acid_pct
S_4acid_pct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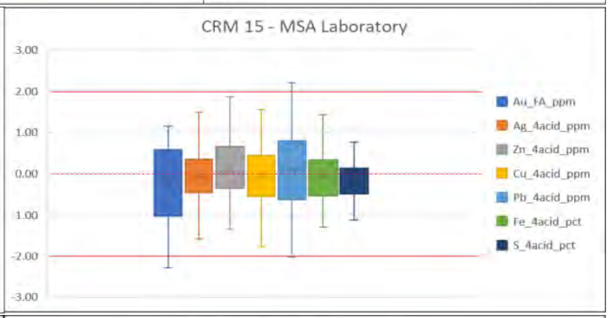
Pb_4acid_ppm
Fe_4acid_pct
S_4acid_pct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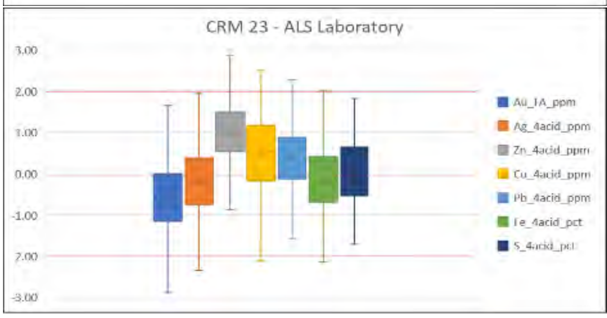
Zn_4acid_ppm
Cu_4acid_ppm
Pb_4acid_ppm
Fe_4acid_pct
S_4acid_pct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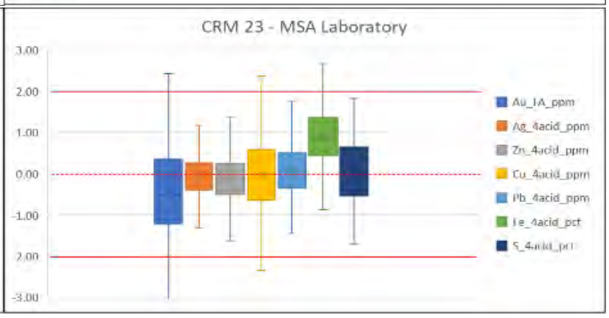
Cu_4acid_ppm
Pb_4acid_ppm
Fe_4acid_pct
S_4acid_pct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-2.00
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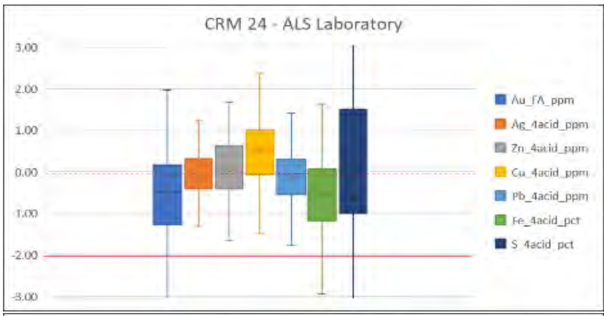
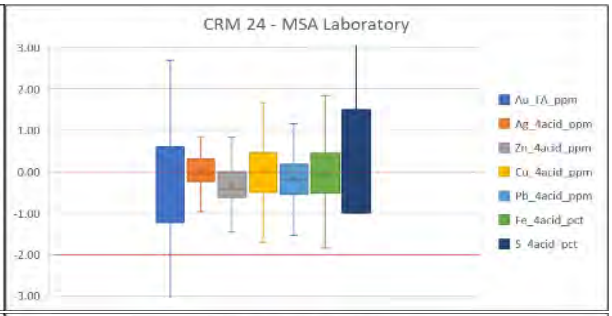
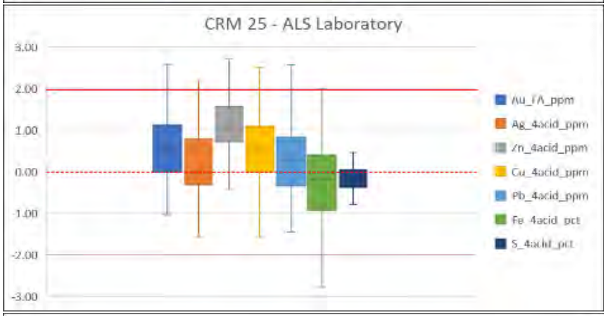
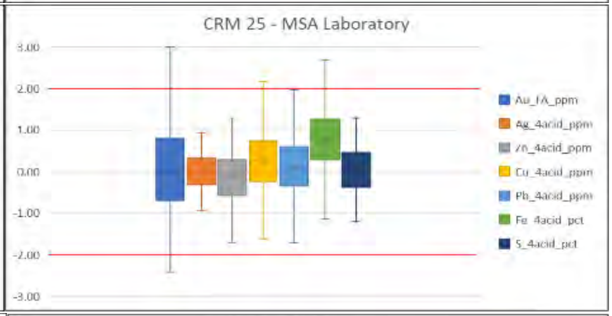
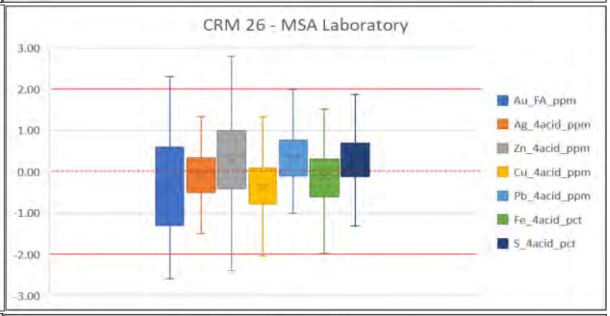
Cu_4acid_ppm
Pb_4acid_ppm
Fe_4acid_pct
S_4acid_pct

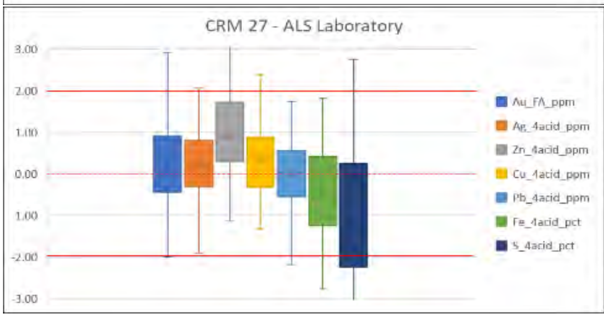


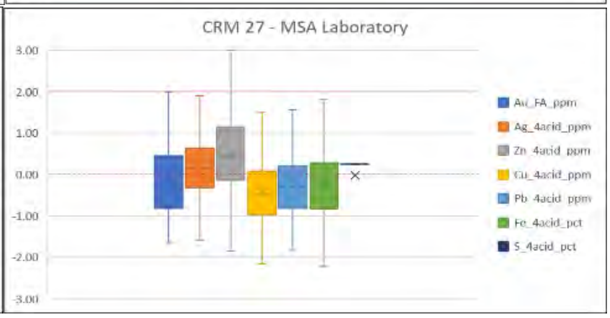






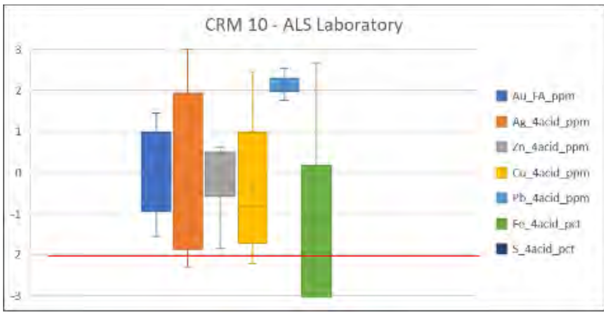


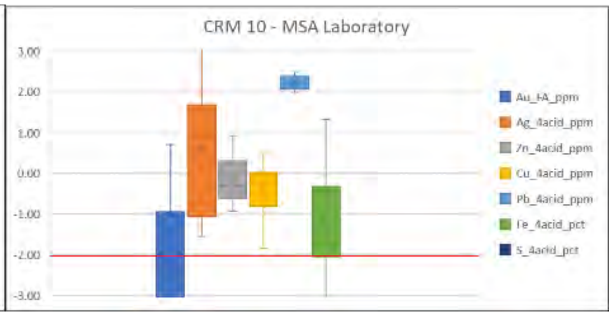


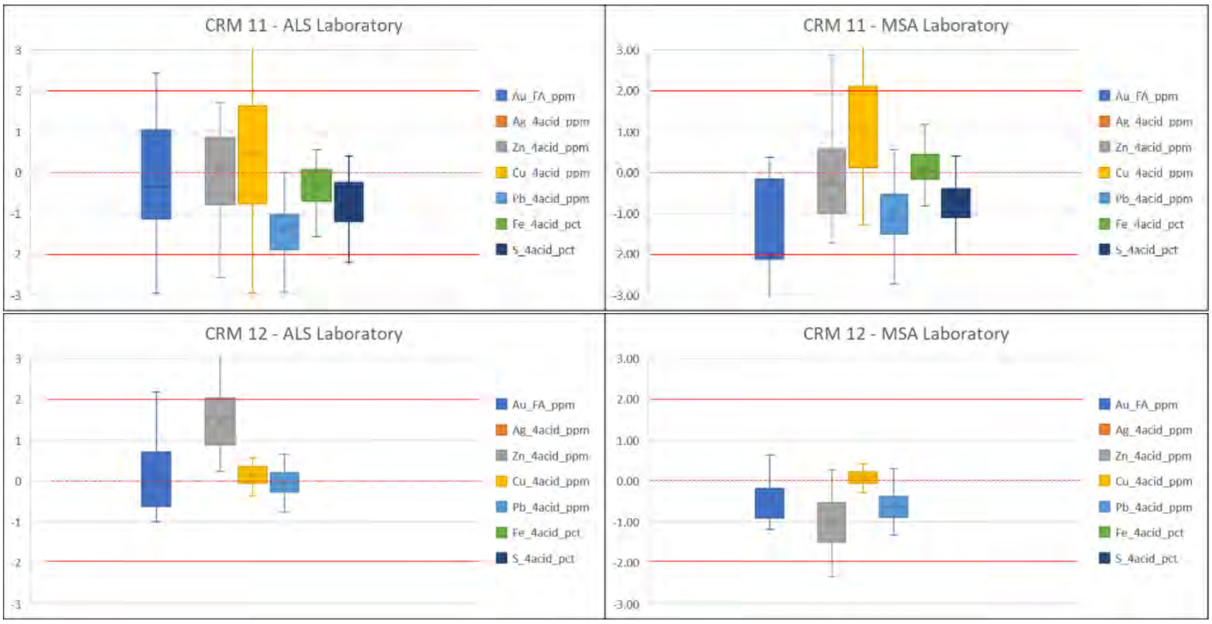


INTEM sourced CRM analysed at ALS and MSA laboratories

Box and whisker plots for CRM sourced from INTEM, inserted into sample sequences of drill core, RC and channel samples and analysed at ALS, MSA laboratories.

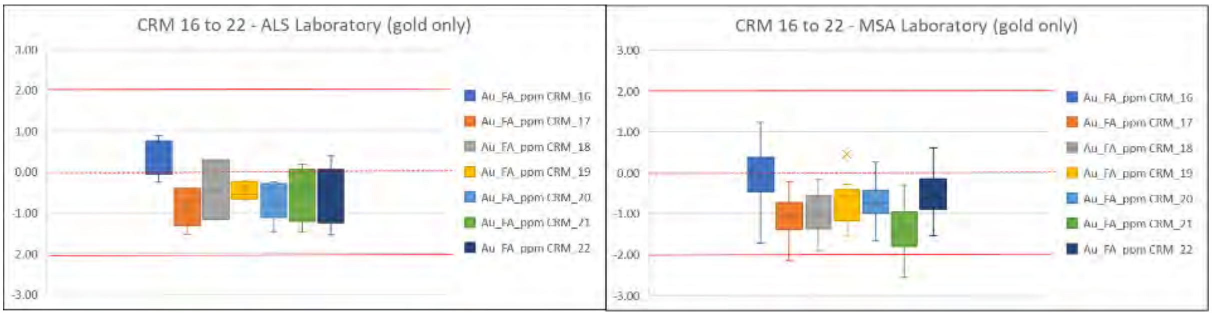






GeoStats sourced CRM analysed at ALS and MSA laboratories

Box and whisker plots for CRM sourced from GeoStats (gold only), inserted into sample sequences of drill core, RC and channel samples and analysed at ALS, MSA laboratories.



Appendix 3 – Define the Process

Appendix 3 – Define the Process – Trade off 1 – Metallurgical process and mining.pdf

Appendix 3 - Define the Process - Trade o

Appendix 4 – OP and UG Combined

Appendix 4 – OP & UG Combined – Trade-off 2 – 16 cases -Pit size and UG COG.pdf

Appendix 4 - OP &
UG Combined - Trade

Appendix 5 – Hydrochemistry

The project area and its zone of influence is characterised by the scarcity of permanent watercourses. The existing river courses are of a temporary nature, with limited activity in the summer season during the development of local rains and storms. This situation makes the hydrochemical characterisation of surface water within the limits of the mining property difficult, and for this reason the sampling plan has been concentrated on slopes at the foot of small mountain ranges approximately 10 km south of the project.

In the region, there is historical monitoring of the La Ciénaga slope, carried out by CRAS during the month of April 1974 (Furque, 1983). The results of this study have been incorporated into the analysis of the hydrochemical information of the Hualilan Project.

First Sampling was conducted in 2007. Challenger carried out a second program in 2020 and from September 2022 to the present it has implemented a monthly monitoring program with 7 active sites (spring 1, slope 2, slope 3, slope 4, slope 5A, slope 5B and well). The sites "slope 5A and 5B" began the sampling record in the year 2020 (Figure 27-1). In total, there are 39 hydrochemical analytical results, which are shown in Table 27-2.

During the month of February 2023, the "PA1" well was built for hydrogeological purposes within the mining property. From the pumping test carried out, data on the pH and conductivity of groundwater at a depth of 37.72 metres have been obtained, which has been incorporated in the analysis.

Although there is not a sufficiently long sampling period in which to analyse seasonal variations over the years and thus establish a solid baseline, these results allow at least identifying trends in natural water quality in the environment.

Classification of water types

From the data of field and laboratory physico-chemical parameters, the type of water present in the region was interpreted. Considering the type of source, the spatial location and depending on the distribution and participation of the majority anions and cations, three differentiated groups of water types can be distinguished: (1) springs 1 to 4, (2) springs 5, 5A and 5B, and (3) well. Table 27-1 shows the amount of data per parameter per group together with the minimum, maximum and average values. For the purposes of statistical calculations, in the case of results that fall below the laboratory's quantification limit, half of the value has been used to carry them out. For the visualization and classification of the types of water, Piper and Stiff diagrams have been used.

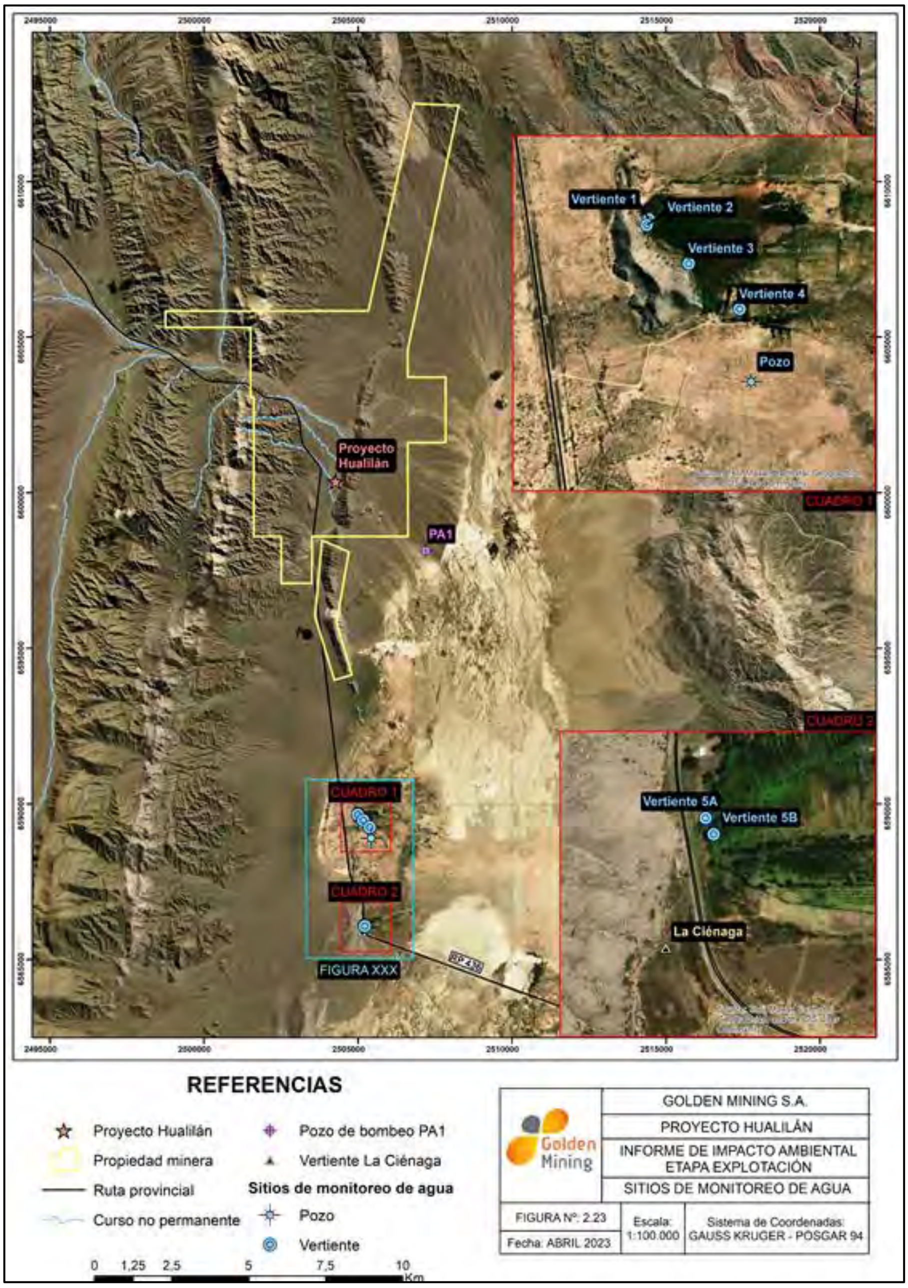


Figure 27-1: Well sampling sites

Table 27-1: Sites and dates of the Hualilan Water Sampling Program

Site	Coordinates Gauss-Krüger		Samples	Dates						
	P94-F2 X	P94-F2 Y		2007	2020	9--2022	10--2022	11--2022	12--2022	1--2023
Spring 1	2505002	6589678,81	2							
Spring 2	2504993	6589651,8	7							
Spring 3	2505169,07	6589465,73	6							
Spring 4	2505382,15	6589247,64	7							
Spring 5	2505242,46	6586221,34	1							
Spring 5A	2505210,08	6586098,38	5							
Spring 5B	2505223,09	6586072,37	5							
Well (Pozo)	2505431,17	6588899,5	6							
Well PA1	2507200,68	6598142,13	1							

Springs 1 to 4 are characterised by being neutral waters, with an average pH of 7.5 measured in the field and 7.9 in the laboratory. It constitutes the group of sampling sites with the highest conductivity values in the region, giving the water an average salinity, with an average value of 828 uS/cm and a maximum of up to 1210 uS/cm. The average dissolved oxygen is 5.5 mg/l, which guarantees the levels necessary for the development of aquatic life.

The type of water is calcium sodium sulphate (Figure 27-2). Although there is a considerable participation of bicarbonate and chloride anions, the dominant one is sulphate. The relationship for this type of water is: SO4H2- > HCO3- > Cl- and Na ++ K+ ≥ Ca2+ >> Mg2+. The average concentration of sulphate is 214 mg/l with a range between 148 and 470 mg/l. These values are the highest recorded in the area. Alkalinity is controlled by the bicarbonate ion in all cases, with the participation of carbonate being negligible. For slopes 1 to 4, the bicarbonate average is 98 mg/l.

