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Ollachea Gold Project – NI 43-101 Technical Report

(Preliminary Economic Assessment)

Puno Region, Peru

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Minera IRL LIMITED

Minera Kuri Kullu S.A.

Effective Date: August 27th, 2021

Report Date: August 27th, 2021

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Minera IRL LIMITED	**Ollachea Gold Project – NI 43-101 Technical Report**	**REVISION**	
		No.	DATE
	Preliminary Economic Assessment	00	27/08/21

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Ollachea Gold Project – NI 43-101 Technical Report (Preliminary Economic Assessment)

Prepared for Minera IRL Limited and Minera Kuri Kullu S.A.

Effective Date: August 27th, 2021

Report Date: August 27th, 2021

(Original Signed)

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1 SUMMARY

1.1 Property, Access and Permits

The Property is located in southeast Peru and Region of Puno, approximately equidistant from the cities of Cusco and Juliaca.

The Property consists of 18 contiguous mining concessions ("Concession Minera"), some concessions partially overlap and a gap is recorded between Oyaechea [sic] concessions 1, 2, and 3 (Figure 4-2). Considering overlaps and gaps, the total footprint of the property is approximately 9899 hectares.

In 2006, Minera IRL entered in to a 30-year agreement with the Ollachea Farming Community, to allow access to the Minapampa and Minapampa Far East areas of the Property, the main areas of economic interest considered in the PEA.

An Environmental Impact Assessment ("EIA") for the property has been approved and is valid throughout the life of the mine. The EIA considers an underground mine and 3000 tonnes per day (tpd) processing plant, consisting of crushing, milling, gravity concentration, leaching and desorption processes for producing doré bars.

All permits required prior to applying for authorization to mine have been acquired (i.e., explosives, water discharge). These permits are valid for a given period and require regular renewal. MIRL intends to apply for permissions to commence mining following completion of the PEA.

1.2 Geology and Mineral Resources

The MRE relates to the Minapampa Zone, and the Minapampa Far East Zone ("MFE") of the Property. These zones are within the Ollachea 3 mining concession and are entirely covered by the community agreement reported in Section 4 of the Report. It is important to note that mineralization extends beyond the Ollachea 3 mining concession into an area held by a third party. This portion of mineralization has been excluded from the MRE Statement.

The MRE has been based on a subset of the drilling data (the drill hole database) reported in Section 10 of the Technical Report. Drill holes not in the Minapampa or Minapampa Far East zones of the Property, and drill holes without downhole survey data have been excluded from the MRE. The subset of drilling data includes 192 diamond drill holes (166 in Minapampa, and 26 DDH in Minapampa Far East) and totalling 70,151.75 m of drilled core.

Dr. Fowler (QP) has undertaken a visual comparison of block model sections against drill traces; a review of statistics; and undertaken check estimates, and he is satisfied that the MRE is consistent with the CIM mineral resource, mineral reserve estimation best practice guidelines.

The MRE for Ollachea, with an effective date of June 30, 2021, has been constrained by optimised underground stope shapes and is reported at a cut-off grade of 1.4 g/t Au. The MRE has been categorized in accordance with the CIM Definition Standards (CIM, 2014) and comprises an Indicated and Inferred Mineral Resource summarised in Table 1-1.

Table 1-1: Mineral Resource Estimate for the Ollachea Project by classification and Zone

Mineral Resource Estimate for the Ollachea Project - June 30, 2021						
Zone	Indicated			Inferred		
	Tonnes (Mt)	Au g/t	Au Ounces (Moz)	Tonnes (Mt)	Au g/t	Au Ounces (Moz)
Minapampa	10.7	3.28	1.13	1.8	3.0	0.2
Minapampa Far East	-	-	-	5.5	2.6	0.5
Total	10.7	3.28	1.13	7.3	2.7	0.6

1. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
2. All figures are rounded to reflect the relative accuracy of the estimates.
3. The Mineral Resource was estimated by Ms. Muñoz and supervised by Dr. A. Fowler, MAusIMM, CP(Geo), Independent Qualified Person under NI 43-101., of Mining Plus Consultants who takes responsibility for it.
4. Composite gold grades were capped where appropriate.
5. Mineral Resources are diluted and are reported within optimized underground stope shapes.
6. The stope shapes were optimized at a gold cut-off value of 1.4 grams per tonne, considering metal prices of US$1700 per ounce of gold, and assuming metal recovery of 87% for gold, and total operating costs of $61.18/t.
7. Tonnages reported are metric tonnes and ounces of contained gold are troy ounces.
8. Mining Plus is not aware of any environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues that could materially affect the potential development of the Mineral Resource Estimate.