Table 27-2: Basic Statistics for the three differentiated water groups in the area

Niveles guía Ley 24.585			pH in situ	pH a 25°C	SDT in situ (mg/l)	SDT (mg/l)	OD in situ (mg/l)	Nitrito (mg/l)	Nitrato (mg/l)	Fluoruro (mg/l)	Al total (mg/l)	Sb total (mg/l)	As total (mg/l)	Ba total (mg/l)	Be total (mg/l)	B total (mg/l)	Cd total (mg/l)
Agua para bebida humana			6,5-8,5	6,5-8,5	1000	1000	5	1	10	1,5	0,2	0,01	0,05	1	0,000039		0,005
Agua para Irrigación			6,5-8,5	6,5-8,5	1000	1000	5			1	5		0,1			0,5	0,01
Agua para bebida de ganado			6,5-8,5	6,5-8,5	1000	1000	5			1	5		0,5		0,1	5	0,02
Locacion	**ID Muestra**	**Fecha**															
La Ciénega	-	17/4/1974	7,7	-	-	-	-	-	2,3	0,2	-	-	-	-	-	0,56	-
Vertiente 2	5472-P-2007	2007	-	8,2	-	409	-	0	-	0,1	-	-	-	-	-	-	-
Vertiente 4	5472-P-2007	2007	-	9,1	-	368	-	0	-	0,1	-	-	-	-	-	-	-
Vertiente 1	319-P-2020	2020	-	8,1	-	470	-	0	-	0,1	-	-	-	-	-	-	-
Vertiente 2	319-P-2020	2020	-	8,2	-	512	-	0	-	0,2	-	-	-	-	-	-	-
Vertiente 3	319-P-2020	2020	-	8,4	-	524	-	0	-	0,15	-	-	-	-	-	-	-
Vertiente 4	319-P-2020	2020	-	8,1	-	463	-	0	-	0,2	-	-	-	-	-	-	-
Vertiente 5	319-P-2020	2020	-	8,6	-	452	-	0	-	0,2	-	-	-	-	-	-	-
Pozo	319-P-2020	2020	-	8,1	-	125	-	0	-	0,14	-	-	-	-	-	-	-
Vertiente 1	Hualil-00001	12/9/2022	-	7,9	-	562	-	<0,04	7	<0,5	<0,01	<0,01	<0,01	0,021	<0,01	0,23	<0,01
Vertiente 2	Hualil-00002	12/9/2022	-	7,9	-	724	-	<0,04	11	<0,5	0,036	<0,01	<0,01	0,017	<0,01	0,35	<0,01
Vertiente 3	Hualil-00003	12/9/2022	-	7,7	-	652	-	<0,04	15	<0,5	0,898	<0,01	<0,01	0,024	<0,01	0,46	<0,01
Vertiente 4	Hualil-00004	12/9/2022	-	7,7	-	438	-	<0,04	7	<0,5	0,452	0,014	<0,01	0,025	<0,01	0,2	<0,01
Pozo	Hualil-00005	12/9/2022	-	8,7	-	298	-	<0,04	<2	<0,5	0,019	0,01	<0,01	0,02	<0,01	0,16	<0,01
Vertiente 5A	Hualil-00006	12/9/2022	-	7,8	-	340	-	<0,04	9	<0,5	<0,01	<0,01	<0,01	0,027	<0,01	0,19	<0,01
Vertiente 5B	Hualil-00007	12/9/2022	-	7,9	-	336	-	<0,04	9	<0,5	0,541	<0,01	<0,01	0,035	<0,01	0,21	<0,01
Vertiente 2	Hualil-00008	26/10/2022	8,1	7,7	-	736	6,9	0,96	10	<0,5	0,062	<0,01	<0,01	0,021	<0,01	0,49	<0,01
Vertiente 3	Hualil-00009	26/10/2022	7,7	7,6	-	506	3,9	0,05	6	<0,5	0,604	<0,01	<0,01	0,04	<0,01	0,29	<0,01
Vertiente 4	Hualil-00010	26/10/2022	7,8	7,9	-	492	4,4	<0,04	7	<0,5	0,285	<0,01	<0,01	0,043	<0,01	0,3	<0,01
Pozo	Hualil-00012	26/10/2022	9,1	8,6	-	334	3,4	<0,04	<2	<0,5	0,207	<0,01	<0,01	0,017	<0,01	0,24	<0,01
Vertiente 5A	Hualil-00013	26/10/2022	7,8	7,9	-	382	6,6	<0,04	9	<0,5	0,042	0,019	<0,01	0,029	<0,01	0,24	<0,01
Vertiente 5B	Hualil-00014	26/10/2022	7,8	7,9	-	384	6,6	<0,04	9	<0,5	0,017	0,012	<0,01	0,028	<0,01	0,23	<0,01
Vertiente 2	Hualil-00016	18/11/2022	7	7,7	567	810	7,2	0,09	10	<0,5	1,25	<0,01	<0,01	0,06	<0,01	0,63	<0,01
Vertiente 3	Hualil-00017	18/11/2022	7,1	7,7	384	540	4,2	0,19	4	<0,5	0,273	<0,01	<0,01	0,09	<0,01	0,33	<0,01
Vertiente 4	Hualil-00018	18/11/2022	7,5	7,9	380	450	3,9	<0,04	7	<0,5	0,113	<0,01	<0,01	0,025	<0,01	0,36	<0,01
Pozo	Hualil-00020	18/11/2022	8,4	8,6	261	300	2,1	<0,04	<2	<0,5	0,156	<0,01	<0,01	0,018	<0,01	0,27	<0,01
Vertiente 5A	Hualil-00021	18/11/2022	7,8	7,9	307	390	6,4	<0,04	9	<0,5	0,029	<0,01	<0,01	0,032	<0,01	0,3	<0,01
Vertiente 5B	Hualil-00022	18/11/2022	7	8,1	298	380	9,2	<0,04	8	<0,5	1,223	<0,01	<0,01	0,048	<0,01	0,33	<0,01
Vertiente 2	Hualil-00024	2/12/2022	6,7	7,8	559	768	6,7	<0,04	10	<0,5	0,039	0,015	<0,01	0,023	<0,01	0,54	<0,01
Vertiente 3	Hualil-00026	2/12/2022	7,2	7,6	378	496	6,3	0,42	2	<0,5	0,316	<0,01	<0,01	0,13	<0,01	0,34	<0,01
Vertiente 4	Hualil-00027	2/12/2022	8,3	7,9	364	492	4,7	<0,04	7	<0,5	0,02	<0,01	<0,01	0,022	<0,01	0,32	<0,01
Pozo	Hualil-00028	2/12/2022	8,4	8,1	289	366	3,6	<0,04	<2	<0,5	<0,01	0,012	<0,01	0,019	<0,01	0,26	<0,01
Vertiente 5A	Hualil-00029	2/12/2022	8	7,9	303	400	6	<0,04	9	<0,5	<0,01	<0,01	<0,01	0,031	<0,01	0,3	<0,01
Vertiente 5B	Hualil-00030	2/12/2022	8	8	301	390	6	<0,04	9	<0,5	0,245	<0,01	<0,01	0,058	<0,01	0,32	<0,01
Vertiente 2	Hualil-00032	26/1/2023	6,6	7,7	560	808	6,7	0,1	11	<0,5	0,073	<0,01	<0,01	0,029	<0,01	0,66	<0,01
Vertiente 3	Hualil-00033	26/1/2023	7,2	7,6	381	504	6,3	0,05	8	<0,5	0,224	<0,01	<0,01	0,036	<0,01	0,36	<0,01
Vertiente 4	Hualil-00034	26/1/2023	8,4	7,7	363	578	4,7	<0,04	8	<0,5	0,019	<0,01	<0,01	0,035	<0,01	0,41	<0,01
Pozo	Hualil-00035	26/1/2023	8,6	8,1	288	466	3,6	<0,04	<2	<0,5	0,032	<0,01	<0,01	0,05	<0,01	0,36	<0,01
Vertiente 5A	Hualil-00037	26/1/2023	8,1	7,8	307	414	6	<0,04	10	<0,5	0,019	<0,01	<0,01	0,04	<0,01	0,35	<0,01
Vertiente 5B	Hualil-00038	26/1/2023	8,1	7,9	311	414	6	<0,04	10	<0,5	0,022	<0,01	<0,01	0,042	<0,01	0,37	<0,01
PA1	-	15/2/2023	8,06	-	-	-	5,57	-	-	-	-	-	-	-	-	-	-

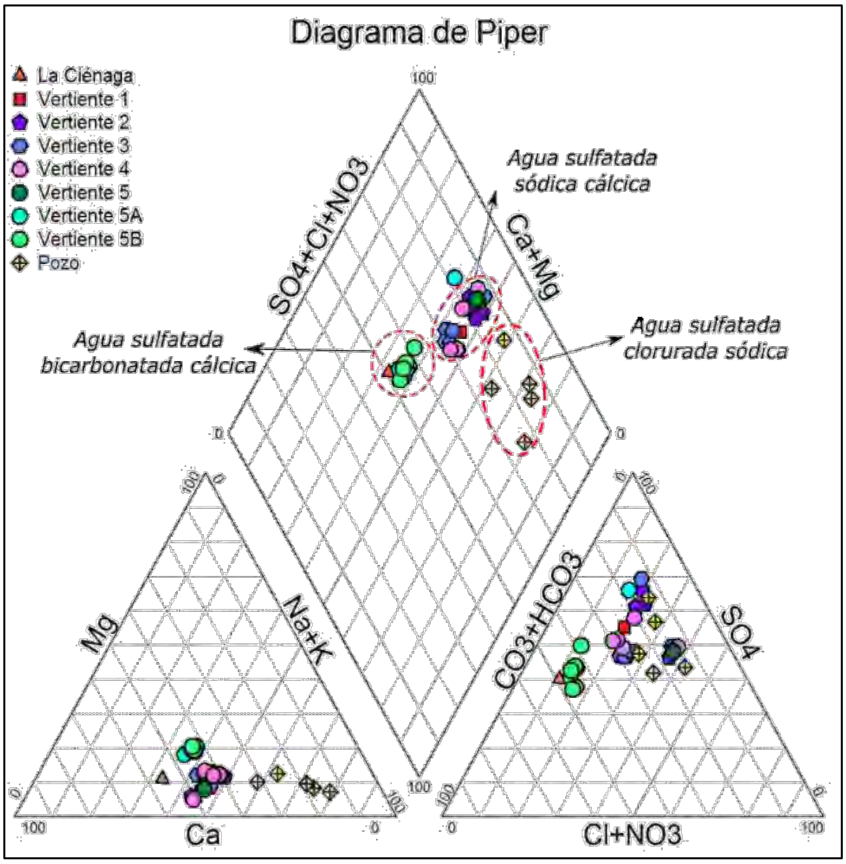


Figure 27-2: Piper diagram for all sampling points and water type classification

In Figure 27-3 and Figure 27-4 for the years 2007, 2020 and the first campaigns of 2022, it can be seen that the proportion of calcium and sodium is even, even in situations where calcium predominates, but the trend of the last samplings in general is that dominate sodium. Regarding the anions, in some initial monitoring, chloride has been even with sulphate, but the latter is markedly dominant. According to the classification of Custodio and Lamas (1976), the water is very hard with an average value of 260 mg/l CaCO3.

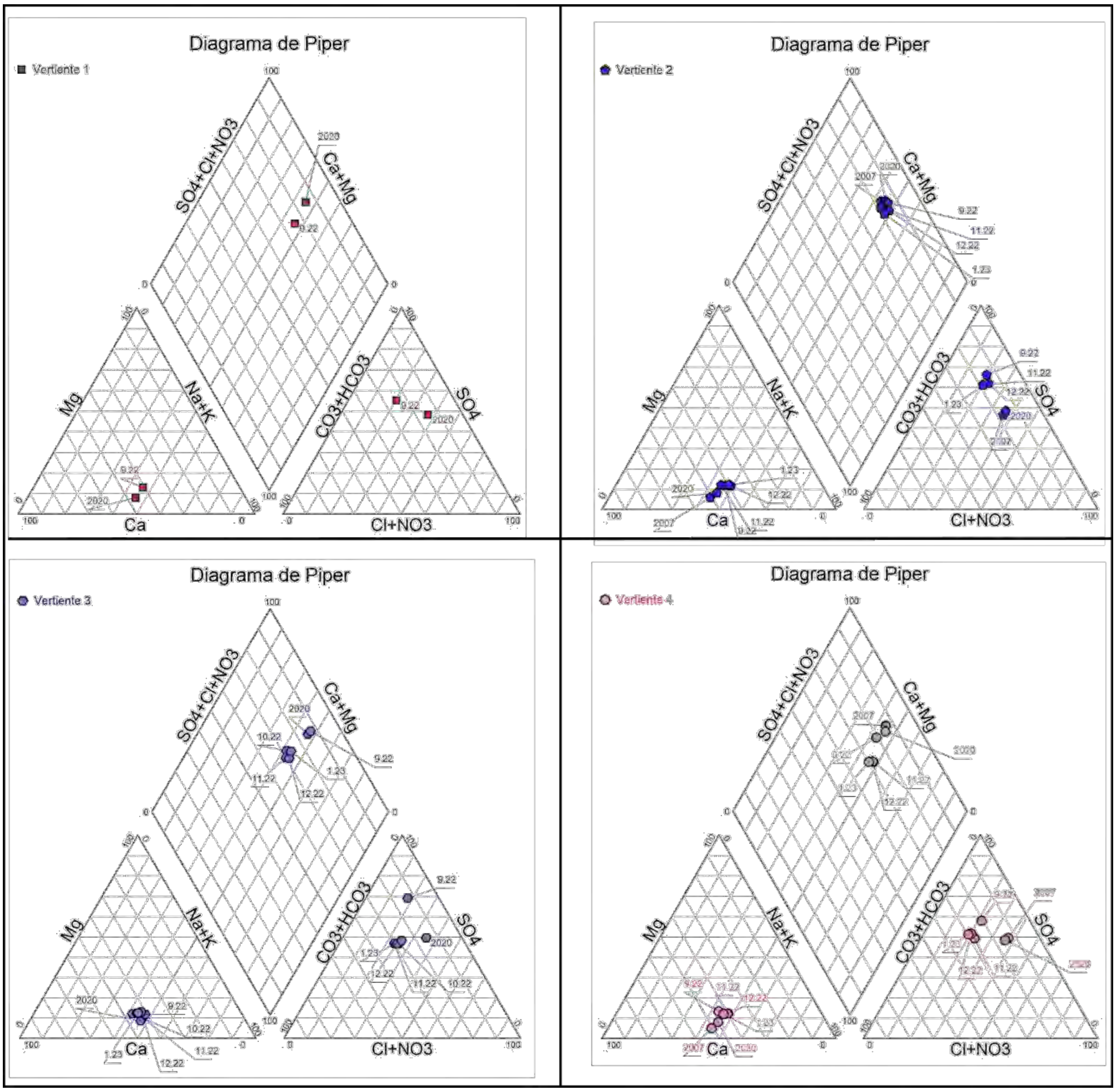


Figure 27-3: Piper diagrams for slopes 1 to 4 and sampling dates

The participation of nitrates stands out, with an average value of 8.1 mg/l and a maximum value of 15 mg/l. This situation may be attributable to the presence of animals on the site given the natural presence of surface water in an environment of extreme aridity. Regarding the metallic elements detected (average values), it has only been recorded on a few occasions: aluminium (0.3 mg/l), antimony (0.01 mg/l), copper (0.03 mg/l), chromium (0.01 mg/l), iron (0.04 mg/l), lead (0.01 mg/l), selenium (0.01 mg/l), vanadium (0.02 mg/l) and zinc (0.1mg/l). The average contents for barium and boron are 0.04 and 0.4 mg/l respectively. Arsenic, beryllium, cadmium, cobalt, hexavalent chromium, mercury, molybdenum, nickel, gold, palladium, silver and uranium have not been detected. Neither petroleum hydrocarbons nor cyanide have been detected.

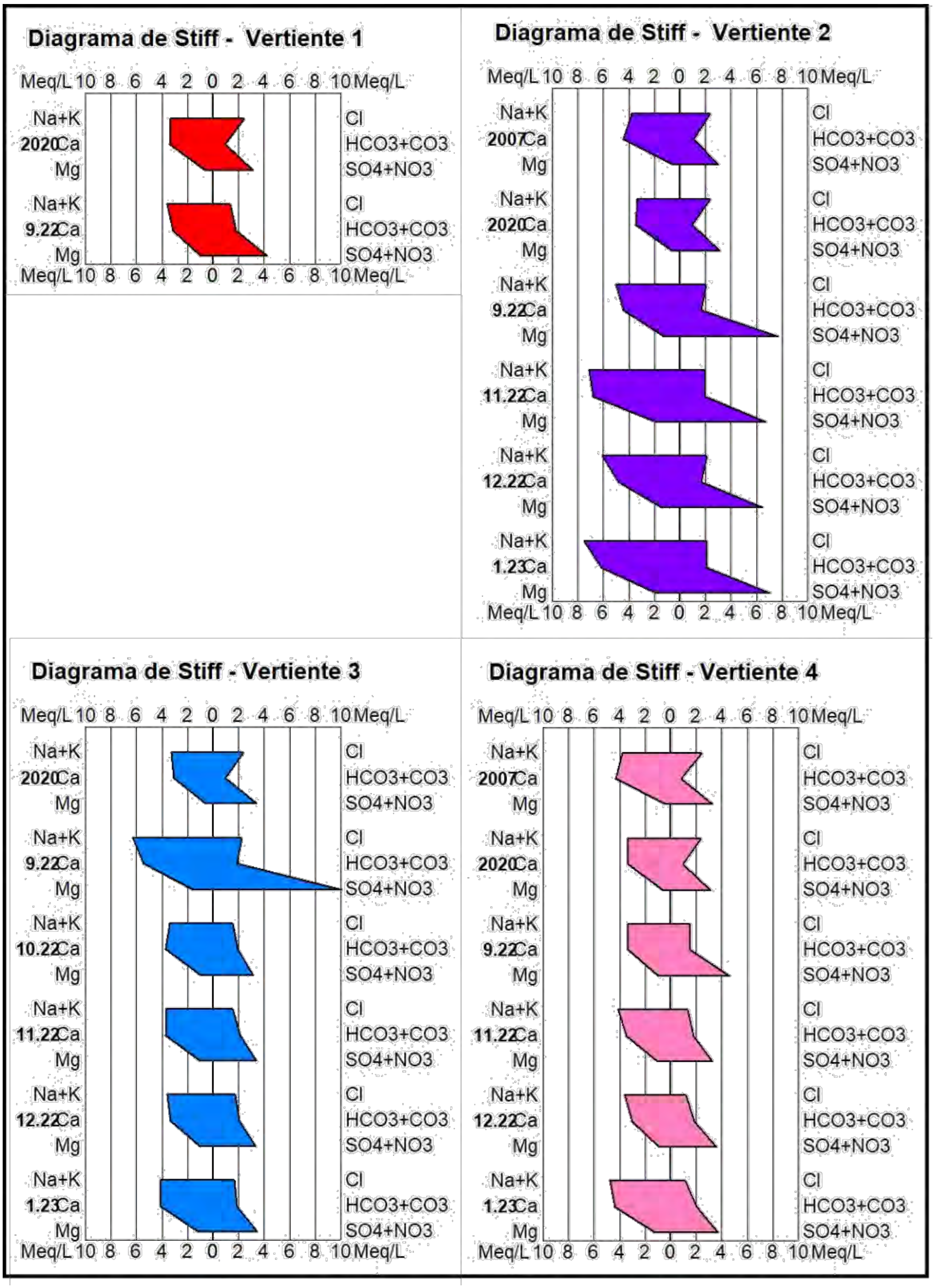


Figure 27-4: Stiff diagrams for slopes 1 to 4 and sampling dates

Springs 5A and 5B, together with the previous value of spring 5 and the historical record of the La Ciénaga spring, show to be the same type of water. The pH is neutral to slightly alkaline, with an average value of 7.8 pH measured in the field and 8.0 pH measured in the laboratory. Salinity is medium with an average conductivity of 607 uS/cm, lower than that registered in the other group of slopes. The average dissolved oxygen is 6.6 mg/l, which guarantees the levels necessary for the development of aquatic life.

From the point of view of the concentration and participation of majority anions and cations, the type of water is sulphated calcium bicarbonate. Although there is a fairly even participation of the bicarbonate and sulphate anions, the latter is slightly dominant. The relationship for this type of water is: SO42- ≥ HCO3- > Cl- and Ca2+ > Na ++ K+ >> Mg_2+. The average sulphate concentration is 114 mg/l, with a range between 49 and 175 mg/l. Regarding alkalinity, the average bicarbonate value is 129 mg/l, the highest in the entire region.

In Figure 27-5 and Figure 27-6, during the sampling of the year 2020, an important participation of chloride is observed and sodium + potassium equals calcium, but later, in the successive samplings, the proportions of anions and cations remain quite constant with sulphate and calcium dominance. This type of water is very similar to that recorded in the La Ciénaga spring in 1974. According to the classification of Custodio and Lamas (1996), the water is hard with an average value of 220 mg/l $CaCO_3$.

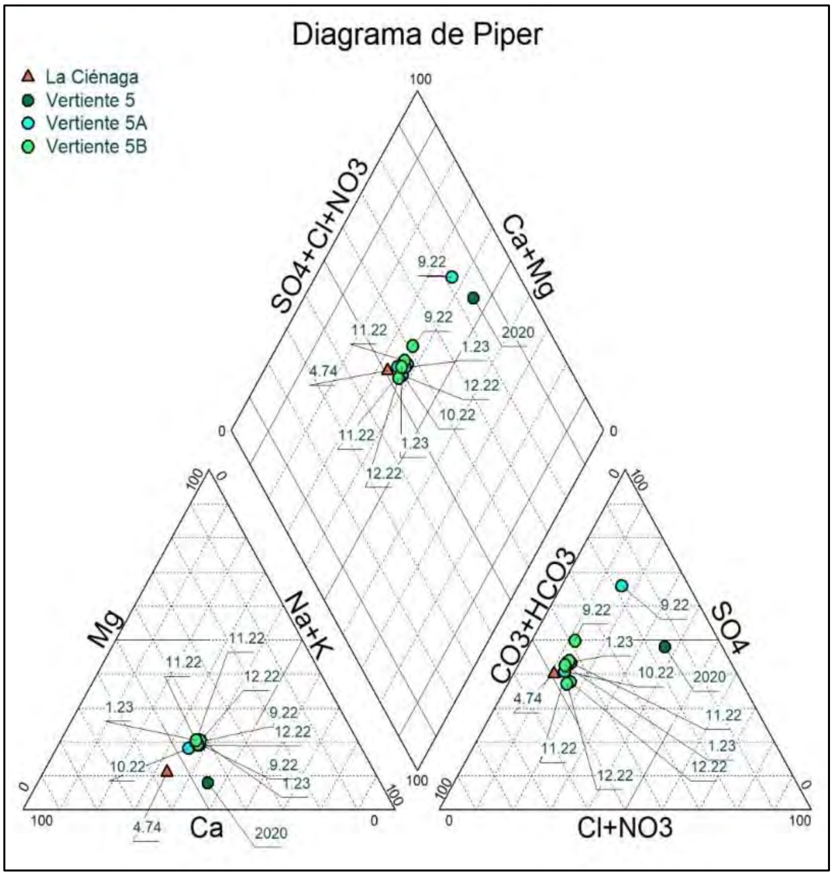


Figure 27-5: Piper diagrams for slopes 5, 5A, 5B and La Ciénaga and sampling dates

The participation of nitrates stands out, with an average value of 8.5 mg/l and a maximum value of 10 mg/l. In the same way as in the other slopes, this situation may be attributable to the presence of animals on the site given the natural presence of surface water in an environment of extreme aridity. Regarding the metallic elements detected (average values), it has only been recorded on a few occasions: aluminium (0.2 mg/l), antimony (0.01 mg/l), copper (0.03 mg/l), iron (0.04 mg/l), selenium (0.01 mg/l), vanadium (0.02 mg/l) and zinc (0.1

mg/l). The average contents for barium and boron are 0.04 and 0.3 mg/l respectively. Arsenic, beryllium, cadmium, cobalt, chromium, hexavalent chromium, mercury, molybdenum, nickel, gold, palladium, silver, lead and uranium have not been detected. Neither petroleum hydrocarbons nor cyanide have been detected.

The water registered in the well shows certain differences with respect to what is registered in the springs. The type of water is slightly alkaline to alkaline, with an average pH of 8.6 pH measured in the field and 8.4 pH in the laboratory. Thus, this water is the most alkaline of the three sites. Salinity is medium, with an average conductivity of 579 uS/cm. The average dissolved oxygen is 3.2 mg/l, insufficient for the proper development of aquatic life. This dissolved oxygen value is a reflection of groundwater, compatible with a lower degree of contact with the atmosphere.

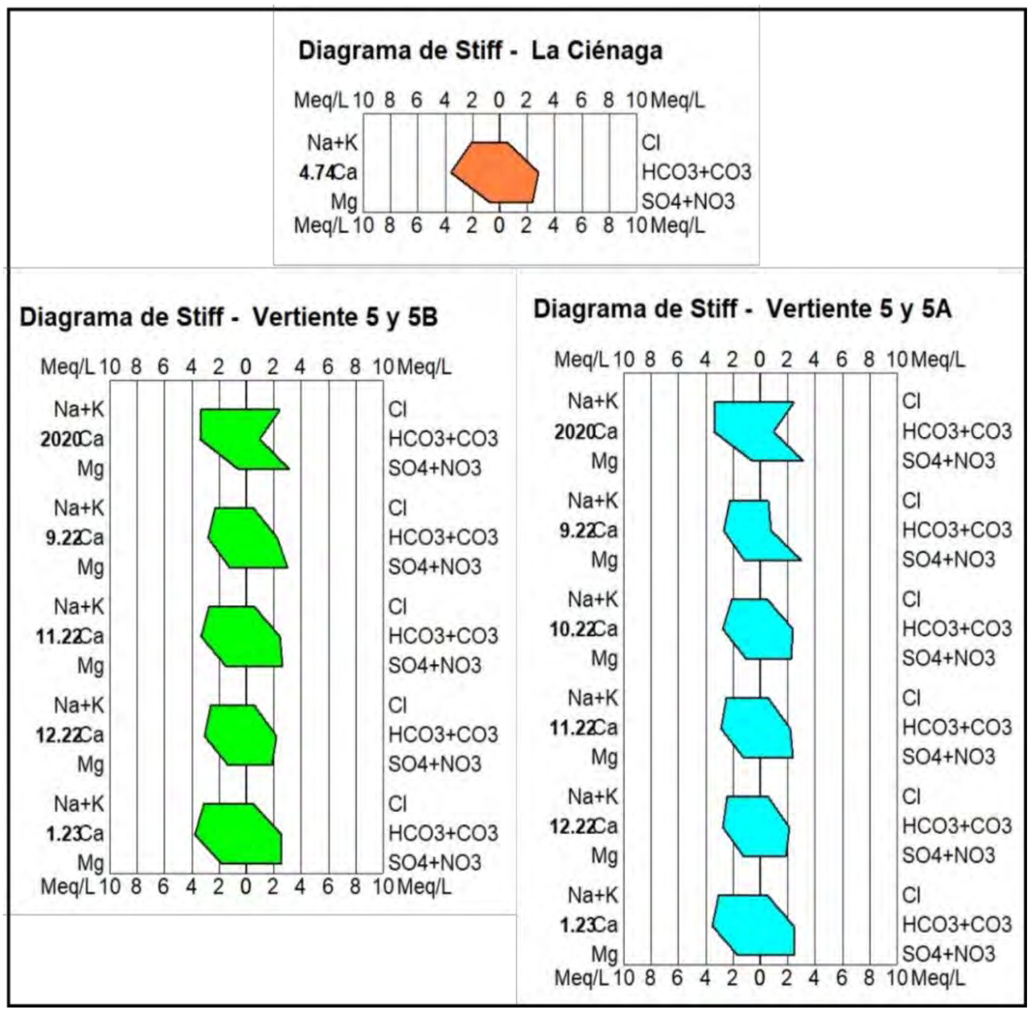


Figure 27-6: Stiff diagrams for slopes 5A, 5B and La Ciénaga and sampling dates

From the point of view of the concentration and participation of majority anions and cations, the type of water is sulphated sodium chloride. The relationship for this type of water is: $SO_4^{2-} > Cl^- > HCO_3^-$ and $Na^+ + K^+ >> Ca^{2+} > Mg^{2+}$.

In Figure 27-7 and Figure 27-8, the distinctive pattern of this type of water stands out, where sulphate dominates and, secondly, chloride. Regarding cations, sodium + potassium stands out above the rest. The average sulphate concentration is 126 mg/l with a range between 81 and 206 mg/l. Alkalinity is the lowest of the three groups, with an average content of 55.2 mg/l of bicarbonate. According to the classification of Custodio and Lamas (1996), the water is somewhat hard with an average value of 87 mg/l $CaCO_3$.

Unlike the springs, neither nitrates nor nitrites are detected in solution. Regarding the metallic elements detected (average values), it has only been recorded on a few occasions: aluminium (0.1 mg/l), antimony (0.01 mg/l), copper (0.03 mg/l), iron (0.1 mg/l), vanadium (0.02 mg/l) and zinc (0.1 mg/l). The average contents for barium and boron are 0.02 and 0.3 mg/l respectively. Arsenic, beryllium, cadmium, cobalt, chromium, hexavalent chromium, mercury, molybdenum, nickel, gold, palladium, silver, lead, selenium, and uranium have not been detected. Neither petroleum hydrocarbons nor cyanide have been detected.

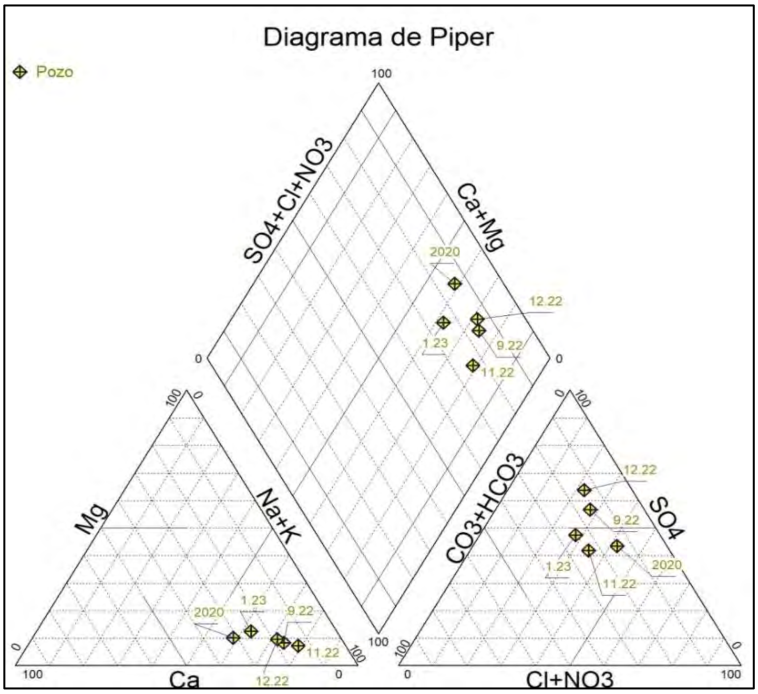


Figure 27-7: Piper diagrams for the well and sampling dates

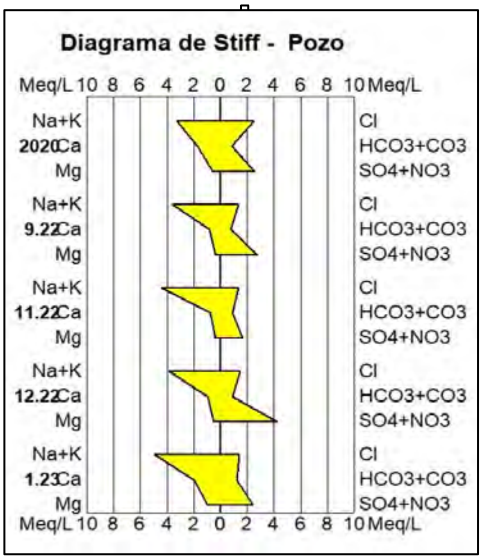


Figure 27-8: Stiff Diagrams of the well and sampling dates

In Figure 27-8 the spatial distribution of these water groups can be observed more clearly from the use of Stiff graphs with the average values per station. As a common factor, the neutral to alkaline pH of the waters, the average salinity and the dominance of sulphate over the rest of the anions stand out. It is to be expected that, during the natural course of the water, it will go from being bi-carbonated to sulphated and finally chlorinated. The sampled sites are within the framework of an endorheic basin under a climate of intense aridity and water deficit, this situation contributes to a greater presence of sulphate and chloride over bicarbonate. For all cases the presence of metallic elements is very poor, where those that are usually recorded, in at least one monitoring are: aluminium, antimony, barium, boron, copper, chromium, iron, lead, selenium, vanadium and zinc. It should be noted that elements with a high degree of toxicity such as arsenic, cadmium, cobalt, hexavalent chromium, mercury, petroleum hydrocarbons or cyanide have not been registered in solution.

The pH and conductivity data obtained from the PA1 hydrogeological well are within the previously mentioned levels. For this site, the pH registered during February 2023 was 8.06 pH and the conductivity was 785 uS/cm.

Appendix 6 – SGS Report

Appendix 6 - SGS 18000-01 & 02 - Final Report.pdf

Appendix 6 - SGS
18000-01 & 02 - Final

Appendix 7 – GRG Test Report

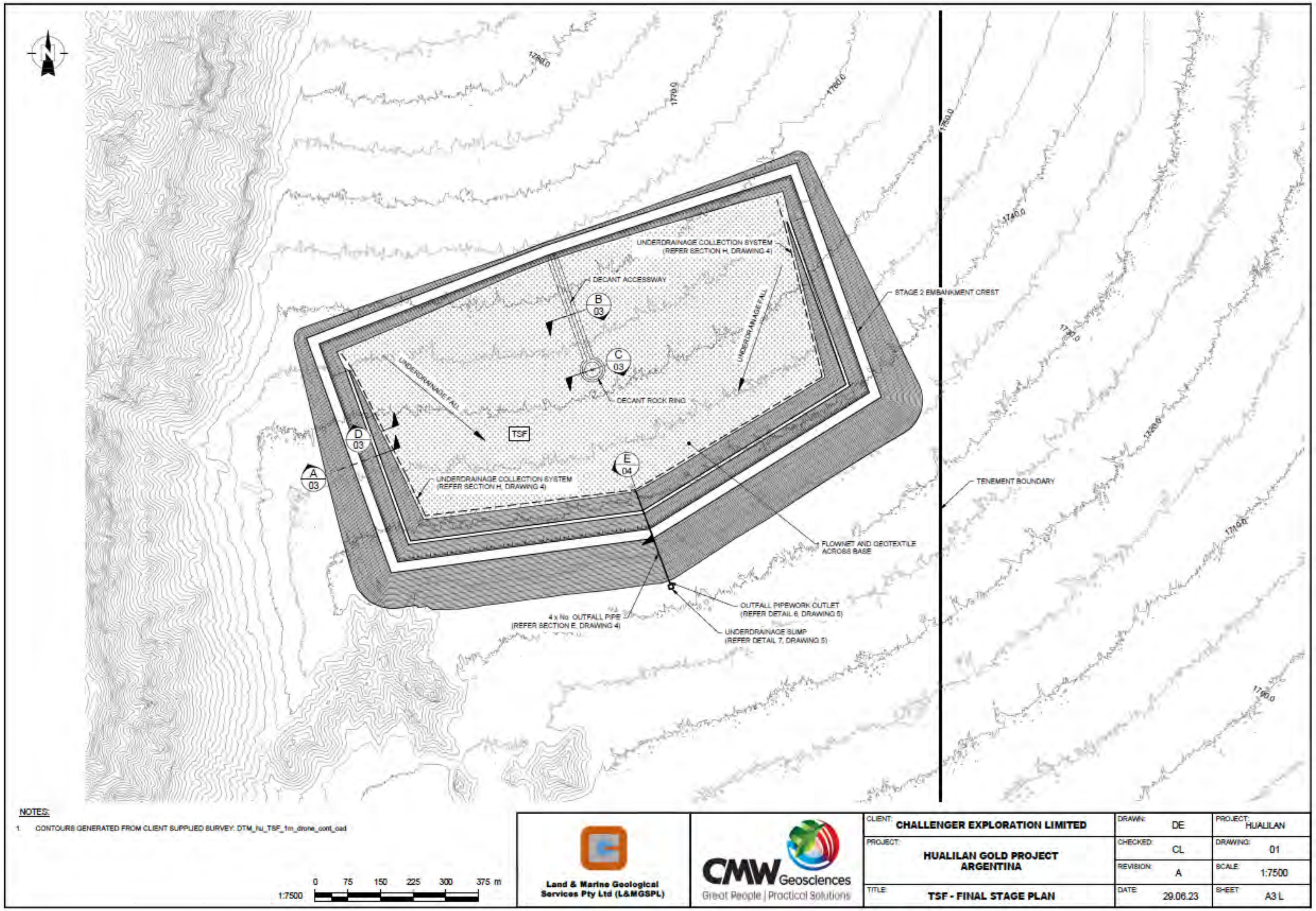
Appendix 7 - FLSmidth Gravity Modelling rev 1.pdf

Appendix 7 -
FLSmidth Gravity Mod

Appendix 8 – TSF Drawings

Appendix 9 - Combined Drawings HUALILAN-02 TSF 191023.pdf

Appendix 9 -
Combined Drawings H


UNDERDRAINAGE COLLECTION SYSTEM
(REFER SECTION H, DRAWING 4)
DECANT ACCESSWAY
STAGE 2 EMBANKMENT CREST
UNDERDRAINAGE FALL
DECANT ROCK RING
TSF
TENEMENT BOUNDARY
UNDERDRAINAGE COLLECTION SYSTEM
(REFER SECTION H, DRAWING 4)
FLOWNET AND GEOTEXTILE
ACROSS BASE
OUTFALL PIPEWORK OUTLET
(REFER DETAIL 6, DRAWING 5)
4 x No. OUTFALL PIPE
(REFER SECTION E, DRAWING 4)
UNDERDRAINAGE SUMP
(REFER DETAIL 7, DRAWING 5)
1740.0
1720.0
1710.0
NOTES:
1. CONTOURS GENERATED FROM CLIENT SUPPLIED SURVEY: DTM_hu_TSF_1m_drone_cont_cad
0 75 150 225 300 375 m
1:7500
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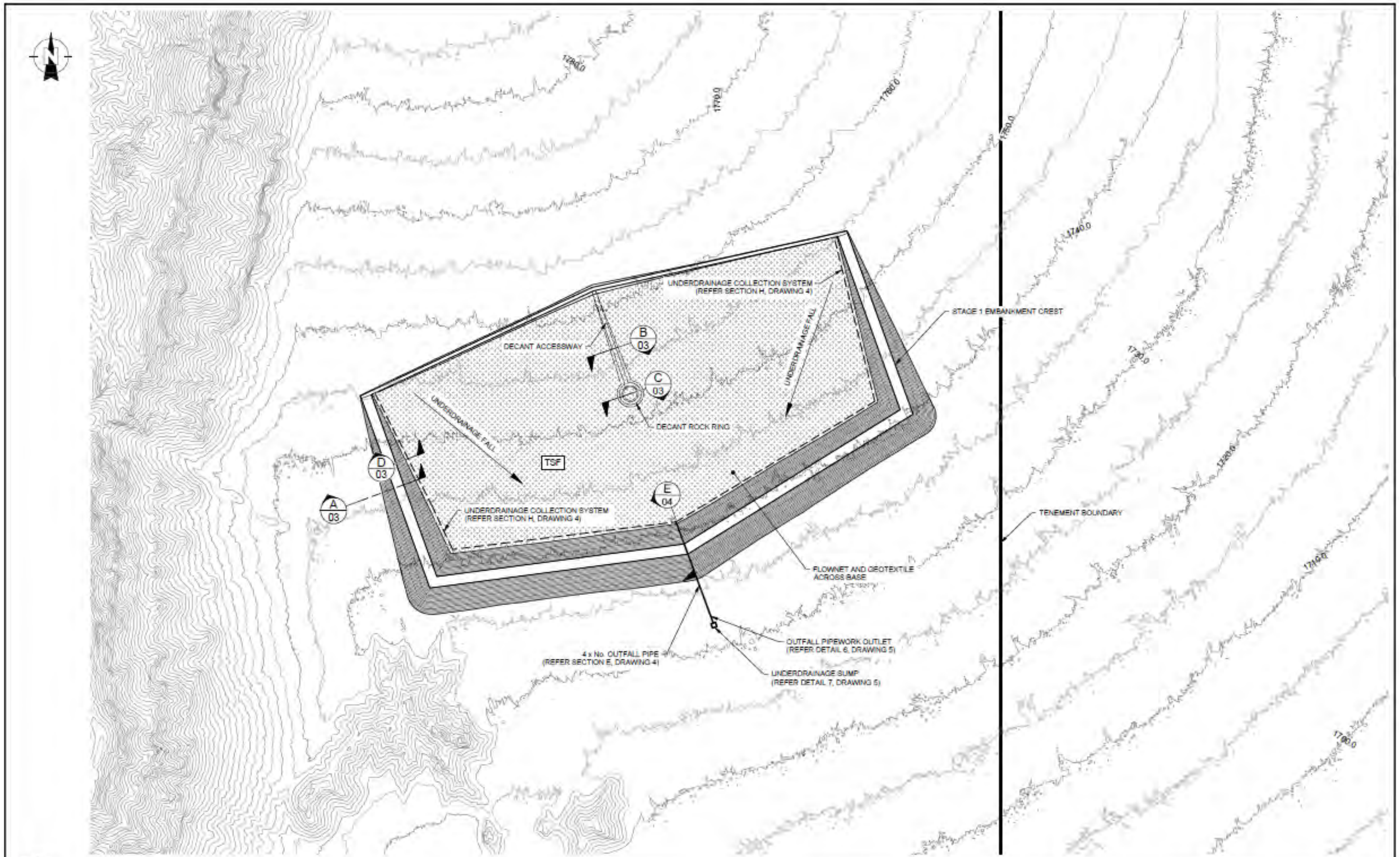
CLIENT: CHALLENGER EXPLORATION LIMITED
PROJECT: HUALILAN GOLD PROJECT ARGENTINA
TITLE: TSF - FINAL STAGE PLAN
DRAWN: DE
CHECKED: CL
REVISION: A
DATE: 29.06.23
PROJECT: HUALILAN
DRAWING: 01
SCALE: 1:7500
SHEET: A3 L



NOTES:

1 CONTOURS GENERATED FROM CLIENT SUPPLIED SURVEY: DTM_hu_TSF_1m_drone_cont_cad

0 75 150 225 300 375 m

1:7500

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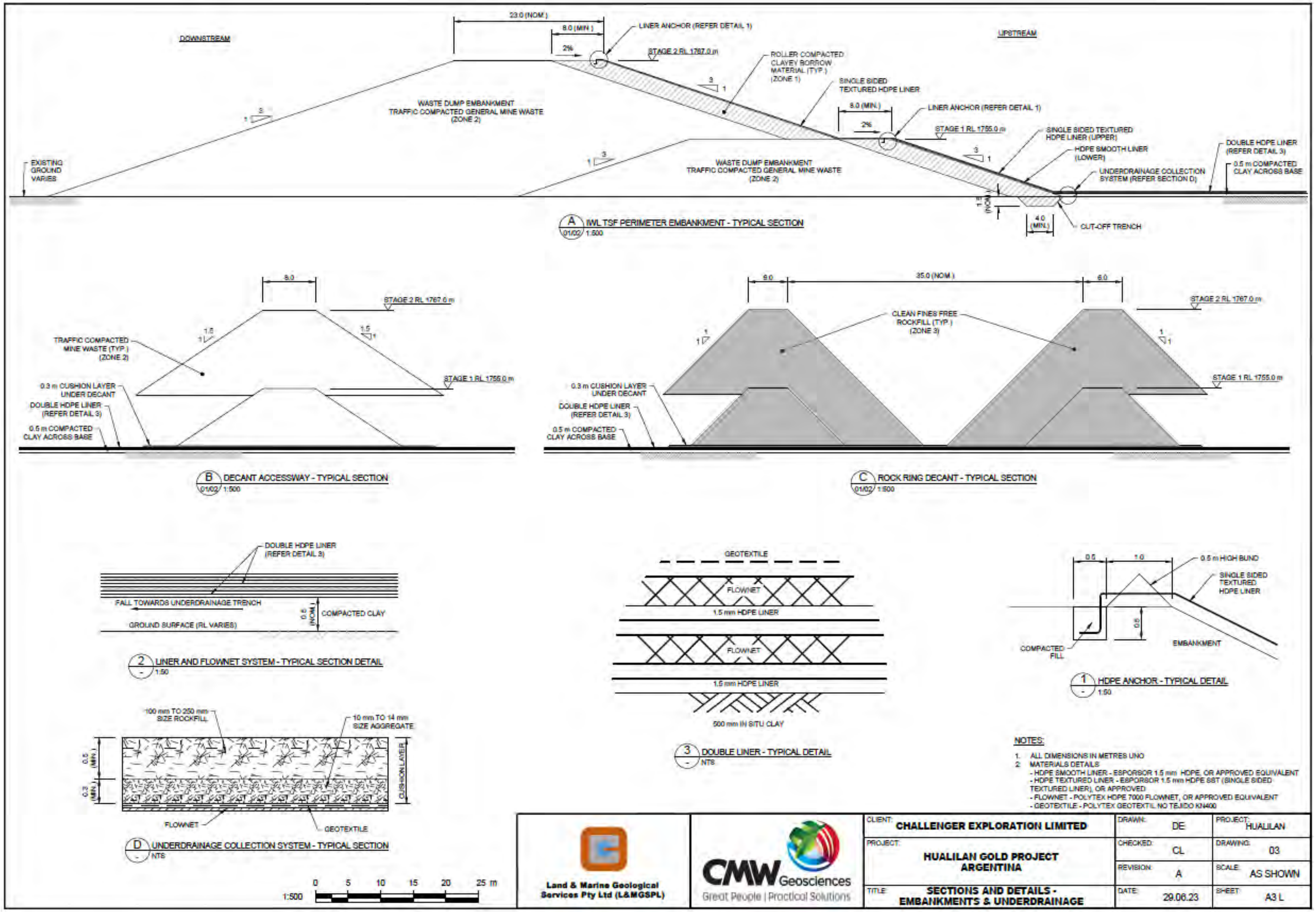
PROJECT: HUALILAN GOLD PROJECT ARGENTINA	CHECKED: CL	DRAWING: 02
	REVISION: A	SCALE: 1:7500
TITLE: TSF - STAGE 1 PLAN	DATE: 29.06.23	SHEET: A3 L