1.3 Mining and Mine Plan

Bottom-up Long Hole Open Stoping (“LHOS”) with paste-fill is considered as the optimal mining method for the Property. LHOS is defined as a moderate production, non-entry, bulk mining method most applicable to large, regular mineralized bodies.

Stopes will be accessed longitudinally (along strike) on each level by, one, two or three strike drives, dependent on lode thickness.

Mining Plus was commissioned to re-evaluate the geotechnical parameters, specifically stable stoping spans and ground support requirements. Based on the review of available geotechnical information, the rock mass conditions appear to be generally favourable, with relatively high Rock Quality Designation (“RQD”) numbers, low inflows, and largely unaltered rock. The rock mass conditions in the mineralized zones and the immediate hanging wall are amenable to the LHOS with paste fill.

Mining Plus was retained to consider the viability of a low-CAPEX start-up for Ollachea, with a carbon in leach ("CIL") plant designed to treat 1,500 tonnes per day ("tpd") initially (targeting a defined and remodelled high-grade area), ramping up to 3,000 tpd once the mine was in production and generating cash flow.

In order to balance the compromise between cut-off grade, mining efficiency, and ounces produced, a 3.0 g/t cut off was selected in the initial years of the mine life, then reverting to 2.1 g/t for the remainder of the mine life. Additional stopes at an incremental cut-off grade of 1.4 g/t were also added where no additional development was required to mine them.

A mining recovery factor of 96.2% was applied to all stopes, and a dilution factor of 17.5% was applied when determining actual stope tonnages.

Access to the mine will be via two portals. Development has already commenced from the lower portal, with the exploration ramp. The updated design continues from the point at which the exploration ramp stops.

The Ollachea Mine Plan and production schedule is based on subset of the mineral resources and considers an 11-year life of mine. Production during years 1 to 3 will be at 1500 tpd before expanding to 3000 tpd from year 4 to 11.

The production schedule consists of 95.9% indicated material and 4.1% inferred material.

Average annual production over a four-year ramp-up period of approximately 66,000 ounces of gold at 1,500 tpd, with an estimated peak of 111,000 ounces in year five following the expansion to 3,000 tpd. The total of 1,003,957 ounces is mined over the 11-year life of mine ("LOM").

Figure 1-1 shows the production profile of tonnes versus mined grade.