DOWNSTREAM
UPSTREAM
23.0 (NOM.)
8.0 (MIN.)
2%
LINER ANCHOR (REFER DETAIL 1)
STAGE 2 RL 1767.0 m
ROLLER COMPACTED CLAYEY BORROW MATERIAL (TYP.) (ZONE 1)
SINGLE SIDED TEXTURED HDPE LINER
WASTE DUMP EMBANKMENT TRAFFIC COMPACTED GENERAL MINE WASTE (ZONE 2)
STAGE 1 RL 1755.0 m
SINGLE SIDED TEXTURED HDPE LINER (UPPER)
HDPE SMOOTH LINER (LOWER)
UNDERDRAINAGE COLLECTION SYSTEM (REFER SECTION D)
DOUBLE HDPE LINER (REFER DETAIL 3)
0.5 m COMPACTED CLAY ACROSS BASE
EXISTING GROUND VARIES
CUT-OFF TRENCH
A IWL TSF PERIMETER EMBANKMENT - TYPICAL SECTION 01/02 1:500
TRAFFIC COMPACTED MINE WASTE (TYP.) (ZONE 2)
0.3 m CUSHION LAYER UNDER DECANT
B DECANT ACCESSWAY - TYPICAL SECTION 01/02 1:500
35.0 (NOM.)
CLEAN FINES FREE ROCKFILL (TYP.) (ZONE 3)
C ROCK RING DECANT - TYPICAL SECTION 01/02 1:500
FALL TOWARDS UNDERDRAINAGE TRENCH
COMPACTED CLAY
GROUND SURFACE (RL VARIES)
2 LINER AND FLOWNET SYSTEM - TYPICAL SECTION DETAIL 1:50
GEOTEXTILE
FLOWNET
1.5 mm HDPE LINER
500 mm IN SITU CLAY
3 DOUBLE LINER - TYPICAL DETAIL NTS
0.5 m HIGH BUND
SINGLE SIDED TEXTURED HDPE LINER
COMPACTED FILL
EMBANKMENT
1 HDPE ANCHOR - TYPICAL DETAIL 1:50
100 mm TO 250 mm SIZE ROCKFILL
10 mm TO 14 mm SIZE AGGREGATE
CUSHION LAYER
D UNDERDRAINAGE COLLECTION SYSTEM - TYPICAL SECTION NTS
NOTES:
1. ALL DIMENSIONS IN METRES UNO
2. MATERIALS DETAILS
- HDPE SMOOTH LINER - ESPORSOR 1.5 mm HDPE, OR APPROVED EQUIVALENT
- HDPE TEXTURED LINER - ESPORSOR 1.5 mm HDPE SST (SINGLE SIDED TEXTURED LINER), OR APPROVED
- FLOWNET - POLYTEX HDPE 7000 FLOWNET, OR APPROVED EQUIVALENT
- GEOTEXTILE - POLYTEX GEOTEXTIL NO TEJIDO KN400
1:500
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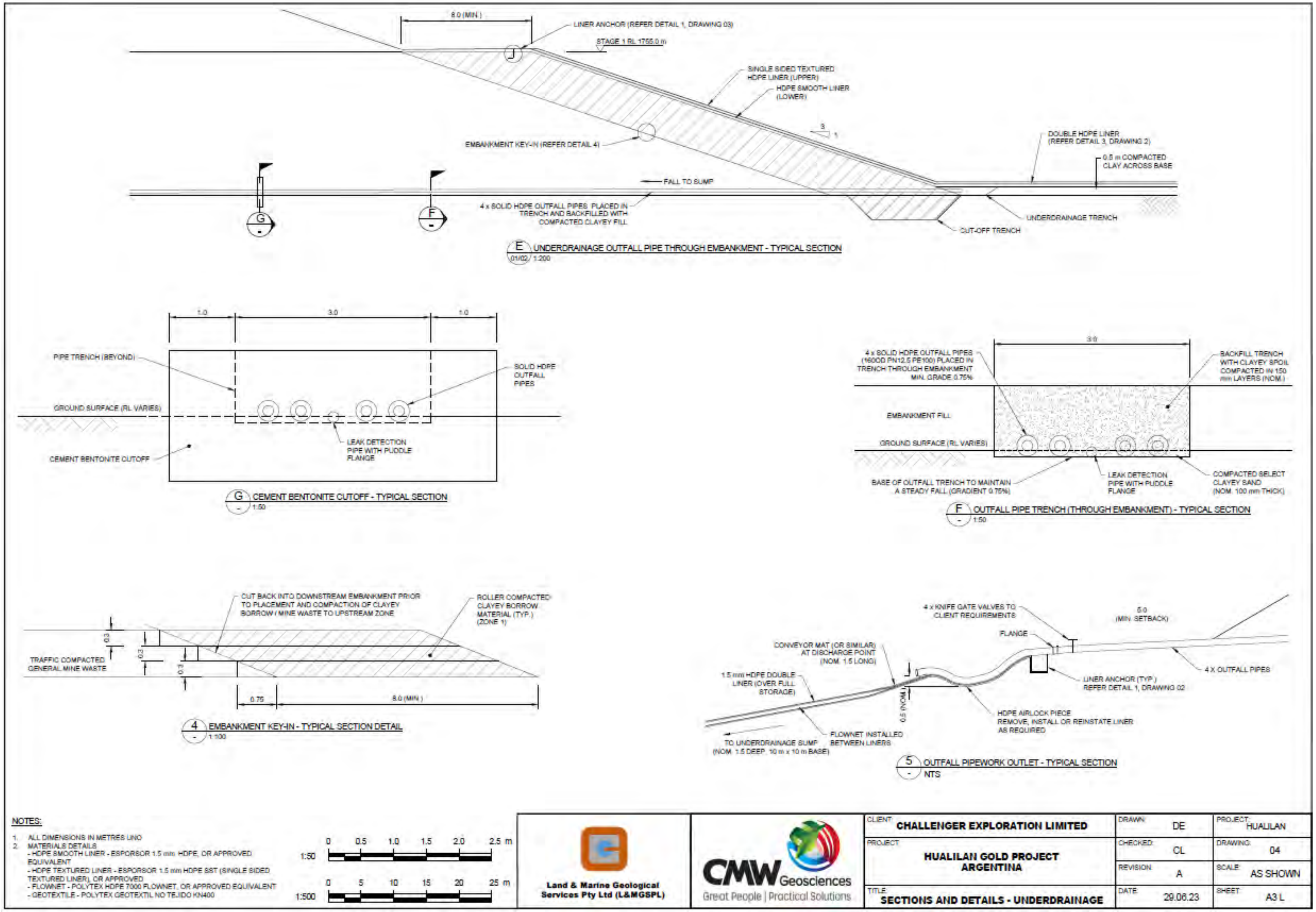
PROJECT: HUALILAN GOLD PROJECT ARGENTINA
TITLE: SECTIONS AND DETAILS - EMBANKMENTS & UNDERDRAINAGE
DRAWN: DE
CHECKED: CL
REVISION: A
DATE: 29.06.23
PROJECT: HUALILAN
DRAWING: 03
SCALE: AS SHOWN
SHEET: A3 L


8.0 (MIN.)
LINER ANCHOR (REFER DETAIL 1, DRAWING 03)
STAGE 1 RL 1755.0 m
SINGLE SIDED TEXTURED HDPE LINER (UPPER)
HDPE SMOOTH LINER (LOWER)
EMBANKMENT KEY-IN (REFER DETAIL 4)
DOUBLE HDPE LINER (REFER DETAIL 3, DRAWING 2)
0.5 m COMPACTED CLAY ACROSS BASE
FALL TO SUMP
4 x SOLID HDPE OUTFALL PIPES PLACED IN TRENCH AND BACKFILLED WITH COMPACTED CLAYEY FILL
UNDERDRAINAGE TRENCH
CUT-OFF TRENCH
E UNDERDRAINAGE OUTFALL PIPE THROUGH EMBANKMENT - TYPICAL SECTION
01/02 1:200
PIPE TRENCH (BEYOND)
GROUND SURFACE (RL VARIES)
CEMENT BENTONITE CUTOFF
SOLID HDPE OUTFALL PIPES
LEAK DETECTION PIPE WITH PUDDLE FLANGE
G CEMENT BENTONITE CUTOFF - TYPICAL SECTION
1:50
4 x SOLID HDPE OUTFALL PIPES (160OD PN12.5 PE100) PLACED IN TRENCH THROUGH EMBANKMENT MIN. GRADE 0.75%
BACKFILL TRENCH WITH CLAYEY SPOIL COMPACTED IN 150 mm LAYERS (NOM.)
EMBANKMENT FILL
GROUND SURFACE (RL VARIES)
BASE OF OUTFALL TRENCH TO MAINTAIN A STEADY FALL (GRADIENT 0.75%)
LEAK DETECTION PIPE WITH PUDDLE FLANGE
COMPACTED SELECT CLAYEY SAND (NOM. 100 mm THICK)
F OUTFALL PIPE TRENCH (THROUGH EMBANKMENT) - TYPICAL SECTION
1:50
CUT BACK INTO DOWNSTREAM EMBANKMENT PRIOR TO PLACEMENT AND COMPACTION OF CLAYEY BORROW / MINE WASTE TO UPSTREAM ZONE
ROLLER COMPACTED CLAYEY BORROW MATERIAL (TYP.) (ZONE 1)
TRAFFIC COMPACTED GENERAL MINE WASTE
0.75
8.0 (MIN.)
4 EMBANKMENT KEY-IN - TYPICAL SECTION DETAIL
1:100
4 x KNIFE GATE VALVES TO CLIENT REQUIREMENTS
5.0 (MIN. SETBACK)
FLANGE
CONVEYOR MAT (OR SIMILAR) AT DISCHARGE POINT (NOM. 1.5 LONG)
1.5 mm HDPE DOUBLE LINER (OVER FULL STORAGE)
4 X OUTFALL PIPES
LINER ANCHOR (TYP.) REFER DETAIL 1, DRAWING 02
HDPE AIRLOCK PIECE REMOVE, INSTALL OR REINSTATE LINER AS REQUIRED
0.5 (NOM.)
TO UNDERDRAINAGE SUMP (NOM. 1.5 DEEP, 10 m x 10 m BASE)
FLOWNET INSTALLED BETWEEN LINERS
5 OUTFALL PIPEWORK OUTLET - TYPICAL SECTION
NTS
NOTES:
1. ALL DIMENSIONS IN METRES UNO
2. MATERIALS DETAILS
- HDPE SMOOTH LINER - ESPORSOR 1.5 mm HDPE, OR APPROVED EQUIVALENT
- HDPE TEXTURED LINER - ESPORSOR 1.5 mm HDPE SST (SINGLE SIDED TEXTURED LINER), OR APPROVED
- FLOWNET - POLYTEX HDPE 7000 FLOWNET, OR APPROVED EQUIVALENT
- GEOTEXTILE - POLYTEX GEOTEXTIL NO TEJIDO KN400
1:50 0 0.5 1.0 1.5 2.0 2.5 m
1:500 0 5 10 15 20 25 m
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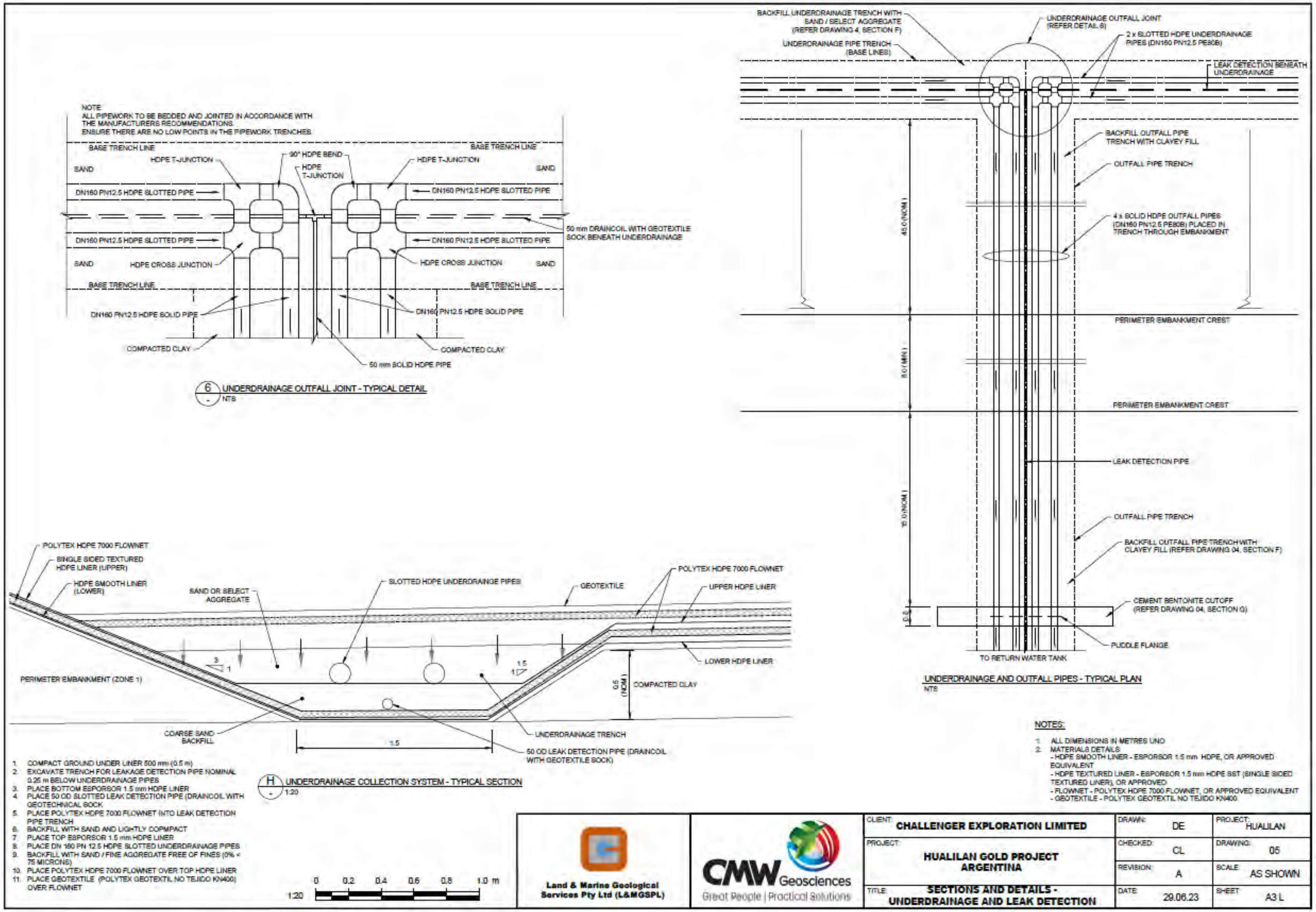
CLIENT: CHALLENGER EXPLORATION LIMITED
PROJECT: HUALILAN GOLD PROJECT ARGENTINA
TITLE: SECTIONS AND DETAILS - UNDERDRAINAGE
DRAWN: DE
CHECKED: CL
REVISION: A
DATE: 29.06.23
PROJECT: HUALILAN
DRAWING: 04
SCALE: AS SHOWN
SHEET: A3 L


NOTE
ALL PIPEWORK TO BE BEDDED AND JOINTED IN ACCORDANCE WITH THE MANUFACTURERS RECOMMENDATIONS.
ENSURE THERE ARE NO LOW POINTS IN THE PIPEWORK TRENCHES.
BASE TRENCH LINE
SAND
HDPE T-JUNCTION
90° HDPE BEND
HDPE T-JUNCTION
DN160 PN12.5 HDPE SLOTTED PIPE
50 mm DRAINCOIL WITH GEOTEXTILE SOCK BENEATH UNDERDRAINAGE
HDPE CROSS JUNCTION
DN160 PN12.5 HDPE SOLID PIPE
COMPACTED CLAY
50 mm SOLID HDPE PIPE
6 UNDERDRAINAGE OUTFALL JOINT - TYPICAL DETAIL
NTS
POLYTEX HDPE 7000 FLOWNET
SINGLE SIDED TEXTURED HDPE LINER (UPPER)
HDPE SMOOTH LINER (LOWER)
SAND OR SELECT AGGREGATE
SLOTTED HDPE UNDERDRAINAGE PIPES
GEOTEXTILE
UPPER HDPE LINER
LOWER HDPE LINER
PERIMETER EMBANKMENT (ZONE 1)
COMPACTED CLAY
COARSE SAND BACKFILL
UNDERDRAINAGE TRENCH
50 OD LEAK DETECTION PIPE (DRAINCOIL WITH GEOTEXTILE SOCK)
1. COMPACT GROUND UNDER LINER 500 mm (0.5 m)
2. EXCAVATE TRENCH FOR LEAKAGE DETECTION PIPE NOMINAL 0.25 m BELOW UNDERDRAINAGE PIPES
3. PLACE BOTTOM ESPORSOR 1.5 mm HDPE LINER
4. PLACE 50 OD SLOTTED LEAK DETECTION PIPE (DRAINCOIL WITH GEOTECHNICAL SOCK
5. PLACE POLYTEX HDPE 7000 FLOWNET INTO LEAK DETECTION PIPE TRENCH
6. BACKFILL WITH SAND AND LIGHTLY COPMPACT
7. PLACE TOP ESPORSOR 1.5 mm HDPE LINER
8. PLACE DN 160 PN 12.5 HDPE SLOTTED UNDERDRAINAGE PIPES
9. BACKFILL WITH SAND / FINE AGGREGATE FREE OF FINES (0% < 75 MICRONS)
10. PLACE POLYTEX HDPE 7000 FLOWNET OVER TOP HDPE LINER
11. PLACE GEOTEXTILE (POLYTEX GEOTEXTIL NO TEJIDO KN400) OVER FLOWNET
H UNDERDRAINAGE COLLECTION SYSTEM - TYPICAL SECTION
1:20
BACKFILL UNDERDRAINAGE TRENCH WITH SAND / SELECT AGGREGATE (REFER DRAWING 4, SECTION F)
UNDERDRAINAGE PIPE TRENCH (BASE LINES)
UNDERDRAINAGE OUTFALL JOINT (REFER DETAIL 6)
2 x SLOTTED HDPE UNDERDRAINAGE PIPES (DN160 PN12.5 PE80B)
LEAK DETECTION BENEATH UNDERDRAINAGE
BACKFILL OUTFALL PIPE TRENCH WITH CLAYEY FILL
OUTFALL PIPE TRENCH
4 x SOLID HDPE OUTFALL PIPES (DN160 PN12.5 PE80B) PLACED IN TRENCH THROUGH EMBANKMENT
PERIMETER EMBANKMENT CREST
LEAK DETECTION PIPE
BACKFILL OUTFALL PIPE TRENCH WITH CLAYEY FILL (REFER DRAWING 04, SECTION F)
CEMENT BENTONITE CUTOFF (REFER DRAWING 04, SECTION G)
PUDDLE FLANGE
TO RETURN WATER TANK
UNDERDRAINAGE AND OUTFALL PIPES - TYPICAL PLAN
NTS
NOTES:
1. ALL DIMENSIONS IN METRES UNO
2. MATERIALS DETAILS
- HDPE SMOOTH LINER - ESPORSOR 1.5 mm HDPE, OR APPROVED EQUIVALENT
- HDPE TEXTURED LINER - ESPORSOR 1.5 mm HDPE SST (SINGLE SIDED TEXTURED LINER), OR APPROVED
- FLOWNET - POLYTEX HDPE 7000 FLOWNET, OR APPROVED EQUIVALENT
- GEOTEXTILE - POLYTEX GEOTEXTIL NO TEJIDO KN400
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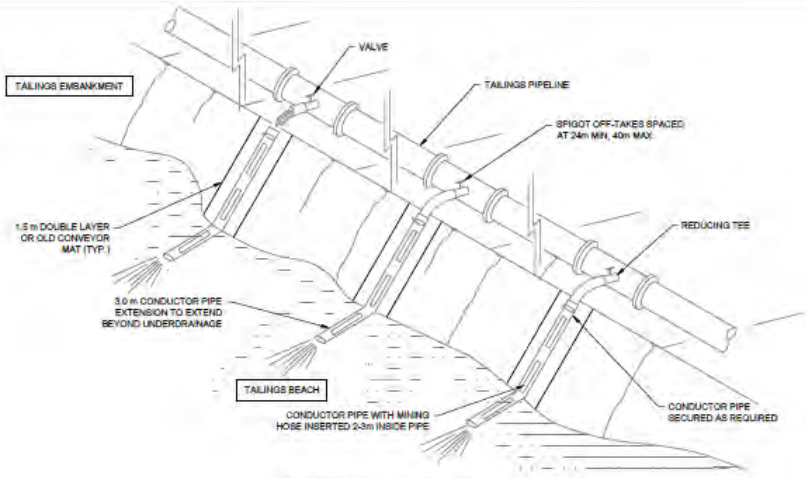
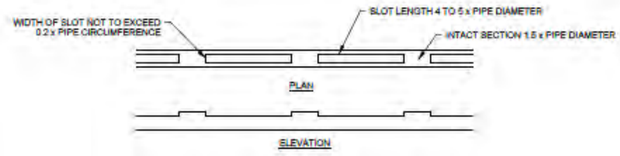
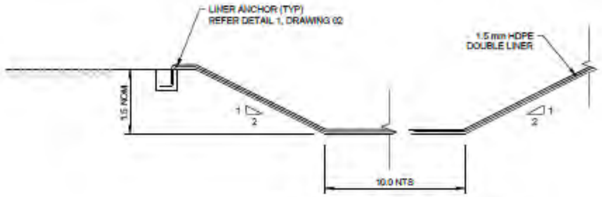
CLIENT: CHALLENGER EXPLORATION LIMITED
PROJECT: HUALILAN GOLD PROJECT ARGENTINA
TITLE: SECTIONS AND DETAILS - UNDERDRAINAGE AND LEAK DETECTION
DRAWN: DE
CHECKED: CL
REVISION: A
DATE: 29.06.23
PROJECT: HUALILAN
DRAWING: 05
SCALE: AS SHOWN
SHEET: A3 L



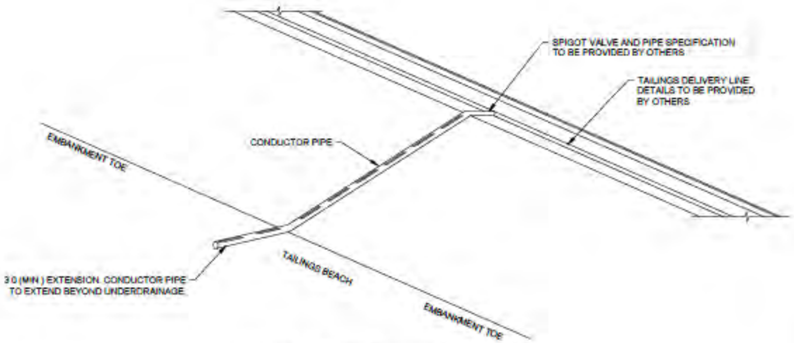
PROPOSED SPIGOT ARRANGEMENT
ISOMETRIC VIEW (NTS)



7 CONDUCTOR PIPE (SLOTTED PVC) - TYPICAL DETAIL
1:50



8 UNDERDRAINAGE SUMP - TYPICAL DETAIL
1:100



DRAINAGE / DEPOSITION EMBANKMENT
PERSPECTIVE VIEW (NTS)

NOTES:

1. ALL DIMENSIONS IN METRES UNO
2. MATERIALS DETAILS
   - HDPE SMOOTH LINER - ESPORSOR 1.5 mm HDPE, OR APPROVED EQUIVALENT
   - HDPE TEXTURED LINER - ESPORSOR 1.5 mm HDPE SST (SINGLE SIDED TEXTURED LINER), OR APPROVED
   - FLOWNET - POLYTEX HDPE 7000 FLOWNET, OR APPROVED EQUIVALENT
   - GEOTEXTILE - POLYTEX GEOTEXTIL NO TEJIDO KN400

1:100 — 0 1 2 3 4 5 m

1:50 — 0 0.5 1.0 1.5 2.0 2.5 m

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PROJECT: HUALILAN GOLD PROJECT ARGENTINA	CHECKED: CL	DRAWING: 06	
	REVISION: A	SCALE: AS SHOWN	
TITLE: SECTIONS AND DETAILS - PIPEWORK AND UNDERDRAINAGE SUMP	DATE: 29.06.23	SHEET: A3 L	

JORC Code, 2012 Edition – Table 1 report template

Section 1 Sampling Techniques and Data -Hualilan Project

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	- *Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling.* - *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* - *Aspects of the determination of mineralisation that are Material to the Public Report.* - *In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases, more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.*	Diamond core (HQ3 and NQ3) was cut longitudinally on site using a diamond saw or split using a hand operated hydraulic core sampling splitter. Samples lengths are generally from 0.5m to 2.0m in length (average 1.74m). Sample lengths are selected according to lithology, alteration, and mineralization contacts. For reverse circulation (RC) drilling, 2-4 kg sub-samples from each 1m drilled were collected from a face sample recovery cyclone mounted on the drill machine. Channel samples are cut into underground or surface outcrop using a hand-held diamond edged cutting tool. Parallel saw cuts 3-5cm apart are cut 2-4cm deep into the rock which allows for the extraction of a representative sample using a hammer and chisel. The sample is collected onto a plastic mat and collected into a sample bag. Core, RC and channel samples were crushed to approximately 85% passing 2mm. A 500g or a 1 kg sub-sample was taken and pulverized to 85% passing 75µm. A 50g charge was analysed for Au by fire assay with AA determination. Where the fire assay grade is > 10 g/t gold, a 50g charge was analysed for Au by Fire assay with gravimetric determination. A 10g charge was analysed for at least 48 elements by 4-acid digest and ICP-MS determination. Elements determined include Ag, As, Ba, Be, Bi, Ca, 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 and Zr. For Ag > 100 g/t, Zn, Pb and Cu > 10,000 ppm and S > 10%, overlimit analysis was done by the same method using a different calibration. Unused pulps are returned from the laboratory to the Project and stored in a secure location, so they are available for any further analyses. Remaining drill core is stored undercover for future use if required. Visible gold observed has been observed in only 1 drill core sample only. Coarse gold is not likely to result in sample bias. Historic Data: There is little information provided by previous explorers to detail sampling techniques. Selected drill core was cut with a diamond saw longitudinally and one half submitted for assay. Assay was generally done for Au. In some drill campaigns, Ag and Zn were also analysed. There is limited multielement data available. No information is available for RC drill techniques and sampling.