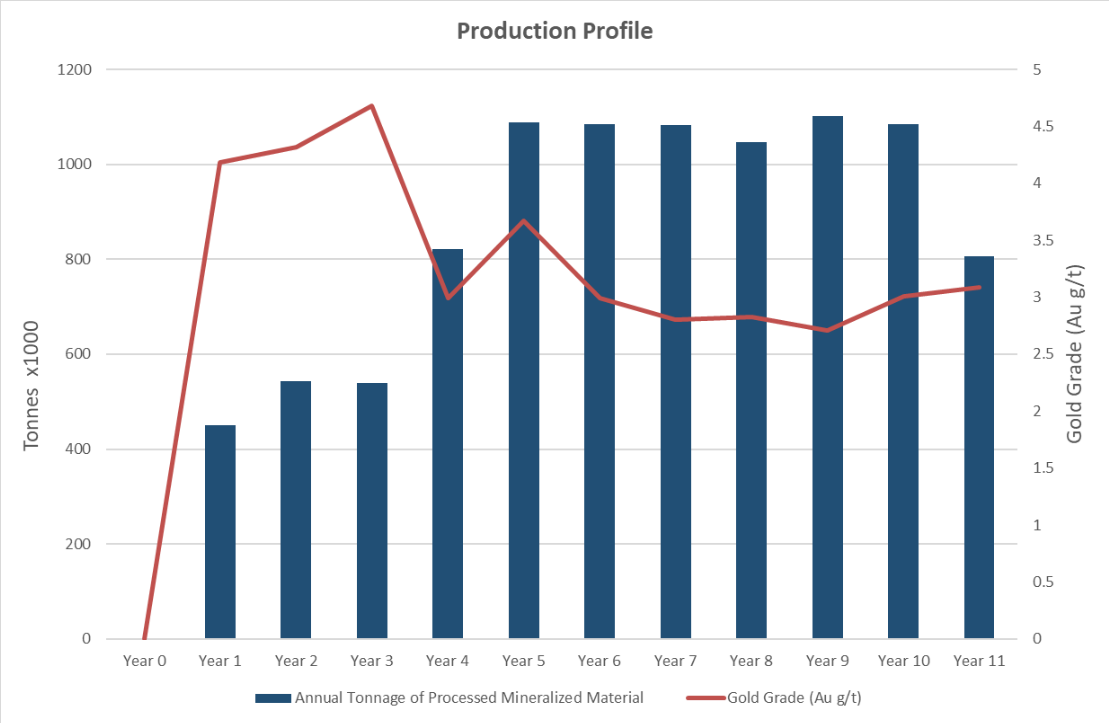

Figure 1-1: The production profile of tonnes versus mined grade

1.4 Metallurgical Test Work and Process Design

Extensive test work was carried out as support for the 2012 feasibility study prepared by AMEC Engineering ("2012 FS"). The test work developed a process flow sheet which used gravity separation and Carbon-in-leach (CIL) leaching of a gravity concentrate and the gravity tailings, essentially all of the feed. Subsequent work showed that higher recoveries could be achieved by gravity concentration if a higher mass of concentrate were produced, and if the mass of concentrate were increased to 15% of the feed, only the gravity concentrate need be leached to achieve gold recoveries of approximately 90%. This was comparable to the recovery achieved when the whole mass of the mineralized material was leached (gravity concentrate and tailings), but instead of two leach circuits, only one was needed, and this was only 15% of the capacity of that specified in the 2012 FS, resulting in a lower plant capital cost. An important difference is that only gravity concentrate is leached, and this is lower in organic carbon than gravity concentrate in tailings, as carbon is rejected in gravity concentration, which results in higher leach extractions.

The rest of the process flow sheet remained unchanged, with three-stage crushing, ball milling to a P_{80} of 75 microns, 2 stages of gravity concentration, CIL leaching with recovery of gold using the Zadra process, cyanide destruction using sulfur dioxide air with filtration of all the tailings for production of paste fill or for co-disposal with waste rock.

Instead of starting the project at 3000 tpd as proposed in the 2012 FS, the project will start at a production rate of 1500 tpd and treat mineralized material from the higher-grade area of the Minapampa zone. Production will double in year 4 when lower grade mineralization will be mined. This reduces initial capital costs.

The plant will be located on three platforms as was planned in a previous study (2012 FS) and which has been permitted. The mineralized material stockpile and crushing plant will be located on the upper platform, the mill and gravity concentration circuits will be located on the middle platform and the tailings filtration plant will be located on an extension to this platform. The leach and elution circuits will be located on the lower of the three platforms.

1.5 Waste Disposal

The Ollachea mine waste management concept has been developed to minimize the impacts of tailings and waste rock materials. The concept includes the following key aspects:

1. 43% of tailings to be returned to the mine as paste backfill.
2. Remaining 57% of tailings to be filtered to a low moisture content, and stacked in a system of co-disposed mine waste rock and filtered tailings product.
3. Co-disposal will occur at two locations: the Lower Portal Co-Disposal Facility (“LPCDF”) and the Cuncurchaca Co-Disposal Facility (“CCDF”).

The LPCDF will have a final, maximum height of 125 m; and the CCDF will have a final, maximum height of 150 m.