Criteria	JORC Code explanation	Commentary
Drilling techniques	- *Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	CEL drilling of HQ3 core (triple tube) was done using various truck and track mounted drill machines that are operated by various drilling contractors based in Mendoza and San Juan. The core has not been oriented as the rock is commonly too broken to allow accurate core orientation. CEL drilling of reverse circulation (RC) drill holes was done using a track-mounted LM650 universal drill rig set up for reverse circulation drilling. Drilling was done using a 5.25 inch hammer bit. Collar details for historic drill holes, DD drill holes, RC drill holes completed by CEL that are used in the resource estimate are detailed in CEL ASX releases: 1 June 2022 (Maiden MRE): https://announcements.asx.com.au/asxpdf/20220601/pdf/459jfk8g7x2mty.pdf and 29 March 2023 (MRE update): https://announcements.asx.com.au/asxpdf/20230329/pdf/45n49jlm02grm1.pdf Collar locations for drill holes are surveyed using DGPS. Three DD holes and 3 RC holes have hand-held GPS collar surveys. Historic Data: Historic drill hole data is archival, data cross checked with drill logs and available plans and sections where available. Collar locations have been checked by CEL using differential GPS (DGPS) to verify if the site coincides with a marked collar, tagged drill site or likely drill pad location. In most cases the drill collars coincide with historic drill site, some of which (but not all) are tagged. The collar check surveys were reported in POSGAR (2007) projection and converted to WGS84.
Drill sample recovery	- *Method of recording and assessing core and chip sample recoveries and results assessed.* - *Measures taken to maximise sample recovery and ensure representative nature of the samples.* - *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	Drill core is placed into wooden boxes by the drillers and depth marks are indicated on wooden blocks at the end of each run. These depths are reconciled by CEL geologists when measuring core recovery and assessing core loss. Triple tube drilling has been being done by CEL to maximise core recovery. - 761 CEL diamond drill holes completed have been used for the CEL resource estimate. Some of these holes are located outside the resource area. Total drilled is 224,180.60 metres, including cover drilled of 22,041.30 metres (9.8 %). Of the remaining 202,139.30 metres of bedrock drilled, core recovery is 96.8%. RC sub-samples are collected from a rotary splitter mounted to the face sample recovery cyclone. A 2-4 kg sub-samples is collected for each metre of RC drilling. Duplicate samples are taken at the rate of I every 25-30 samples using a riffle splitter to split out a 2-4 kg sub-sample. The whole sample recovered is weighed to measure sample recovery and consistency in sampling. - 37 CEL RC drill holes have been used in the CEL resource estimate. Total metres drilled is 2,923m. Cover drilled is 511 m (17.5%) Channel samples have been weighed to ensure a consistency between sample lengths and weights. The channel samples are collected from saw-cut channels and the whole sample is collected for analysis. There is no correlation between sample length and assay values. - 193 surface and underground channels have been used in the CEL resource estimate. Channels total 2597.70 metres in length. The average weight per metre sampled is 3.7 kg/m which is adequate

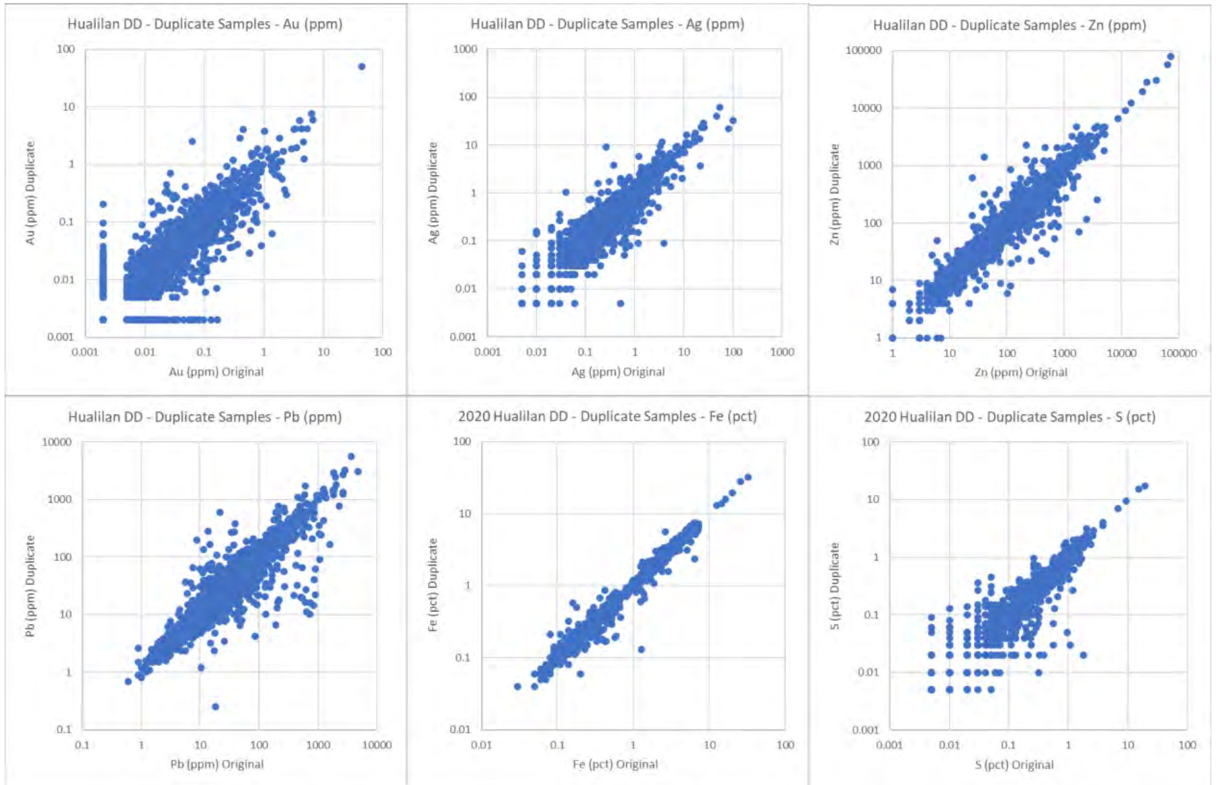
Criteria	JORC Code explanation	Commentary
		for the rock being sampled and compares well with the expected weight for ½ cut HQ3 drill core of 4.1 kg/m. A possible relationship has been observed in historic drilling between sample recovery and Au Ag or Zn values whereby low recoveries have resulted lower reported values. Historic core recovery data is incomplete. Core recovery is influenced by the intensity of natural fracturing in the rock. A positive correlation between recovery and RQD has been observed. The fracturing is generally post mineral and not directly associated with the mineralisation.
Logging	- *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation mining studies and metallurgical studies.* - *Whether logging is qualitative or quantitative in nature. Core (or costean channel etc) photography.* - *The total length and percentage of the relevant intersections logged.*	For CEL drilling, all the core (100%) is photographed and logged for recovery, RQD, weathering, lithology, alteration, mineralization, and structure to a level that is suitable for geological modelling, Mineral Resource Estimation and metallurgical test work. RC drill chips are logged for geology, alteration and mineralisation to a level that is suitable for geological modelling resource estimation and metallurgical test work. Where possible logging is quantitative. Geological logging is done into MS Excel in a format that can readily be cross-checked and is back-up transferred to a secure, offsite, cloud-based database which holds all drill hole logging sample and assay data. No specialist geotechnical logging has been undertaken. Detailed logs are available for most of the historical drilling. Some logs have not been recovered. No core photographs from the historic drilling have been found. No drill core has survived due to poor storage and neglect. No historic RC sample chips have been found.
Sub-sampling techniques and sample preparation	- *If core whether cut or sawn and whether quarter half or all core taken.* - *If non-core whether riffled tube sampled rotary split etc and whether sampled wet or dry.* - *For all sample types the nature quality and appropriateness of the sample preparation technique.* - *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* - *Measures taken to ensure that the sampling is representative of the in-situ material collected including for instance results for field duplicate/second-half sampling.* - *Whether sample sizes are appropriate to the grain size of the material being sampled.*	CEL samples have been submitted to the MSA laboratory in San Juan, the ALS laboratory in Mendoza and the former SGS laboratory in San Juan for sample preparation. The sample preparation technique is considered appropriate for the style of mineralization present in the Project. Sample sizes are appropriate for the mineralisation style and grain size of the deposit. Sample intervals are selected based on lithology, alteration, and mineralization boundaries. Representative samples of all of the core are selected. Sample length averages 1.74m. Second-half core or ¼ core samples have been submitted for a mineralised interval in 1 drill hole only and for some metallurgical samples. The second half of the core samples has been retained in the core trays for future reference. Competent drill core is cut longitudinally using a diamond saw for sampling of ½ the core. Softer core is split using a wide blade chisel or a manual core split press. The geologist logging the core, marks where the saw cut or split is to be made to ensure half-core sample representivity. From GNDD073 and later holes, duplicate core samples consisting of two ¼ core samples over the same interval have been collected approximately every 30-50m drilled. Duplicate core sample results and correlation plots (log scale for Au, Ag, Zn, Pb, Fe and S) are shown below:

	count	RSQ	mean		median		variance	
			original	duplicate	original	duplicate	original	duplicate
Au (ppm)	3,523	0.960	0.076	0.077	0.007	0.006	0.640	0.816

Criteria	JORC Code explanation	Commentary

Ag (ppm)	3,523	0.696	0.53	0.48	0.17	0.16	7.99	3.55
Cd (ppm)	3,523	0.979	1.34	1.26	0.08	0.08	160.63	144.11
Cu (ppm)	3,523	0.451	14.84	13.85	3.40	3.30	4.3E+03	2.5E+03
Fe (%)	3,523	0.990	1.997	1.996	1.700	1.710	3.74	3.75
Pb (ppm)	3,523	0.940	64.7	62.4	13.7	13.4	1.9E+05	2.7E+05
S (%)	3,523	0.973	0.333	0.330	0.140	0.140	0.346	0.332
Zn (ppm)	3,523	0.976	254	243	73	72	3.8.E+06	3.5.E+06

RSQ = R squared

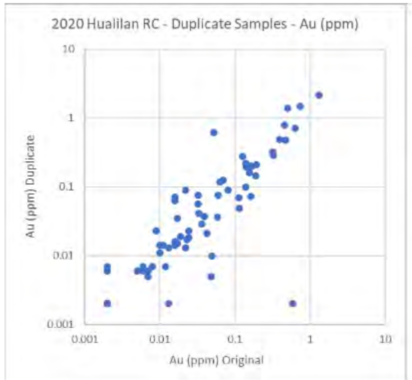


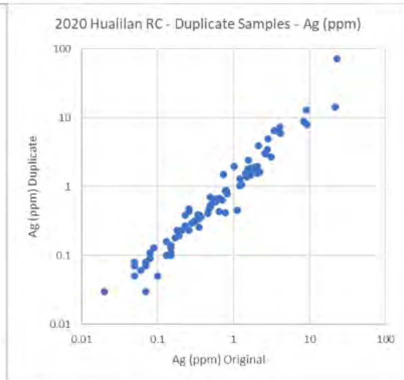
RC sub-samples over 1m intervals are collected at the drill site from a cyclone mounted on the drill rig. A duplicate RC sample is collected for every 25-30m drilled.

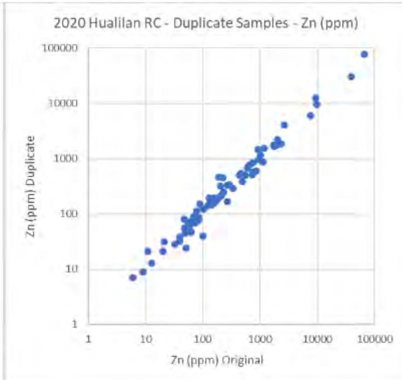
Criteria	JORC Code explanation	Commentary
		The duplicate RC sample results and correlation plots (log scale for Au, Ag, Zn, Pb, Fe and S) are shown below:

	count	RSQ	mean		median		variance	
			original	duplicate	original	duplicate	original	duplicate
Au (ppm)	85	0.799	0.101	0.140	0.017	0.016	0.041	0.115
Ag (ppm)	85	0.691	1.74	2.43	0.59	0.58	13.59	64.29
Cd (ppm)	85	0.989	15.51	16.34	0.41	0.44	4189	4737
Cu (ppm)	85	0.975	47.74	53.86	5.80	5.70	2.4E+04	3.1E+04
Fe (%)	85	0.997	1.470	1.503	0.450	0.410	7.6	7.6
Pb (ppm)	85	0.887	296.0	350.6	26.3	32.4	6.0E+05	7.4E+05
S (%)	85	0.972	0.113	0.126	0.020	0.020	0.046	0.062
Zn (ppm)	85	0.977	3399	3234	158	177	2.5.E+08	2.1.E+08

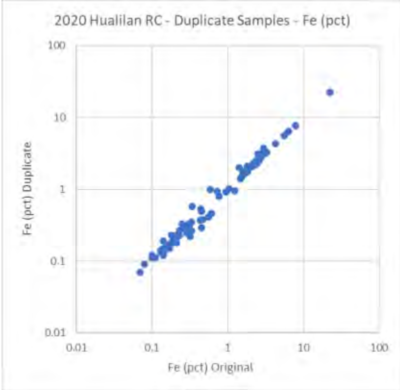
RSQ = R squared

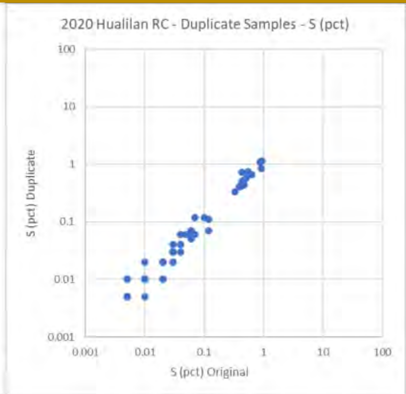


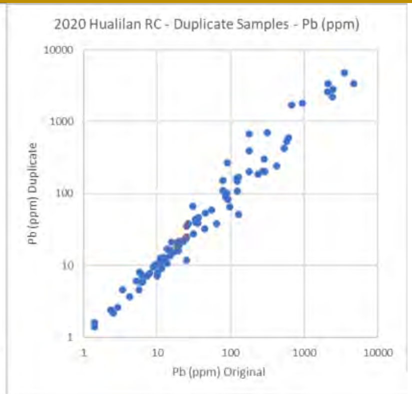




Criteria	JORC Code explanation	Commentary







45 duplicate channel sample assays have been collected from the underground and surface sampling program. These data show more scatter due to surface weathering.

The duplicate channel sample results and correlation plots (log scale for Au, Ag, Zn, Pb, Fe and S) are shown below:

	count	RSQ	mean		median		variance	
			original	duplicate	original	duplicate	original	duplicate
Au (ppm)	45	0.296	1.211	2.025	0.042	0.039	8.988	23.498
Ag (ppm)	45	0.037	8.42	23.25	1.09	1.22	177.31	3990.47
Cd (ppm)	45	0.373	124.23	77.85	7.54	7.80	61687.10	26171.51
Cu (ppm)	45	0.476	713.23	802.79	46.20	37.40	2.8E+06	3.0E+06
Fe (%)	45	0.428	4.266	5.745	1.390	1.560	44.4	107.0
Pb (ppm)	45	0.007	955.4	3776.0	75.3	60.7	3.5E+06	3.0E+08
S (%)	45	0.908	1.307	1.432	0.040	0.030	14.294	16.234
Zn (ppm)	45	0.509	15117	12684	1300	763	8.8.E+08	5.2.E+08

RSQ = R squared

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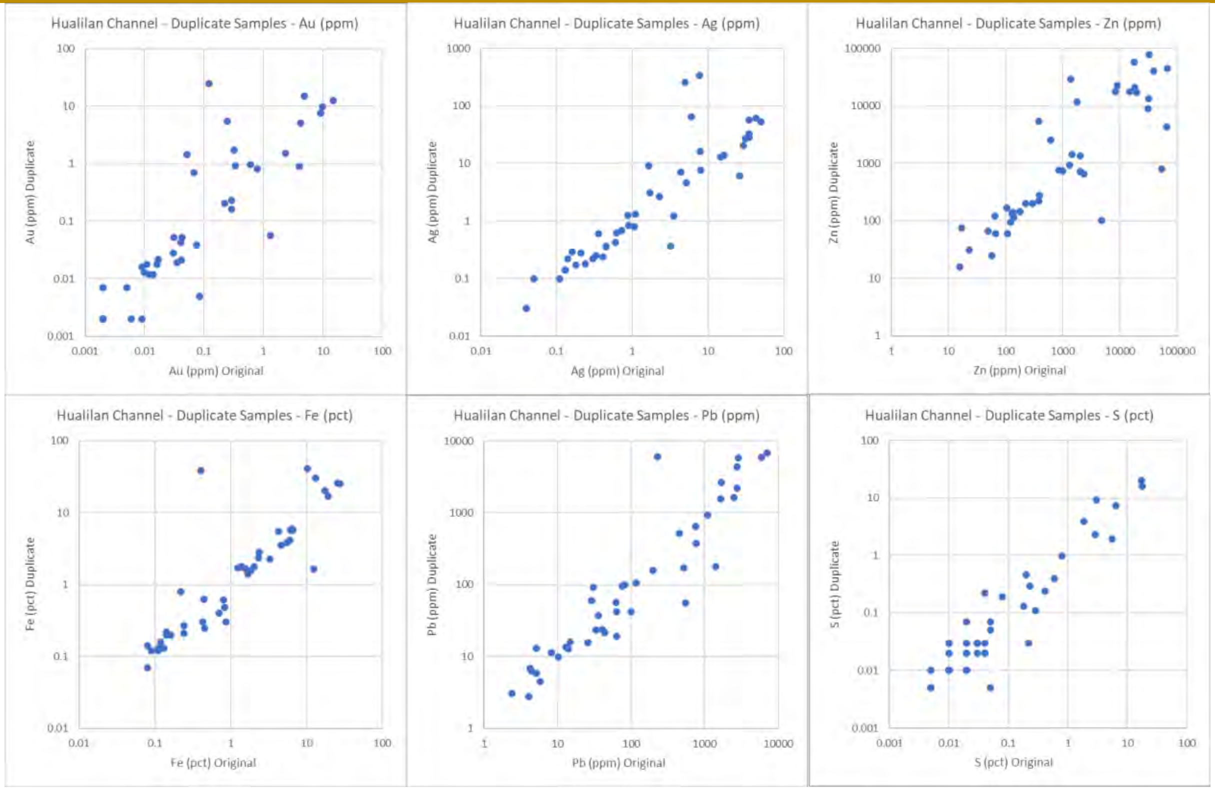
<tr><td></td><td></td><td>
</td></tr>
<tr><td>Quality of assay data and laboratory tests</td><td>- The nature quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.
- For geophysical tools spectrometers handheld XRF instruments etc the parameters used in determining the analysis including instrument make and model reading times</td><td>The MSA laboratory used for sample preparation in San Juan was inspected by Stuart Munroe (Exploration Manager) and Sergio Rotondo (CEL Director) prior to any samples being submitted. The laboratory has been visited and revied most recently by Stuart Munroe (Exploration Manager) in May 2022. The laboratory procedures are consistent with international best practice and are suitable for samples from the Project. The SGS laboratory in San Juan and the ALS laboratory in Mendoza has not yet been inspected by CEL representatives due to COVID-19 restrictions. Each laboratory presents internal laboratory standards for each job to gauge precision and accuracy of assays reported.