Table 1-2: Storage of Waste Rock and Tailings by Location

Location	Waste Rock (Mt)	Tailings (Mt)	Total by Location (Mt)
Lower Portal CDF	1.65	0.85	2.5
Cuncurchaca CDF	1.29	4.6	5.89
Underground Backfill	--	4.2	4.2
Total by Waste Type (Mt)	2.94	9.65	12.59

The total mine life presented in this Technical Report is approximately 11 years. Filtered tailings will be placed at the LPCDF during the first 2.5 years, approximately. For the remaining years, the filtered tailings will be transported approximately 4.0 km from the plant site to the CCDF using 15 m^3 capacity trucks. The trucks will be equipped with covered beds to minimize dusting and spillage during transport. The haul route includes approximately 2.0 km along the Interoceanic Highway and 2.0 km along access roads at the process plant and the CCDF.

Filtered tailings was selected as the most suitable tailings processing, primarily to obtain the required storage volume within a relatively limited distance from the process plant. This was not possible with conventional slurry tailings disposal or thickened tailings disposal methods, due to topographic limitations in the project area. Additional benefits offered by filtered tailings, relative to conventional or thickened tailings, include reduced land disturbance, and reduced Tailings Storage Facility ("TSF") seepage/effluent.

Tailings from the CIL circuit will be thickened to 60% solids and pass-through cyanide detoxification prior to being dewatered using pressure filtration. The filtered tailings are anticipated to be dewatered to a moisture content of approximately 16%, as required to achieve sufficient compaction at the co-disposal facilities.

Contingency planning for 'out-of-spec' tailings, that have a higher moisture content due to upset conditions at the filtering station, consists of the use of geotube tailings storage. Geotubes are very large geosynthetic bags, designed to retain the tailings solids, while allowing water to drain out, and thereby allowing consolidation of the tailings to a low moisture content, similar to mechanically-filtered tailings. The geotubes will be located within the body of the CDFs, such that separate contingency areas are not required.

1.6 Operating Cost Estimates

Operating cost estimates have been developed to provide an estimate suitable for the Technical Report ("PEA"), including costs for mining, processing and waste disposal. The expected accuracy range of the operating cost estimate is +30%/-30%.

LOM operating costs are summarized in Table 1-3.

Table 1-3: Estimated LOM Operating Costs

Operating Costs	LOM (US$ M)	$/tonne leached	$/oz Au
Mining (1)	$406	$42.10	$464
Processing	$127	$13.11	$144
Tailings and Waste Rock Disposal	$35	$3.66	$40
Onsite G&A (2)	$35	$3.65	$40
Total Operating Costs	**$603**	**$62.52**	**$688**
Treatment & Refining Charges	$4	$0.44	$5
Government Royalty	$35	$3.63	$40
Royalties (3)	$41	$4.21	$46
Community Interest	$11	$1.14	$13
Total Cash Costs	**$694**	**$71.94**	**$792**
Sustaining Capital	$1	$0.13	$2
All-in Sustaining Costs (AISC)	**$695**	**$72.08**	**$794**

(1) Includes paste backfill, supervision and stope definition drilling costs.
(2) Includes mine closure bond.
(3) Includes NSR of 2.9%.

Contingencies have not been considered when estimating operating costs.

1.7 Capital Cost Estimates

The capital cost estimate has been developed to provide an estimate suitable for the 2021 PEA, including costs to design, procure, construct, and commission the facilities.

The PEA estimates an initial CAPEX of US$89M to start with a design production capacity of 1,500 tpd. A plant expansion is anticipated during the fourth year to increase production capacity to 3,000 tpd. The waste disposal expansion is required in year two. The expansion capital cost estimate is approximately US$37M. Both estimates include a 25% contingency.

Capital costs estimated have been summarized in Table 1-4.

Table 1-4: Estimated Capital Costs

Description	US$
Start-up (Stage 1)(1)	
Mine	$27M
Process Plant (2)	$37M
Tailings and Waste Rock Disposal	$5M
Owner's Costs	$2M
Start-up Capital Costs Pre-Contingency	**$71M**
Contingency (25%)	$18M
Total Start-up Capital	**$89M**
Expansion (Stage 2)(3)	
Process Plant	$16M
Tailings and Waste Rock Disposal	$13M
Owner's Costs	$1M
Expansion Capital Costs Pre-Contingency	**$30M**
Contingency (25%)	$7M
Total Expansion Capital	**$37M**

(1) Includes mine development and plant construction with a design capacity of 1500 tpd.
(2) Includes EPCM costs. Also applicable to expansion.
(3) Includes Tailings Storage Facility construction and process plant ramp-up from 1500 tpd to the designed capacity of 3000 tpd.

1.8 Economic Analysis

The financial analysis was carried out using a discounted cash flow ("DCF") methodology. Net annual cash flows were estimated projecting yearly cash inflows (revenues) and subtracting projected yearly cash outflows (capital and operating costs, royalties, and taxes). These annual cash flows were discounted back to the date of beginning of capital expenditure at mid-year 2022 and totalled to determine the Net Present Value ("NPV") of the project at selected discount rates. A discount rate of 7% was used as the base discounting rate.

In addition, the Internal Rate of Return ("IRR"), expressed as the discount rate that yields an NPV of zero, and the payback period, expressed as the estimated time from the start of production until all initial capital expenditures have been recovered.

The economic analysis shows that using a base case gold price of US$1,600/oz, the Pre-Tax Net Present Value discounted at 7% ("NPV7%") is US$327M and with a 54% IRR, and the after-tax NPV7% is $189M with a 38% IRR.

Start-up CAPEX is estimated at $89M (including 25% contingency), with an after-tax payback period of 2.5 years.

Sensitivities of pre-tax and post-tax NPV and IRR to gold prices per ounce are presented in Table 1-5.

Table 1-5: Economic Sensitivity to Gold Prices

Gold Price ($/oz)	US$1400	US$1600	US$1800
Pre-Tax $NPV_{7\%}$	$223M	$327M	$430M
Pre-Tax IRR	40%	54%	68%
Pre-Tax Payback	2.5 years	2 years	1.7 years
After-Tax $NPV_{7\%}$	$125M	$189M	$253M
After-Tax IRR	28%	38%	47%
After-Tax Payback	3 years	2.5 years	2.2 years

Average gold recovery is 90.3% during the first three years, with average recovery of 86.2% over the remaining LOM. The recovery is variable with gold grade.

Sensitivities to variations in gold price, initial capital costs and operating costs were carried out to identify potential impacts on NPV and IRR. The After-Tax Economic Sensitivity to Gold Price, Operating and Capital Costs is shown in Figure 1-2.