CEL have used two different blank samples, submitted with drill core and subjected to the same preparation and assay as the core samples, RC sub-samples and channel samples. The blank samples are sourced from surface gravels in the Las Flores area of San Juan and from a commercial dolomite quarry near San Juan. In both cases the blank material is commonly for construction. Commonly, the blank samples are strategically placed in the sample sequence immediately</td></tr>
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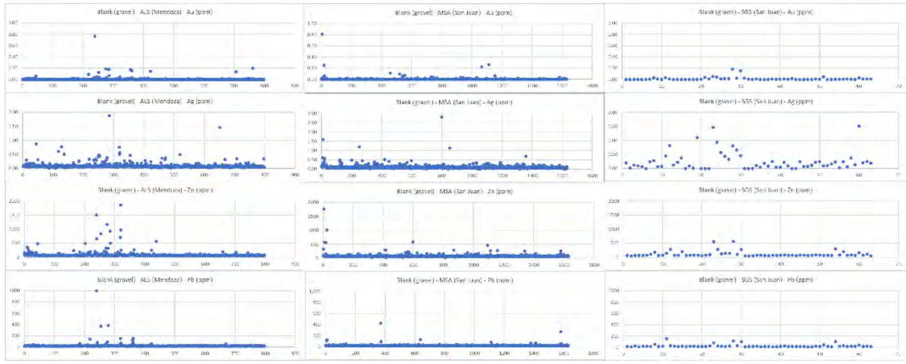
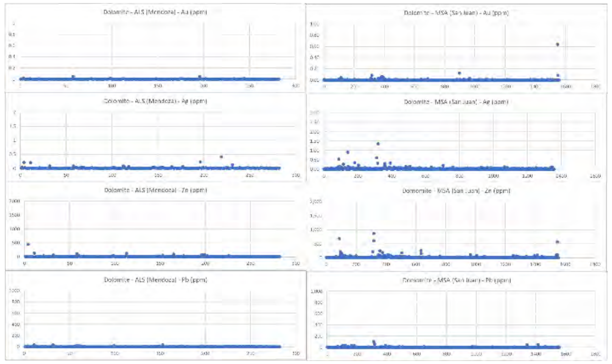
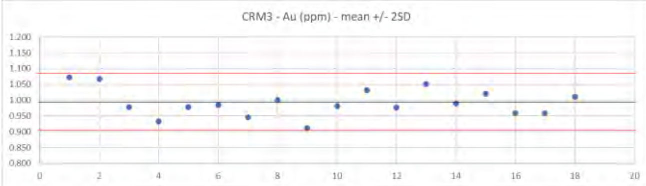
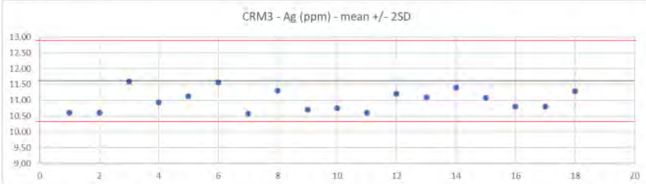
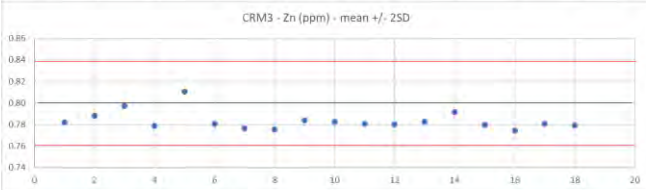
Criteria	JORC Code explanation	Commentary
	calibrations factors applied and their derivation etc. - *Nature of quality control procedures adopted (eg standards blanks duplicates external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	after samples that were suspected of containing higher grade Au, Ag, S or base metals to test the lab preparation and contamination procedures. The values received from the blank samples suggest only rare cross contamination of samples during sample preparation.   For GNDD001 – GNDD010 samples analysed by MSA in 2019, three different Certified Standard Reference pulp samples (CRM) with known values for Au Ag Pb Cu and Zn were submitted with samples of drill core to test the precision and accuracy of the analytic procedures MSA laboratory in Canada. 26 reference analyses were analysed in the samples submitted in 2019. The standards demonstrate suitable precision and accuracy of the analytic process. No systematic bias is observed.

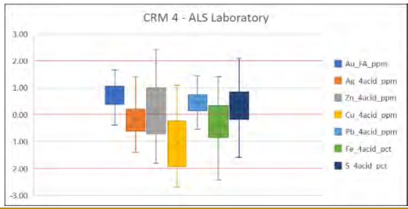
Criteria	JORC Code explanation	Commentary

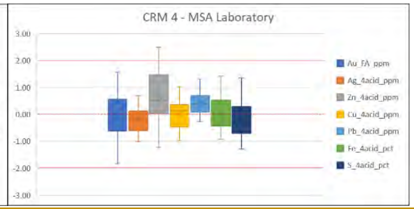




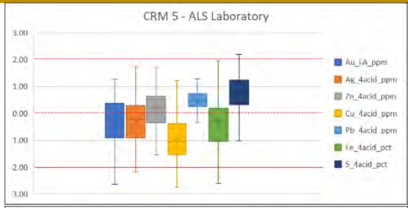


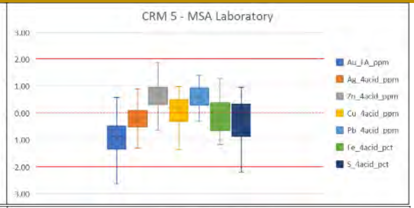
For drill holes from GNDD011 plus unsampled intervals from the 2019 drilling, 17 different multi-element Certified Standard Reference pulp samples (CRM) with known values for Au Ag Fe S Pb Cu and Zn. 7 different CRM's with known values for Au only have been submitted with samples of drill core, RC chips and channel samples to test the precision and accuracy of the analytic procedures of the MSA,ALS and SGS laboratories used. In the results received to date there has been no systematic bias is observed. The standards demonstrate suitable precision and accuracy of the analytic process. A summary of the standard deviations from the expected values for CRM's used is summarised below. Generally, an average of standard deviations close to zero indicates a high degree of accuracy and a low range of standard deviations with a low fail count indicates a high degree of precision.

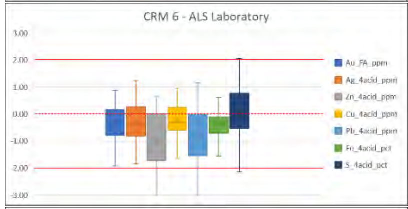


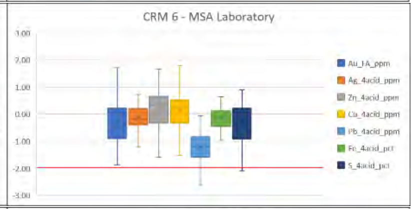


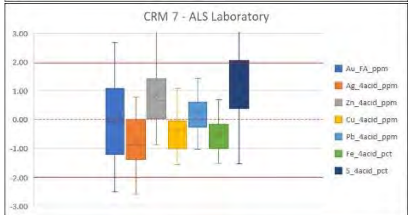
Criteria	JORC Code explanation	Commentary

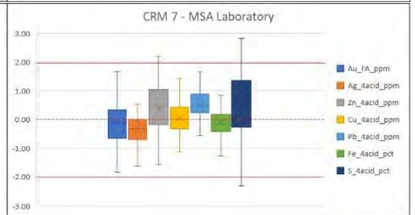


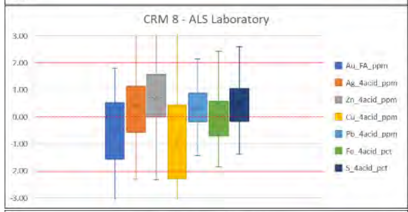


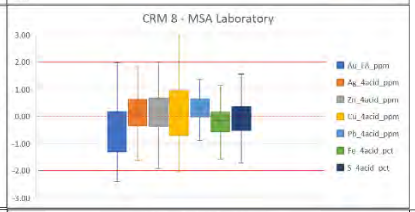


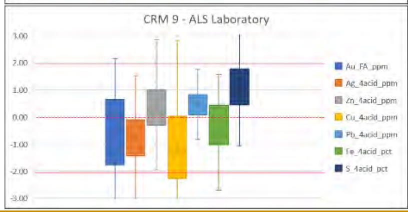


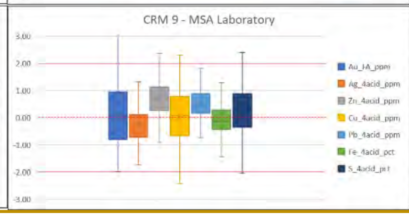


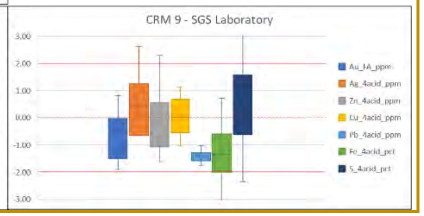












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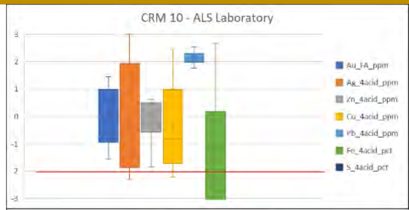

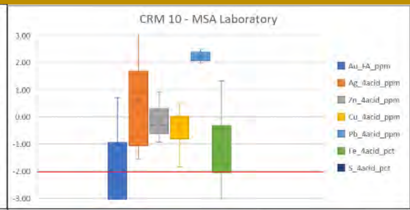

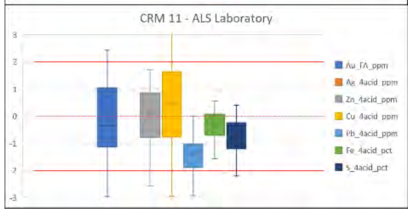

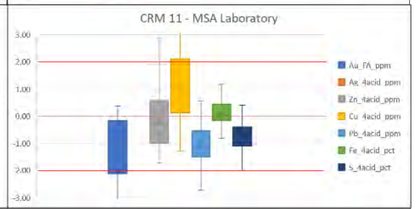

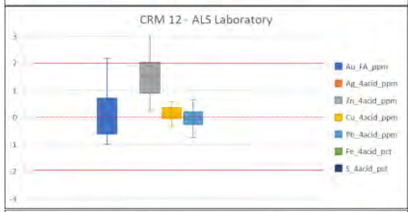

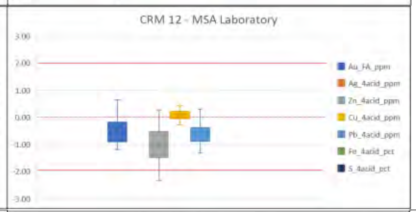

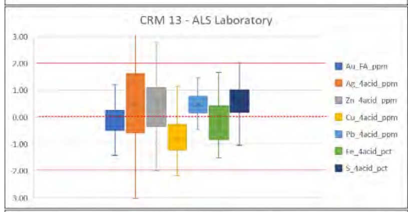

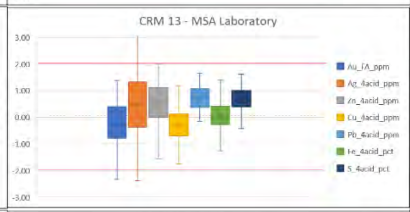

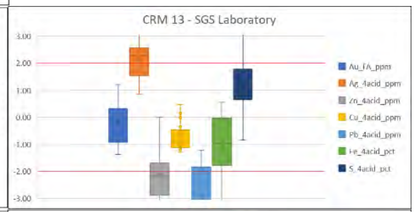

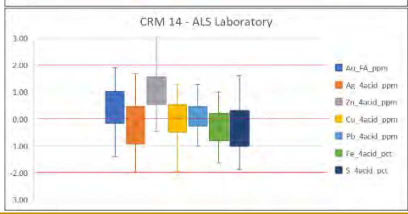

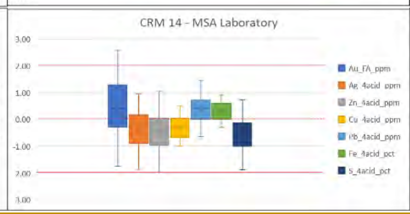

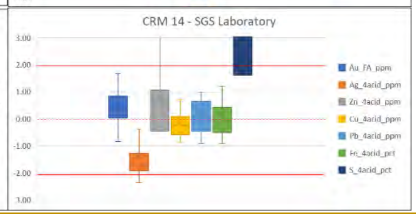

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		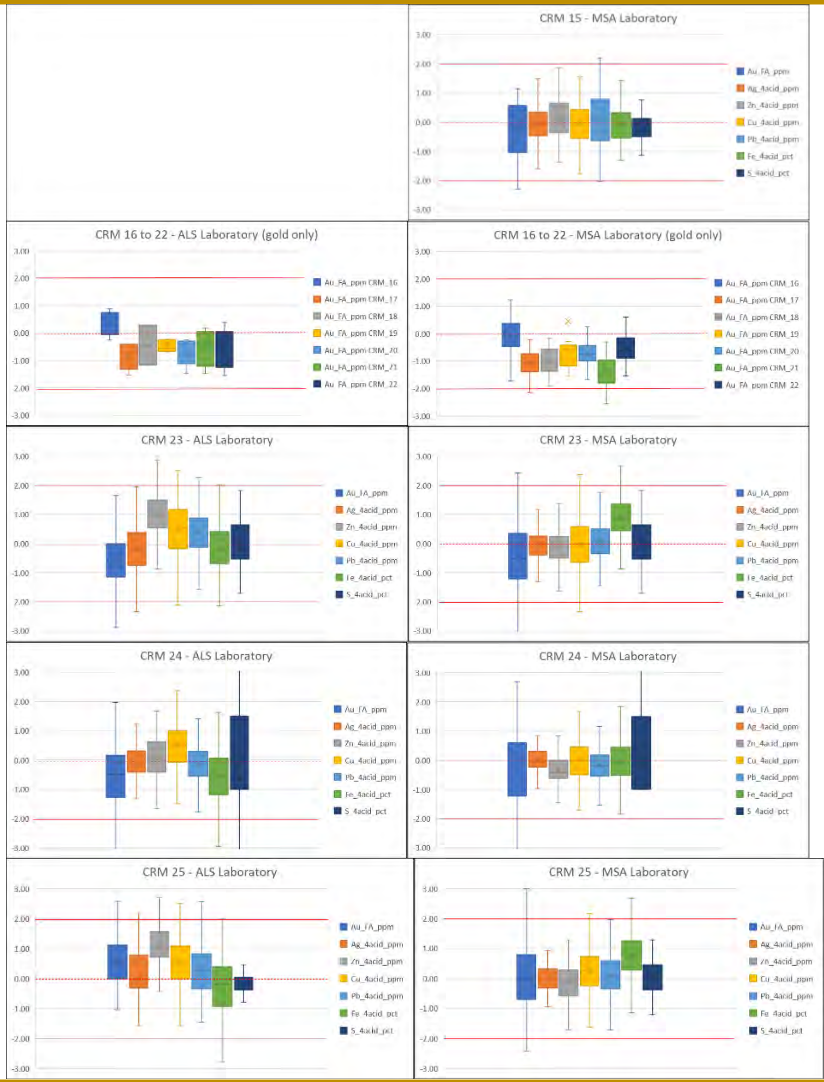

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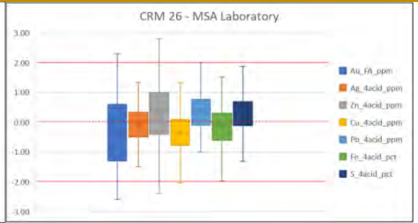
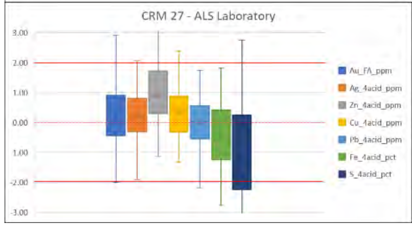
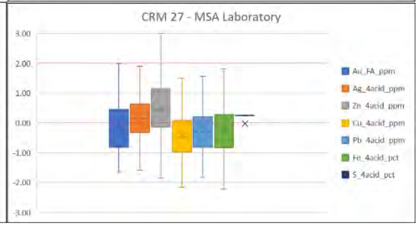






Verification of sampling and assaying

- *The verification of significant intersections by either independent or alternative company personnel.*
- *The use of twinned holes.*
- *Documentation of primary data entry procedures data verification data storage (physical and electronic) protocols.*
- *Discuss any adjustment to assay data.*

Final sample assay analyses are received by digital file in PDF and CSV format. There is no adjustment made to any of the assay values received. The original files are backed-up and the data copied into a cloud-based drill hole database, stored offsite from the project. The data is remotely accessible for geological modelling and resource estimation.

Assay results summarised in the context of this report have been rounded appropriately to 2 significant figures. No assay data have been otherwise adjusted. Replicate assay of 186 coarse reject samples from 2019 drilling has been done to verify assay precision. Original core samples were from the 2019 DD drilling which were analysed by MSA (San Juan preparation and Vancouver analysis). Coarse reject samples were analysed by ALS (Mendoza preparation and Vancouver analysis). The repeat analysis technique was identical to the original. The repeat analyses correlate very closely with the original analyses providing high confidence in precision of results between MSA and ALS. A summary of the results for the 186 sample pairs for key elements is provided below:

	Mean		**Median**		**Std Deviation**		
Element	**MSA**	**ALS**	**MSA**	**ALS**	**MSA**	**ALS**	**Correlation coefficient**
Au (FA and GFA ppm)	4.24	4.27	0.50	0.49	11.15	11.00	0.9972
Ag (ICP and ICF ppm)	30.1	31.1	5.8	6.2	72.4	73.9	0.9903
Zn ppm (ICP ppm and ICF %)	12312	12636	2574	2715	32648	33744	0.9997
Cu ppm (ICP ppm and ICF %)	464	474	74	80	1028	1050	0.9994
Pb ppm (ICP ppm and ICF %)	1944	1983	403	427	6626	6704	0.9997
S (ICP and ICF %)	2.05	1.95	0.05	0.06	5.53	5.10	0.9987

Criteria	JORC Code explanation	Commentary

Cd (ICP ppm)	68.5	68.8	12.4	12.8	162.4	159.3	0.9988
As (ICP ppm))	76.0	79.5	45.8	47.6	88.1	90.6	0.9983
Fe (ICP %)	4.96	4.91	2.12	2.19	6.87	6.72	0.9994
REE (ICP ppm)	55.1	56.2	28.7	31.6	98.2	97.6	0.9954

Cd values >1000 are set at 1000.
REE is the sum off Ce, La, Sc, Y. CE > 500 is set at 500. Below detection is set at zero

Replicate assay of 192 coarse reject samples from 2021 drilling has been done to verify assay precision. Original core samples were from the 2021 DD drilling which were analysed by SGS Laboratories (San Juan preparation and Lima analysis). Coarse reject samples were prepared and analysed by ALS (Mendoza preparation and Lima analysis). The repeat analysis technique was identical to the original. Except for Mo (molybdenum), the repeat analyses correlate closely with the original analyses providing confidence in precision of results between SGS and ALS. A summary of the results for the 192 sample pairs for key elements is provided below:

		Mean		Median		Std Deviation		
Element	count	SGS	ALS	SGS	ALS	SGS	ALS	Correlation coefficient
Au (FA and GFA ppm)	192	1.754	1.680	0.432	0.441	20.8	21.5	0.9837
Ag (ICP and ICF ppm)	192	12.14	11.57	0.93	1.03	7085	5925	0.9995
Zn (ICP and ICF ppm)	192	6829	7052	709	685	4.54E+08	5.34E+08	0.9942
Cu (ICP and ICF ppm)	192	203.4	202.9	25.7	24.5	3.30E+05	3.35E+05	0.9967
Pb (ICP and ICF ppm)	192	1768	1719	94.7	91.6	5.04E+07	4.39E+07	0.9959
S (ICP and ICF %)	192	2.23	2.10	0.94	0.87	16.51	15.56	0.9953
Cd (ICP ppm)	192	43.9	42.4	4.1	4.0	19594	18511	0.9956
As (ICP ppm))	192	45.4	45.2	16.0	16.9	10823	9893	0.9947
Fe (ICP %)	189	3.07	3.30	2.38	2.31	4.80	9.28	0.9781
REE (ICP ppm)	192	63.5	72.8	39.4	44.3	3414	4647	0.9096
Mo (ICP and ICF ppm)	192	7.69	1.68	6.74	0.97	85.83	10.33	0.3026

Values below detection were set to half the detection limit
Limit of detection for Fe was exceeded for 3 samples submitted to SGS with no overlimit analysis
REE is the sum off Ce, La, Sc, Y. Vaues below detection were set at zero.

Replicate assay of 140 pulp reject samples from the 2022 drill (parts of drill holes GNDD654 and GNDD666) was done to check assay precision. The original pulps were analysed by MSA laboratories (San Juan preparation and Vancouver, Canada analysis). Replicate pulps were analysed by ALS (Lima, Peru). The analytic techniques were identical at both laboratories.

Criteria	JORC Code explanation	Commentary
		(table below)

Element	count	Mean SGS	Mean ALS	Median SGS	Median ALS	Std Deviation SGS	Std Deviation ALS	Correlation coefficient
Au (FA ppm)	140	0.27	0.30	0.01	0.02	0.98	1.05	0.9829
Ag (ICP ppm)	140	1.16	1.14	0.16	0.16	6.15	6.31	0.9965
Zn (ICP ppm)	140	555	565	50	56	2471	2469	0.9996
Pb (ICP ppm)	140	92.3	95.4	13.6	13.5	338	351	0.9977
S (ICP %)	140	0.64	0.61	0.17	0.17	1.22	1.12	0.9982
Fe (ICP %)	140	1.62	1.59	0.64	0.66	1.91	1.88	0.9991

CEL have sought to twin and triplicate some of the historic and recent drill holes to check the results of previous exploration. A preliminary analysis of the twin holes indicates similar widths and grades for key elements assayed. The twin holes are:

GNDD003 – DDH34 and 04HD08
GNRC110 – DDH53
GNDD144 – GNDD021 – 05HD39
GNRC107 – GNDD008/008A
GNDD206 – DDH54
GNDD421 – GNDD424

Criteria	JORC Code explanation	Commentary
Location of data points	- *Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys) trenches mine workings and other locations used in Mineral Resource estimation.* - *Specification of the grid system used.* - *Quality and adequacy of topographic control.*	Following completion of drilling, collars are marked and surveyed using a differential GPS (DGPS) relative to a nearby Argentinian SGM survey point. The collars have been surveyed in POSGAR 2007 zone 2 and converted to WGS84 UTM zone 19s. Following completion of the channel sampling, the location of the channel samples is surveyed from a survey mark at the entrance to the underground workings, located using differential GPS. The locations have been surveyed in POSGAR 2007 zone 2 and converted to WGS84 UTM zone 19s. The drill machine is set-up on the drill pad using hand-held survey equipment according to the proposed hole design. Diamond core drill holes up to GNDD390 are surveyed down-hole at 30-40m intervals down hole using a down-hole compass and inclinometer tool. RC drill holes and diamond core holes from GNDD391 were continuously surveyed down hole using a gyroscope to avoid magnetic influence from the drill string and rocks. The gyroscope down-hole survey data is recorded in the drill hole database at 10m intervals. Ten diamond drill holes have no down hole survey data due to drill hole collapse or blockage of the hole due to loss of drilling equipment. These are GNDD036, 197, 212, 283, 376, 423, 425, 439, 445 and 465. For these holes, a survey of the collar has been used with no assumed deviation to the end of the hole. All current and previous drill collar sites, Minas corner pegs and strategic surface points have been surveyed using DGPS to provide topographic control for the Project. In addition, AWD3D DTM model with a nominal 2.5 metre precision has been

Criteria	JORC Code explanation	Commentary
		acquired for the project and greater surrounding areas. Drone-based topographic survey data with 0.1 meter precision is being acquired over the project to provide more detail where required.
Data spacing and distribution	- *Data spacing for reporting of Exploration Results.* - *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* - *Whether sample compositing has been applied.*	Nominal 80m x 80m, 40m x 80m and 40m x 40m drill spacing is being applied to the drilling to define mineralised areas to Indicated Resource level of confidence, where appropriate. Drilling has been completed to check previous exploration, extend mineralisation along strike, and provide some information to establish controls on mineralization and exploration potential. Samples have not been composited.
Orientation of data in relation to geological structure	- *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known considering the deposit type.* - *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias this should be assessed and reported if material.*	As far as is currently understood and where practicable, the orientation of sampling achieves unbiased sampling of structures and geology controlling the mineralisation. Some exploration holes have drilled at a low angle to mineralisation and have been followed up with drill holes in the opposite direction to define mineralised domains. For underground channel sampling, the orientation of the sample is determined by the orientation of the workings. Where the sampling is parallel with the strike of the mineralisation, plans showing the location of the sampling relative to the orientation of the mineralisation, weighted average grades and estimates of true thickness are provided to provide a balanced report of the mineralisation that has been sampled. Drilling has been designed to provide an unbiased sample of the geology and mineralisation targeted. In exceptional circumstances, where drill access is restricted, drilling may be non-optimally angled across the mineralised zone.
Sample security	- *The measures taken to ensure sample security.*	Samples were under constant supervision by site security, senior technical personnel and courier contractors prior to delivery to the preparation laboratories in San Juan and Mendoza.
Audits or reviews	- *The results of any audits or reviews of sampling techniques and data.*	There has not yet been any independent reviews of the sampling techniques and data.