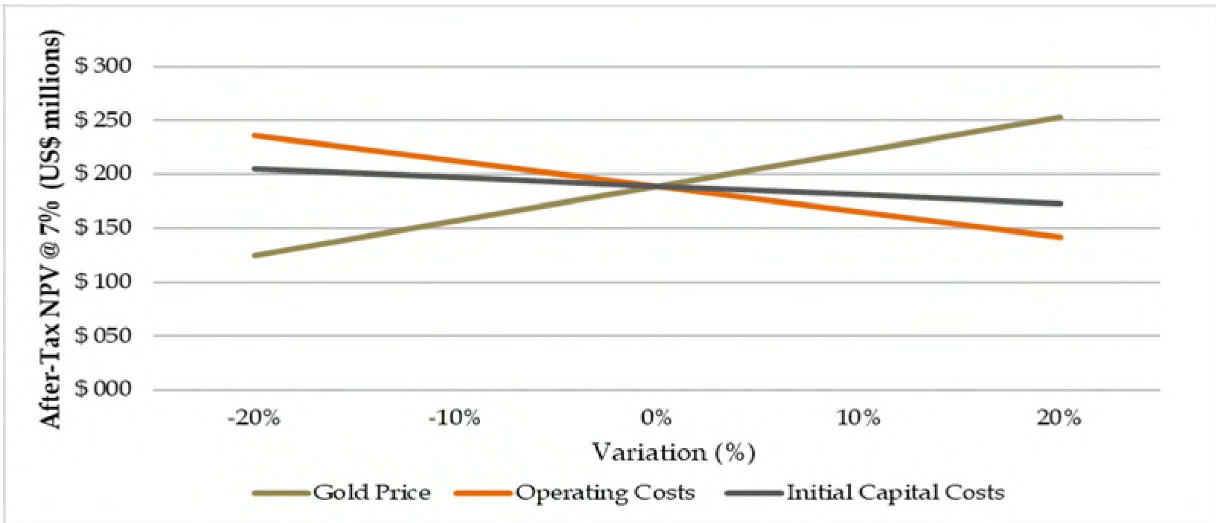

Figure 1-2: After-Tax Economic Sensitivity to Gold Price, Operating and Capital Costs

1.9 Interpretations and Conclusions

Geology and Mineral Resources

Best practices in geological data capture, storage and interpretation were implemented early at the Ollachea Property and have been maintained diligently. Throughout the project, the QAQC results have demonstrated the reliability of the sampling and assaying procedures, and the Mineral Resources have been estimated by independent Qualified Persons.

The Ollachea Property has in place the necessary regulatory licenses and authorisations required for its current tenement status. Furthermore, with the support of the community and government organisations, it is expected that future social license and authorisation requirements to advance the Property will successfully be attained.

Notwithstanding the above, experience in other parts of Peru suggests that social issues may also present a risk of project development delays in the future.

Mining and Mine Plan

Edgard Vilela (QP) considers that long hole open stoping (LHOS) with paste fill is the optimal mining method for the mineralization reported at the Property. Edgard Vilela (QP) notes that mineralization reported at the Property has good continuity along strike, and that he has seen LHOS successfully applied to numerous mines with mineralization with a similar geometry.

This study has indicted that there is a defined area where the mineralized material is amenable to a higher cut-off grade. The mineralized material can be mined at an elevated cut-off grade in the first 3 – 4 years without breaking it up into isolated stopes (which are significantly less economic to mine).

The revised mine plan presented offers an opportunity for a low start-up CAPEX, whilst still maintaining reasonable revenues.

Significant opportunity still exists with respect to Minapampa Far East, and the inclusion of that material in the mine plan and financial model. Further work will need to be completed with respect to waste storage options to increase the mine life significantly, but the mineralized material is present (the inferred resource in Minapampa Far East) and it is a direct extension of the Minapampa area.

Metallurgy and Mineral Process Design

Using the results of the two gravity concentration tests reported by Met-Solve in 2017 and 2021, with head grades of 3.29 and 4.35 g/t Au, respectively, with CIL leaching of all the tailings from the re-grind circuit, predicted overall recoveries of gold that are presented in Table 1-6.

Table 1-6: Summary of Overall Gold Recovery

Head Grade g/t Au	3.29	4.35
Gold Recovery	86.2 %	90.3 %

The assumptions used are:

- Recovery of gold from high-grade concentrates using a shaking table is 50%.
- Tailings grade after recovery of a high mass pull concentrate (15%) is 0.4 g/t Au.
- Tailings grade after CIL leaching (Ammtec 2013) is 0.3 g/t Au.
- Overall process losses in smelting, solution losses in CIL is 1.0 %.

Waste Disposal

The Ollachea tailings and waste rock management concept has been developed to minimize impacts, through implementing Best Available Technologies (BATs) and Best Available Practices (BAPs), in accordance with current, global tailings standards and guidelines. The filtered tailings and co-disposal concepts are state-of-the-art for tailings management and, as such, represent a high-value, low-impact and low-risk option. The technologies and methods involved will require implementation and commissioning during initial stages of operation, in order to optimize the processes and methods for placing the tailings and waste rock. Additionally, during the initial stages, the geotechnical characteristics of the material should be verified, against values assumed in design.

1.10 Recommendations

1.10.1 Geology and Resources

The Company plans to conduct additional exploration activities to add to the existing Mineral Resource, although there is no timeline placed on any exploration work or update to the Mineral Resource Estimate ("MRE") at present. The QPs recommend a survey of artisanal workings be completed, however the cost of this potential work is unknown.