Section 2 Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	- *Type reference name/number location and ownership including agreements or material issues with third parties such as joint ventures partnerships overriding royalties native title interests historical sites wilderness or national park and environmental settings.* - *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	The Hualilan Project comprises fifteen Minas (equivalent of mining leases) and five Demasias (mining lease extensions) held under an farmin agreement with Golden Mining SRL (Cerro Sur) and CIA GPL SRL (Cerro Norte). Fourteen additional Minas and eight exploration licences (Cateos) have been transferred to CEL under a separate farmin agreement. Six Cateos and eight requested mining leases are directly held. This covers all of the currently defined mineralization and surrounding prospective ground. There are no royalties held over the tenements.

Granted mining leases (Minas Otorgadas) at the Hualilan Project

Name	Number	Current Owner	Status	Grant Date	Area (ha)
Cerro Sur					
Divisadero	5448-M-1960	Golden Mining S.R.L.	Granted	30/04/2015	6
Flor de Hualilan	5448-M-1960	Golden Mining S.R.L.	Granted	30/04/2015	6
Pereyra y Aciar	5448-M-1960	Golden Mining S.R.L.	Granted	30/04/2015	6
Bicolor	5448-M-1960	Golden Mining S.R.L.	Granted	30/04/2015	6
Sentazon	5448-M-1960	Golden Mining S.R.L.	Granted	30/04/2015	6
Muchilera	5448-M-1960	Golden Mining S.R.L.	Granted	30/04/2015	6
Magnata	5448-M-1960	Golden Mining S.R.L.	Granted	30/04/2015	6
Pizarro	5448-M-1960	Golden Mining S.R.L.	Granted	30/04/2015	6
Cerro Norte					
La Toro	5448-M-1960	CIA GPL S.R.L.	Granted	30/04/2015	6
La Puntilla	5448-M-1960	CIA GPL S.R.L.	Granted	30/04/2015	6
Pique de Ortega	5448-M-1960	CIA GPL S.R.L.	Granted	30/04/2015	6
Descrubidora	5448-M-1960	CIA GPL S.R.L.	Granted	30/04/2015	6
Pardo	5448-M-1960	CIA GPL S.R.L.	Granted	30/04/2015	6
Sanchez	5448-M-1960	CIA GPL S.R.L.	Granted	30/04/2015	6
Andacollo	5448-M-1960	CIA GPL S.R.L.	Granted	30/04/2015	6

Mining Lease extensions (Demasias) at the Hualilan Project

Name	Number	Current Owner	Status	Grant date	Area (ha)
Cerro Sur					
North of "Pizarro" Mine	195-152-C-1981	Golden Mining S.R.L.	Granted	29/12/1981	2.42
Cerro Norte					

Criteria	JORC Code explanation	Commentary

South of "Andacollo" Mine	545.208-B-94	CIA GPL S.R.L.	Pending Reconsideration	14/02/1994	1.83
South of "Sanchez" Mine	545.209-B-94	CIA GPL S.R.L.	Registered	14/02/1994	3.50
South of "La Toro" Mine	195-152-C-1981	CIA GPL S.R.L.	Granted	29/12/1981	2.42
South of "Pizarro" Mine	545.207-B-94	Golden Mining S.R.L.	Registered	14/02/1994	2.09

Requested Mining Leases (Minas Solicitados)

Name	Number	Status	Area (ha)
Elena	1124.328-G-2021	Registered	2,799.24
Juan Cruz	1124.329-G-2021	Granted	933.69
Paula (over "Lo Que Vendra")	1124.454-G-2021	Application	1,460.06
Argelia	1124.486-G-2021	Registered	3,660.50
Ana Maria (over Ak2)	1124.287-G-2021	Registered	5,572.80
Erica (Over "El Peñón")	1124.541-G-2021	Application	6.00
Silvia Beatriz (over “AK3”)	1124.572-G-2021	Application	2,290.75
Soldado Poltronieri (over 1124188-20, 545867-R-94 and 545880-O-94)	1124.108-2022	Application	777.56

Mining Lease Farmin Agreements

Name	Number	Transfrred to CEL	Status	Area (ha)
Marta Alicia	2260-S-58	In Process	Granted	23.54
Marta	339.154-R-92	In Process	Granted	478.50
Solitario 1-5	545.604-C-94	In Process	Application	685.00
Solitario 1-4	545.605-C-94	In Process	Registered	310.83
Solitario 1-1	545.608-C-94	In Process	Application	TBA
Solitario 6-1	545.788-C-94	In Process	Application	TBA
AGU 3	11240114-2014	No	Granted	1,500.00
AGU 5	1124.0343-2014	No	Granted	1,443.58
AGU 6	1124.0623-2017	No	Granted	1,500.00
AGU 7	1124.0622-S-17	No	Granted	1,500.00
Guillermina	1124.045-S-2019	No	Granted	2,921.05
El Petiso	1124.2478-71	No	Granted	18.00
Ayen/Josefina	1124.495-I-20	No	Granted	2059.6

Criteria	JORC Code explanation	Commentary
		Exploration Licence (Cateo) Farmin Agreements

Name	Number	Transfrred to CEL	Status	Area (ha)
-	295.122-R-1989	In process	Registered	1,882.56
-	338.441-R-1993	In process	Granted	2,800.00
-	545.880-O-1994	In process	Registered	149.99
-	414.998-2005	Yes	Granted	721.90
-	1124.011-I-07	No	Granted	2552
-	1124.012-I-07	No	Registered	6677
-	1124.013-I-07	No	Granted	5818
-	1124.074-I-07	No	Granted	4484.5

Exploration Licence (Cateo) Held (Direct Award)

Name	Number	Transfrred to CEL	Status	Area (ha)
-	1124-248G-20	Yes	Current	933.20
-	1124-188-G-20 (2 zones)	Yes	Current	327.16
-	1124.313-2021	Yes	Current	986.41
-	1124.564-G-2021	Yes	Current	1,521.12
-	1124.632-G-2022	Yes	Current	4,287.38

There are no known impediments to obtaining the exploration licenses or operating the Project.

Criteria	JORC Code explanation	Commentary
Exploration done by other parties	- *Acknowledgment and appraisal of exploration by other parties.*	Intermittent historic sampling has produced a large volume of information and data including sampling, geological maps, reports, trenching data, underground surveys, drill hole results, geophysical surveys, non-JORC resource estimates plus property examinations and detailed studies by multiple geologists. Prior to exploration by CEL, no work has been completed on the Project since 2006. There is at least 6 km of underground workings that pass through mineralised zones at Hualilan. Surveys of the workings are likely to be incomplete. Commonly incomplete records of the underground geology and sampling have been compiled and digitised as has sample data geological mapping adit exposures and drill hole results. Historic geophysical surveys exist but have been superseded by surveys completed by CEL. Historic drilling on or near the Hualilan Project (Cerro Sur and Cerro Norte combined) extends to over 150 drill holes. The key historical exploration drilling and sampling programs are: - 1984 – Lixivia SA channel sampling & 16 RC holes (AG1-AG16) totalling 2,040m - 1995 - Plata Mining Limited (TSE: PMT) 33 RC holes (Hua- 1 to 33) + 1,500 RC chip samples - 1998 – Chilean consulting firm EPROM (on behalf of Plata Mining) systematic underground mapping and channel sampling

Criteria	JORC Code explanation	Commentary
		- 1999 – Compania Mineral El Colorado SA ("CMEC") 59 diamond core holes (DDH-20 to 79) plus 1,700m RC program - 2003 – 2005 – La Mancha (TSE Listed) undertook 7,447m of DDH core drilling (HD-01 to HD-48) - Detailed resource estimation studies were undertaken by EPROM Ltd. (EPROM) in 1996 and CMEC (1999 revised 2000) both of which are well documented and La Mancha 2003 and 2006. The collection of all exploration data by the various operators was of a high standard and appropriate sampling techniques intervals and custody procedures were used. Not all the historic data has been archived and so there are gaps in the availability of the historic data.
Geology	- *Deposit type geological setting and style of mineralisation.*	Mineralisation occurs in all rock types where it preferentially replaces limestone, shale and sandstone and occurs in fault zones and in fracture networks within dacitic intrusions. The mineralisation is Zn-(Pb-Cu-Ag) distal skarn (or manto-style skarn) overprinted with vein-hosted mesothermal to epithermal Au-Ag mineralisation. It has been divided into three phases – prograde skarn, retrograde skarn and a later quartz-rich mineralisation consistent with the evolution of a large hydrothermal system. Precise mineral paragenesis and hydrothermal evolution is the subject of on-going work which is being used for exploration and detailed geometallurgical test work. Gold occurs in native form as inclusions with sulphide (predominantly pyrite) and in pyroxene. The mineralisation commonly contains pyrite, chalcopyrite sphalerite and galena with rare arsenopyrite, pyrrhotite and magnetite. Mineralisation is either parallel to bedding in bedding-parallel faults, in veins or breccia matrix within fractured dacitic intrusions, at lithology contacts or in east-west striking steeply dipping siliceous faults that cross the bedding at a high angle. The faults have thicknesses of 1–4 metres and contain abundant sulphides. The intersection between the bedding-parallel mineralisation and east-striking cross veins seems to be important in localising the mineralisation. Complete oxidation of the surface rock due to weathering is thin. A partial oxidation / fracture oxidation layer near surface is 1 to 40m thick and has been modelled from drill hole intersections.
Drill hole Information	- *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* - *easting and northing of the drill hole collar* - *elevation or RL (Reduced Level – elevation above sea level in metres) of*	Significant intersections previous reported for historic drill holes, DD drill holes, RC drill holes completed by CEL are detailed in CEL ASX releases: 1 June 2022 (Maiden MRE): https://announcements.asx.com.au/asxpdf/20220601/pdf/459jfk8g7x2mty.pdf and 29 March 2023 (MRE update): https://announcements.asx.com.au/asxpdf/20230329/pdf/45n49jlm02grm1.pdf A cut-off grade of 1 g/t Au equivalent has been used with up to 2m of internal diltion or a cut-off grade of 0.2 g/t Au equivalent and up to 4m of internal diltion has been allowed. No metallurcial or recovery factors have been used in the intersections reported.

Criteria	JORC Code explanation	Commentary
	the drill hole collar - *dip and azimuth of the hole* - *down hole length and interception depth* - *hole length.* - *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report the Competent Person should clearly explain why this is the case.*	
Data aggregation methods	- *In reporting Exploration Results weighting averaging techniques maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated.* - *Where aggregate intercepts incorporate short lengths of high-grade results and longer lengths of low-grade results the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* - *The assumptions used for any reporting of metal equivalent values should be clearly stated.*	Weighted average significant intercepts are reported to a gold grade equivalent (AuEq). Results are reported to cut-off grade of a 1.0 g/t Au equivalent and 10 g/t Au equivalent allowing for up to 2m of internal dilution between samples above the cut-off grade and 0.2 g/t Au equivalent allowing up to 10m of internal dilution between samples above the cut-off grade. The following metals and metal prices have been used to report gold grade equivalent (AuEq): Au US$ 1780 / oz Ag US$24 /oz and Zn US$ 2800 /t. Metallurgical recoveries for Au, Ag and Zn have been estimated from the results of interim metallurgical test work completed by SGS Metallurgical Operations in Lakefield, Ontario using a combination of gravity and flotation of a combined metallurgical sample from 5 drill holes. Using data from the interim test results, and for the purposes of the AuEq calculation for drill hole significant intercepts, gold recovery is estimated at 89%, silver at 84% and zinc at 79%. Accordingly, the formula used is AuEq (g/t) = Au (g/t) + [Ag (g/t) x (24/1780) x (0.84/0.89)] + [Zn (%) x (28.00*31.1/1780) x (0.79/0.89)]. Metallurgical test work and geological and petrographic descriptions suggest all the elements included in the metal equivalents calculation have reasonable potential of eventual economic recovery. While Cu and Pb are reported in the table above as they were not yet considered economically significant at the time of the interim metallurgical test results, these metals were not used in the Au equivalent calculation at this early stage of the Project. No top cuts have been applied to the reported grades.
Relationship between mineralisation widths and intercept lengths	- *These relationships are particularly important in the reporting of Exploration Results.* - *If the geometry of the mineralisation with respect to the drill hole angle is known its nature should be reported.* - *If it is not known and only the down hole lengths are reported there should be a clear statement to this effect (eg 'down*	The mineralisation is moderately or steeply dipping and strikes NNE and ENE. For some drill holes, there is insufficient information to confidently establish the true width of the mineralized intersections at this stage of the exploration program. Apparent widths may be thicker in the case where the dip of the mineralisation changes and/or bedding-parallel mineralisation intersects NW or ENE-striking cross faults and veins. Representative cross section interpretations have been provided periodically with releases of significant intersections to allow estimation of true widths from individual drill intercepts.

Criteria	JORC Code explanation	Commentary
	hole length true width not known').	
Diagrams	- *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Representative maps and sections are provided in the body of reports released to the ASX.
Balanced reporting	- *Where comprehensive reporting of all Exploration Results is not practicable representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	All available final data have been reported where possible and plans of all drilling with results.
Other substantive exploration data	- *Other exploration data if meaningful and material should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density groundwater geotechnical and rock characteristics; potential deleterious or contaminating substances.*	Geological context and observations about the controls on mineralisation where these have been made are provided in the body of the report. Specific gravity measurements have been taken from the drill core recovered during the drilling program. These data are used to estimate densities in Resource Estimates. Eight Induced Polarisation (IP) lines have been completed in the northern areas of the Project. Stage 1 surveying was done on 1 kilometre length lines oriented 115° azimuth, spaced 100m apart with a 50m dipole. The initial results indicate possible extension of the mineralisation with depth. Stage 2 surveying was done across the entire field on 1 – 3 kilometre length lines oriented 090°, spaced 400m apart with a 50m dipole. On-going data interpretation is being done as drilling proceeds. Two ground magnetic surveys and a drone magnetic survey have been completed. The results of these data and subsequent geological interpretations are being used to guide future exploration.
Further work	- *The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling).* - *Diagrams clearly highlighting the areas of possible extensions including the main geological interpretations and future drilling areas provided this information is not commercially sensitive.*	• CEL Plans to undertake the following over the next 12 months • Additional resource extension, infill and exploration drilling; • Geophysical tests for undercover areas. • Structural interpretation and alteration mapping using high resolution satellite data and geophysics to better target extensions of known mineralisation. • Field mapping program targeting extensions of known mineralisation. • Further metallurgical test work.

Section 3 Estimation and Reporting of Mineral Resources

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	- *Measures taken to ensure that data has not been corrupted by for example transcription or keying errors between its initial collection and its use for Mineral Resource estimation purposes.* - *Data validation procedures used.*	Geological logging completed by previous explorers was done on paper copies and transcribed into a series of excel spreadsheets. These data have been checked for errors. Checks have been made against the original logs and with follow-up twin and close spaced drilling. Only some of the historic drill holes have been used in the Resource Estimate, including the results presented in Section2. Some drill holes have been excluded where the geology indicates that the drill hole is likely mis-located or where the drill hole has been superseded by CEL drilling. For CEL drilled holes, assay data is received in digital format. Backup copies are backed up into a cloud-based file storage system and the data is entered into a drill hole database which is also securely backed up off site. The drill hole data is backed up and is updated periodically by the CEL GIS and data management team.
Site visits	- *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* - *If no site visits have been undertaken indicate why this is the case.*	The Competent Person has undertaken site visits during exploration. Site visits were undertaken in 2019 and 2020 before COVID-19 closed international travel. Post COVID numerous site visits have undertaken since November 2021. The performance of the drilling program, collection of data, sampling procedures, sample submission and exploration program were initiated and reviewed during these visits.
Geological interpretation	- *Confidence in (or conversely the uncertainty of) the geological interpretation of the mineral deposit.* - *Nature of the data used and of any assumptions made.* - *The effect if any of alternative interpretations on Mineral Resource estimation.* - *The use of geology in guiding and controlling Mineral Resource estimation.* - *The factors affecting continuity both of grade and geology.*	The geological interpretation is considered appropriate given the drill core density of data that has been collected, access to mineralisation at surface and underground exposures. Given the data, geological studies past and completed by CEL, the Competent Person has a high level of confidence in the geological model that has been used to constrain the mineralised domains. It is assumed that networks of fractures controlled by local geological factors have focussed hydrothermal fluids and been the site of mineralisation in both the prograde zinc skarn and retrograde mesothermal – epithermal stages of hydrothermal evolution. The interpretation captures the essential geometry of the mineralised structure and lithologies with drill data supporting the findings from the initial underground sampling activities. Mineralised domains have been built using explicit wireframe techniques from 0.2 – 0.5 g/t AuEq mineralised intersections, joined between holes by the instruction from the geology and structure. Continuity of grade between drill holes is determined by the intensity of fracturing, the host rock contacts (particularly dacite – limestone contacts) and by bedding parallel faults, particularly within limestone, at the limestone and overlying sedimentary rock contact and within the lower sequences of the sedimentary rocks within 40m of the contact. No alternative interpretations have been made form which a Mineral Resource Estimate has been made.
Dimensions	- *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise) plan width and depth below surface to the upper and lower limits of the Mineral*	31 separate domains were interpreted over a strike length of 2.3kms. The domains vary in width and orientation from 2m up to 100m in width. The deepest interpreted domain extends from the surface down approximately 600m below surface.

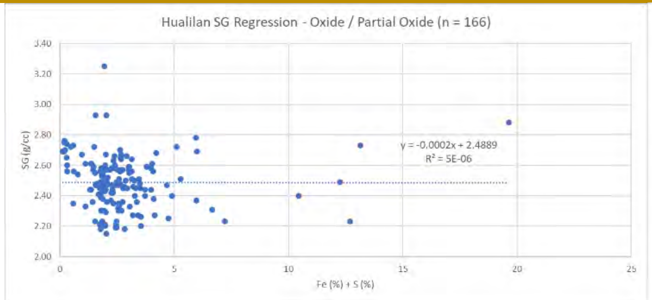
Criteria	JORC Code explanation	Commentary
	Resource.	
Estimation and modelling techniques	- *The nature and appropriateness of the estimation technique(s) applied and key assumptions including treatment of extreme grade values domaining interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* - *The availability of check estimates previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.* - *The assumptions made regarding recovery of by-products.* - *Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).* - *In the case of block model interpolation the block size in relation to the average sample spacing and the search employed.* - *Any assumptions behind modelling of selective mining units.* - *Any assumptions about correlation between variables.* - *Description of how the geological interpretation was used to control the resource estimates.* - *Discussion of basis for using or not using grade cutting or capping.* - *The process of validation the checking process used the comparison of model data to drill hole data and use of reconciliation data if available*	Estimation was made for Au Ag, Zn and Pb being the elements of economic interest. Estimate was also made for Fe and S being the elements that for pyrite which is of economic and metallurgical interest and is also used to estimate the density for bocks in the Mineral Resource Estimate. No previous JORC Resource estimates or non-JORC Foreign Resource estimates were made with similar methods to compare to the current Resource estimate. No production records are available to provide comparisons. A 2m composite length was selected after reviewing the original sample lengths from the drilling which showed an average length of 1.54m for samples taken within the mineralised domains. A statistical analysis was undertaken on the sample composites top cuts for Au, Ag, Zn and Pb composites on a domain-by-domain basis. The domains were then grouped by host rock and mineralisation style and group domain top cuts were applied in order to reduce the influence of extreme values on the resource estimates without downgrading the high-grade composites too severely. The top-cut values were chosen by assessing the high-end distribution of the grade population within each group and selecting the value above which the distribution became erratic. The following table shows the top cuts applied to each group and domain for Au, Ag, Zn and Pb. No top cut was applied to estimation of Fe and S. <table><tr><th>Group</th><th>Au (ppm)</th><th>Ag (ppm)</th><th>Zn (%)</th><th>Pb (%)</th></tr><tr><td>Fault Zone hosted (Magnata and Sanchez) and CAL (limestone) hosted</td><td>80</td><td>300</td><td>20</td><td>5</td></tr><tr><td>LUT (siltstone) hosted</td><td>20</td><td>100</td><td>5</td><td>1</td></tr><tr><td>DAC (intrusive) hosted</td><td>15</td><td>70</td><td>5</td><td>1.8</td></tr></table> Block modelling was undertaken in Surpac™ V6.6 software. A block model was set up with a parent cell size of 10m (E) x 20m (N) x 10m (RL) with standard sub-celling to 2.5m (E) x 5.0m (N) x 2.5m (RL) to maintain the resolution of the mineralised domains. The 20m Y and vertical block dimensions were chosen to reflect drill hole spacing and to provide definition for potential mine planning. The shorter 10m X dimension was used to reflect the geometry and orientation of the majority of the domain wireframes. Group Variography was carried out using Leapfrog Edge software on the two metre composited data from each of the 31 domains for each variable.

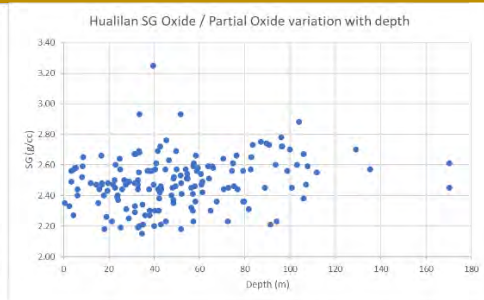
Criteria	JORC Code explanation	Commentary
		All relevant variables; Au, Ag, Pb, Zn, Fe and S in each domain were estimated using Ordinary Kriging using only data from within that domain. The orientation of the search ellipse and variogram model was controlled using surfaces designed to reflect the local orientation of the mineralized structures. An oriented "ellipsoid" search for each domain was used to select data for interpolation. A 3 pass estimation search was conducted, with expanding search ellipsoid dimensions and decreasing minimum number of samples with each successive pass. First passes were conducted with ellipsoid radii corresponding to 40% of the complete range of variogram structures for the variable being estimated. Pass 2 was conducted with 60% of the complete range of variogram structures for the variable being estimated. Pass 3 was conducted with dimensions corresponding to 200% of the semi-variogram model ranges. Blocks within the model where Au was not estimated during the first 3 passes were assigned as unclassified. Blocks for Ag, Pb, Zn, Fe and S that were not estimated were assigned the average values on a per-domain basis. Validation checks included statistical comparison between drill sample grades and Ordinary Kriging block estimate results for each domain. Visual validation of grade trends for each element along the drill sections was also completed in addition to swath plots comparing drill sample grades and model grades for northings, eastings and elevation. These checks show good correlation between estimated block grades and drill sample grades.
Moisture	- *Whether the tonnages are estimated on a dry basis or with natural moisture and the method of determination of the moisture content.*	Tonnage is estimated on a dry basis.
Cut-off parameters	- *The basis of the adopted cut-off grade(s) or quality parameters applied.*	The following metals and metal prices have been used to report gold grade equivalent (AuEq): Au US$ 1900 / oz, Ag US$24 /oz, Zn US$ 4,000 /t and Pb US 2,000/t. Average metallurgical recoveries for Au, Ag, Zn and Pb have been estimated from the results of Stage 1 metallurgical test work completed by SGS Metallurgical Operations in Lakefield, Ontario using a combination of gravity and flotation combined metallurgical samples as detailed in the Criteria below. For the AuEq calculation average metallurgical recovery is estimated as 94.9% for gold, 90.9% for silver, 67.0% for Zn and 57.8% for Pb. Accordingly, the formula used for Au Equivalent is: AuEq (g/t) = Au (g/t) + [Ag (g/t) x (24/1900) x (0.909/0.949)] + [Zn (%) x (40.00*31.1/1900) x (0.670/0.949)] + (Pb (%) x 20.00*31.1/1900) x (0.578/.9490}. Based on the break-even grade for an optimised pit shell for gold equivalent, a AuEq cut-off grade of 0.30 ppm is used to report the resource within an optimised pit shell run at a gold price of US$1,800 per ounce and allowing for Ag, Zn and Pb credits. Under this scenario, blocks with a grade above the 0.30 g/t Au Eq cut off are considered to have reasonable prospects of mining by open pit methods. A AuEq cut-off grade of 1.0 ppm was used to report the resource beneath the optimised pit shell run as these blocks are considered to have reasonable prospects of future mining by underground methods.

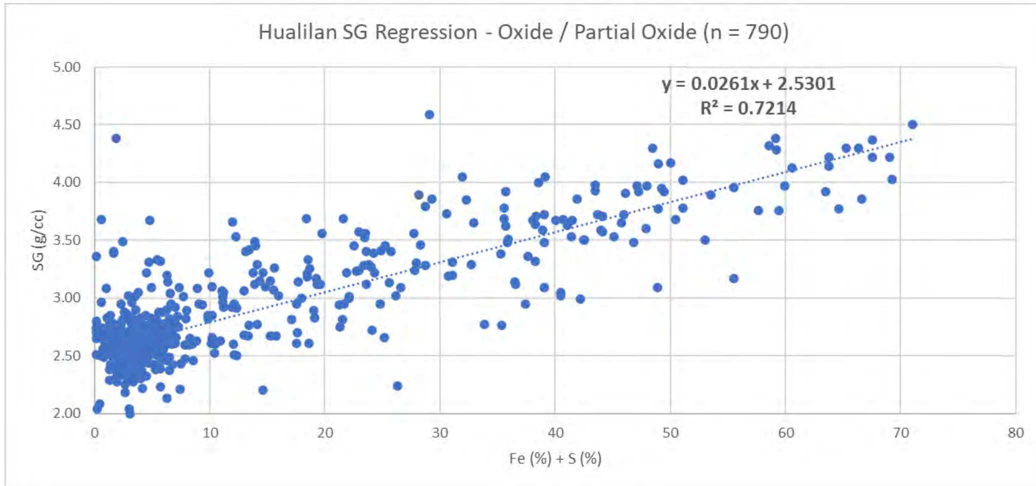
Criteria	JORC Code explanation	Commentary
Mining factors or assumptions	- *Assumptions made regarding possible mining methods minimum mining dimensions and internal (or if applicable external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case this should be reported with an explanation of the basis of the mining assumptions made.*	The Resource estimate has assumed that near surface mineralisation would be amenable to open pit mining given that the mineralisation is exposed at surface and under relatively thin unconsolidated cover. A surface mine optimiser has been used to determine the proportion of the Resource estimate model that would be amenable to eventual economic extraction by open pit mining methods. The surface mine optimiser was bult using the following parameters with prices in USD: - Au price of $1,800 per oz, Ag price of $23.4 per oz, Zn price of $3,825 per tonne and Pb price of $1,980 per tonne - Average metallurgical recoveries of 94.9% for Au, 90.9 % for Ag and 67 % for Zn and 57.8 % for Pb. - Ore and waste mining cost of $2.00 per tonne - Unconsolidated cover removal cost of $0.10 per tonne - Processing cost of $10.00 per tonne - Transport and marketing of $50 / oz of AuEq (road to Jan Juan then rail to Rosario Port) - Royalty of $60 per oz Au, 3% for Ag, Zn and Pb. - Assumed concentrate payability of 94.1% for Au, 82.9% for Ag, 90 % for Zn and 95 % for Pb. - 45° pit slopes on the western side of the pit and 55° on the eastern side of the pit Blocks above a 0.30 g/t AuEq within the optimised open pit shell are determined to have reasonable prospects of future economic extraction by open pit mining and are included in the Resource estimate on that basis. Blocks below the open pit shell that are above 1.0 g/t AuEq are determined to have reasonable prospects of future economic extraction by underground mining methods and are included in the Resource estimate on that basis.
Metallurgical factors or assumptions	- *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case this should be reported with an explanation of the basis of the metallurgical assumptions made.*	CEL has completed Stage 1 metallurgical test work on representative composite sample of mineralisation from: 1. Two separate composite samples of limestone-hosted massive sulphide (manto) Sample A has a weighted average grade of 10.4 g/t Au, 31.7 g/t Ag, 3.2 % Zn and 0.46 % Pb. Sample B has a weighted average grade of 9.7 g/t Au, 41.6 g/t Ag, 4.0% Zn and 0.48% Pb. 2. One dacite (intrusive) composite sample with a weighted average grade of 1.1 g/t Au, 8.1 g/t Ag and 0.10 % Zn and 0.04% Pb. 3. One sediment hosted (fine grained sandstone and siltstone) composite sample with a weighted average grade of 0.68 g/t Au, 7.5 g/t Ag, 0.34 % Zn and 0.06 % Pb. 4. One oxidised limestone (manto oxide) composite sample with a weighted average grade of 7.0 g/t Au, 45 g/t Ag, 3.7% Zn and 0.77% Pb. Gravity recovery and sequential flotation tests of the higher-grade limestone hosted mineralisation involved; 1. primary P80 = 51 micron primary grind, 2. gravity recovery, 3. Pb-Cu followed by Zn rougher flotation,

Criteria	JORC Code explanation	Commentary
		4. p80 = 29 micron regrind of the Zn rougher concentrate, 5. two re-cleaning stages of the Pb/Cu rougher concentrate, 6. four re-cleaning Sages on the Zn rougher concentrate, and 7. additional gravity recovery stages added to the Zn Rougher concentrate This results in the following products that are likely to be saleable - Au-Ag concentrate (118 g/t Au, 286 g/t Ag) with low deleterious elements, - Pb concentrate (65% Pb, 178 g/t Au, 765 g/t Ag) with low deleterious elements, and - Zn concentrate (51% Zn, 10 g/t Au, 178 g/t Ag) with low deleterious elements, relatively high Cd, but at a level that is unlikely to attract penalties. - tailing grades of 2 to 3 g/t Au which respond to intensive cyanide leach with recoveries of 70-80% of any residual gold and silver to a gold doré bar. Gravity recovery and flotation tests of the intrusive-hosted mineralisation involved; 1. primary P80 = 120-80 micron primary grind, 2. gravity recovery, 3. single stage rougher sulphide flotation, 4. P80 = 20-30 micron regrind of the rougher concentrate (5-10% mass), 5. one or two re-cleaning stages of the Au-Ag Rougher concentrate At primary grind of p80 = 76 micron and regrind of p80 = 51 micron an AuAg concentrate can be produced grading 54 g/t Au and 284 g/t Ag with total recoveries of 97% (Au) and 85% (Ag). One test of a sediment hosted composite sample (5-10% of the mineralisation at the Project) was a repeat of the testing done on the intrusive-hosted mineralisation. This produced an Au-Ag concentrate grading 23.6 g/t Au and 234 g/t Ag at total recoveries of 85% (Au) and 87% (Ag). Further test work is likely to be done as part of more detailed studies. It is likely that the concentrate produced from the sediment-hosted mineralisation will be combined with the Au-Ag concentrate from the limestone and intrusive-hosted mineralisation. Applying recoveries of 70% for both gold and silver to the various concentrate tailings components where leaching is likely to be undertaken during production generates recoveries of: ▪ 95% (Au), 93% (Ag), 89% (Zn), 70% (Pb) from the high-grade skarn (manto) component of the mineralisation; ▪ 96% (Au) and 88% (Ag) from the intrusion-hosted component of the mineralisation; ▪ 85% (Au) and 87% (Ag) from the sediment-hosted component of the mineralisation; An intensive cyanide leach test of oxide (limestone and dacite hosted mineralisation has produced recoveries of 78% (Au) and 64% (Ag) which is expected to be recovered into gold doré bar. While the oxide component of

Criteria	JORC Code explanation	Commentary
		the mineralisation comprises only a small percentage of the Hualilan mineralisation its lies in the top 30-40 metres and would be mined early in the case of an open pit operation. Based on the test work to date and the proportions of the various mineralisation types in the current geological model, it is expected that overall average recoveries for potentially saleable metals will be: - 94.9% Au, - 90.9% for Ag - 67.0% for Zn and - 57.8% for Pb Additional Stage 2 work involving comminution and variability testing, blended test work, and pilot plant testing is ongoing and planned.
Environmental factors or assumptions	- *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts particularly for a greenfields project may not always be well advanced the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	It is considered that there are no significant environmental factors which would prevent the eventual extraction of gold from the project. Environmental surveys and assessments have been completed in the past and will form a part of future pre-feasibility studies.
Bulk density	- *Whether assumed or determined. If assumed the basis for the assumptions. If determined the method used whether wet or dry the frequency of the measurements the nature size and representativeness of the samples.* - *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs porosity etc) moisture and differences between rock and*	CEL has collected specific gravity (SG) measurements from drill core, which have been used to estimate block densities for the Resource estimate. Within the mineralised domains there are 956 SG measurements made on drill core samples of 0.1 – 0.2 metres length. Measurements we determined on a dry basis by measuring the difference in sample weight in water and weight in air. For porous samples, the weight in water was measured after wrapping the sample so that no water enters the void space during weighing. In oxidised and partially oxidised rocks, SG clusters around an average of 2.49 g/cc (2,490 kg/m3) which is independent of depth. A density of 2,490 kg/m3 has been used for oxidised, fracture oxidised and partially oxidised blocks.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td>alteration zones within the deposit.
- Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.</td><td>



In fresh rock samples, a regression model for block density determination has been made by plotting assay interval Fe (%) + S (%) from the interval where the SG measurement was made against the SG measurement. Fe and S are the two elements that form pyrite which is the mineral that is commonly associated with gold and base metal mineralisation at Hualilan. SG plotted against (Fe+S) follows a linear trend within the mineralised domains for oxide and fresh rock as shown below.


For fresh rock at zero Fe + S (%) the density is assumed to be 2,530 kg/m³ (2.52 g/cc). The regression slope has a linear increase in density of 26.1 kg/m³ (0.0261 g/cc) for each 1 percent increase in Fe + S (%). The formula used for block density (kg/m³) determination in oxide rock is 2,530 + 26.1 x (Fe % + S%).</td></tr>
<tr><td>Classification</td><td>- The basis for the classification of the Mineral Resources into varying confidence categories.</td><td>The Mineral Resource has been classified based on the guidelines specified in the JORC Code. As a guide to reasonable prospects for economic extraction, the classification level is based upon semi-qualitative assessment of</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	- *Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations reliability of input data confidence in continuity of geology and metal values quality quantity and distribution of the data).* - *Whether the result appropriately reflects the Competent Person's view of the deposit.*	the geological understanding of the deposit, geological and mineralisation continuity, drill hole spacing, QC results, search and interpolation parameters, analysis of available density information and possible mining methods. The estimation search strategy was undertaken in three separate passes with different search distances, and the minimum number of samples used to estimate a block which were then used as a guide for the classification of the resource into Indicated, Inferred and Unclassified. The classification was then further modified to restrict the Indicated Resource to the domains with closer spaced drilling. The potential open pit resource was constrained within an optimised pit shell run using a gold price of US$1,800 per ounce. Resources reported inside the pit shell were reported above a AuEq cut-off grade of 0.3 g/t and Resources outside the pit shell were reported above a AuEq cut-off grade of 1.0 g/t. Scoping study results have indicated that underground mining and open pit mining are both possible allowing for classification of Indicated and Inferred Mineral Resources throughout the estimation. The Competent Person has reviewed the result and determined that these classifications are appropriate given the confidence in the geology, data, results from drilling and possible mining methods as detailed in the scoping study.
Audits or reviews	- *The results of any audits or reviews of Mineral Resource estimates.*	The Mineral Resource estimate has not been independently audited or reviewed.
Discussion of relative accuracy/ confidence	- *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits or if such an approach is not deemed appropriate a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* - *The statement should specify whether it relates to global or local estimates and if local state the relevant tonnages which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* - *These statements of relative accuracy and confidence of the estimate should be compared with production data where available.*	There is sufficient confidence in the data quality drilling methods and analytical results that they can be relied upon. The available geology and assay data correlate well. The approach and procedure is deemed appropriate given the confidence limits. The main factors which could affect relative accuracy are: - domain boundary assumptions - orientation - grade continuity - top cut. Grade continuity is variable in nature in this style of deposit and has not been demonstrated to date and closer spaced drilling is required to improve the understanding of the grade continuity in both strike and dip directions. It is noted that the results from the twinning of three holes by La Mancha are encouraging in terms of grade repeatability. The deposit contains very high grades and there is need for the use of top cuts. No production data is available for comparison.

Appendix 2 – Source of Project Positioning Data

Company Name	ASX Code	ASIC ($AUD/oz)	Current Production (koz)	Capex ($AUDM)	Forecast Production (koz)	Capital Intensity ($AUD/oz)	Original Data Source	CEL Source Verification
Lion One Metals	LLO	930		106	78	1,361	Gold Nerds Sept Update	PEA Tuvutu Gold Project - 29 April 2022
Adriatic Metals	ADT	988		75	176	425	Gold Nerds Sept Update	Definitive Feasibility Study - 18 September 2021
Bellevue Gold	BGL	1,050		130	201	647	Gold Nerds Sept Update	Feasibility Study 2 - 2 September 2021
Black Dragon Gold	BDG	1,193		151	79	1,909	Gold Nerds Sept Update	Preliminary Economic Assessment of the Salave Gold Project. Completed in Q1 2019.
MetalsTech	MTC	1,196		103	67	1,539	Gold Nerds Sept Update	Surtec Gold Mine Scoping Study - 3 August 2022
Medallion Metals	MM8	1,203		163	86	1,895	Gold Nerds Sept Update	Ravensthorpe Gold Project PFS - 23 October 2023
Horizon Gold	HRN	1,209		62	60	1,033	Gold Nerds Sept Update	Gum Creek Scoping Study - 20 November 2019
Emerald Resources	**EMR**	**1,230**	**107**				Gold Nerds Sept Update	September 2023 Quarterly Report
Matador Mining	MZZ	1,231		162	88	1,839	Gold Nerds Sept Update	Cape Ray Gold Project Scoping Study - 6 May 2020
Geopacific Resources	GPR	1,239		248	75	3,311	Gold Nerds Sept Update	Woodlard Gold project Execution Update - 30 November 2020
Challenger Gold	CEL	1,276		220	141	1,560		Source CEL Scoping Study 2023
De Grey Mining	DEG	1,295		1,298	530	2,449	Gold Nerds Sept Update	Hemi Gold project DFS - 28 Sept 2023
Theta Gold Mines	TGM	1,323		90	67	1,357	Gold Nerds Sept Update	Definitive Feasibility Study for Phase 1 of the TGME Underground Gold Project - July 2022
Evolution Mining	**EVN**	**1,370**	**649**				Gold Nerds Sept Update	September 2023 Quarterly Report
Capricorn Metals	**CMM**	**1,376**	**120**				Gold Nerds Sept Update	September 2023 Quarterly Report
Kin Mining	KIN	1,442		77	51	1,510	Gold Nerds Sept Update	Leonora Gold project Feasibility Study - 2 Oct 2017
Tietto Minerals	**TIE**	**1,468**	**44**				Gold Nerds Sept Update	September 2023 Quarterly Report
Antipa Minerals	AZY	1,475		275	170	1,618	Gold Nerds Sept Update	MINYARI DOME PROJEC Scoping Study - August 2022
Investigator Resources	IVR	1,489		131	49	2,700	Gold Nerds Sept Update	Pioneer Dome Scoping Study - 7 Feb 2023
OreCorp	ORR	1,514		752	203	3,700	Gold Nerds Sept Update	Definitive feasibility Study - 5 September 2022
Toubani Resources	TRE	1,542		263	90	2,926	Gold Nerds Sept Update	Toubani DFS - 29 September 2021
Ausgold	AUC	1,549		297	136	2,184	Gold Nerds Sept Update	Katanning Gold Project Revised Scoping Study 22 may 2023
West Wits Mining	WWI	1,550		86	52	1,648	Gold Nerds Sept Update	Scoping Study - 9 March 2022
Rox Resources	RXL	1,568		134	71	1,887	Gold Nerds Sept Update	Youanmi Gold project Scoping Study - 19 Oct 2022

Sihayo Gold	SIH	1,598		351	75	4,654	Gold Nerds Sept Update	Sihayo Gold Project Definitive Feasibility Study - 23 June 2020
Gold Road Resources	**GOR**	**1,600**	**158**				Gold Nerds Sept Update	September 2023 Quarterly Report
Astral Resources	AAR	1,648		191	100	1,910	Gold Nerds Sept Update	Lady Colleen Scoping Study - 15 February 2023
Alkane Resources	**ALK**	**1,650**	**70**				Gold Nerds Sept Update	September 2023 Quarterly Report
Ramelius Resources	**RMS**	**1,650**	**243**				Gold Nerds Sept Update	September 2023 Quarterly Report
Perseus Mining	**PRU**	**1,666**	**482**				Gold Nerds Sept Update	September 2023 Quarterly Report
Resources & Energy Group	REZ	1,681					Gold Nerds Sept Update	September 2023 Quarterly Report
Meeka Metals	MEK	1,684		137	80	1,713	Gold Nerds Sept Update	Murchinson Gold project Feasibility Study - 12 July 2023
Tesoro Gold	TSO	1,695		209	93	2,252	Gold Nerds Sept Update	Phase 1 Scoping Study - 4 April 2023
West African Resources	**WAF**	**1,698**	**191**				Gold Nerds Sept Update	September 2023 Quarterly Report
Northern Star Resources	**NST**	**1,760**	**1,563**				Gold Nerds Sept Update	September 2023 Quarterly Report
Nova Minerals	NVA	1,823		611	132	4,628	Gold Nerds Sept Update	Phase 2 Scoping Study - 15 May 2023
Regis Resources	**RRL**	**1,870**	**458**				Gold Nerds Sept Update	September 2023 Quarterly Report
Black Cat Syndicate	BC8	1,892		34	42	817	Gold Nerds Sept Update	Noosa Gold Presentation (Paulsens, Coyote, Kal east PFS's) 8 July 2023
Kingsgate	**KCN**	**1,893**	**45**				Gold Nerds Sept Update	September 2023 Quarterly Report
Westgold Resources	**WGX**	**1,900**	**257**				Gold Nerds Sept Update	September 2023 Quarterly Report
SSR Mining	**SSR**	**1,910**	**523**				Gold Nerds Sept Update	September 2023 Quarterly Report
Silver Lake Resources	**SLR**	**1,950**	**260**				Gold Nerds Sept Update	September 2023 Quarterly Report
Red 5	**RED**	**1,975**	**194**				Gold Nerds Sept Update	September 2023 Quarterly Report
Calidus Resources	**CAI**	**2,000**	**60**				Gold Nerds Sept Update	September 2023 Quarterly Report
Brightstar Resources	BTR	2,041		22	45	489	Gold Nerds Sept Update	Menzies and Laverton Mine Restart Study - 6 September 2023
Genesis Minerals	**GMD**	**2,113**	**136**				Gold Nerds Sept Update	September 2023 Quarterly Report
Pantoro Resources	**PNR**	**2,300**	**54**				Gold Nerds Sept Update	September 2023 Quarterly Report
Ora Banda Mining	**OBM**	**2,300**	**53**				Gold Nerds Sept Update	September 2023 Quarterly Report
St Barbara	**SBM**	**2,300**	**124**				Gold Nerds Sept Update	September 2023 Quarterly Report
Resolute Mining	**RSG**	**2,348**	**330**				Gold Nerds Sept Update	September 2023 Quarterly Report
Catalyst Metals	**CYL**	**2,643**	**77**				Gold Nerds Sept Update	September 2023 Quarterly Report