1.10.2 Mining and Mine Plan

Further work should be completed to optimise the mine plan, making minor modifications to cut off grade early in the mine life to maximise the ounces produced. This should then be followed by a redesign of the stopes that remain in the high-grade zone after the ramp up to 3000 tpd to a cut-off grade of 2.1 grams per tonne maximising the resource recovery (increase of the ounces in the mine plan).

The sizes of the drive should be matched against the preferred contractor's equipment.

Significant artisanal mining activity continues around the upper portal, and the locations of the portal should be reassessed and modified considering the location of the artisanal mines.

1.10.3 Metallurgy and Mineral Process Design

Gravity concentration tests on samples from other zones of the mineralized material are needed, together with leaching tests on the concentrates produced.

A budget estimate of US$300,000 should be allocated to source and test sufficient samples, although there is further work to be completed to define the drilling and sampling locations.

1.10.4 Tailings and Waste Rock Management

An opportunity exists to eliminate the imported clay / geosynthetic clay liner at the Lower Portal Co-Disposal Facility ("LPCDF"). This is contingent upon demonstrating that the filtered tailings will act as a low-permeability element, as for the Cuncurchaca Co-Disposal Facility ("CCDF"). Further, the concept would need to be presented to regulators for approval. This could reduce the time and cost associated with constructing the LPCDF, as well as simplifying operation.

A further opportunity exists to increase the placement of tailings solids as underground paste backfill. For this PEA, relatively conservative values were used for the solids content of the backfill mix. This would reduce the required storage on-surface.

It is recommended that the mixed placement of the filtered tailings together with the waste rock be planned in detail, prior to beginning operation.

Further information on the geotechnical characteristics of the waste rock should be determined, as inputs to stability and seepage analyses.

Stability analyses must be done on the Co-Disposal Facilities, to confirm that assumed design slopes are safely achievable.

The proposed contingency for off-spec tailings to be discharged into geotextile geotubes, should be trial-tested at site, prior to full commissioning, using smaller, test-size geotubes, to confirm the type of geotextile and flocculant, if required.

2 INTRODUCTION

Minera IRL is publicly listed on the Lima Stock Exchange and the Canadian Securities Exchange (with the ticker "MIRL") and owns the Ollachea Gold Project (the "Property") in the Puno Region of southern Peru via its Peruvian subsidiary, Minera Kuri Kullu SA ("MKK").

MIRL's interest in the Property began in 2006 when the property was acquired from Rio Tinto. Since taking control of the Property, MIRL has advanced the Property to a maiden resource and subsequently, in 2012, a Feasibility Study.

2.1 Terms of Reference

MIRL has commissioned Mining Plus to develop a Preliminary Economic Assessment (the "Technical Report") detailing the Company's revised plans for the Ollachea Property.

The Technical Study considers:

- Additional mineral resources, following drilling in the Minapampa Far East area of the Property.
- A revised gold price in line with the current market value.
- The viability of a low-CAPEX start-up for Ollachea.
- Increased recovery of gold by gravity concentration.
- A gravity concentration and carbon in leach ("CIL") plant designed to treat 1,500 tonnes per day ("tpd") over the first three years (targeting a defined and remodelled high-grade area) ramping up to 3,000 tpd during the fourth year.
- A review of tailings management and storage.

The Technical Report has been prepared to provide technical information to support the July 19th, 2021, press release issued by MIRL titled: "Minera IRL Announces Positive Preliminary Economic Assessment Results for the Ollachea Gold Project".

2.2 Information Sources and References

Mining Plus has developed the Technical Report based on the following sources of information:

- Drilling database compiled by MIRL, including; collar information, downhole surveys, assay data, and geological logs.
- 3D geological and structural model developed by MIRL for the Minapampa area.
- Minera IRL S.A., Ollachea Deposit, Mineral Resource Update March 2014; Prepared by GHD; Dated April 2014
- Ingeniería de Detalle Depósito de Desmonte y Mineral de Baja Ley - Informe Civil e Hidráulico - Proyecto Ollachea - ITE-1306.10.02-300-001 - Revisión B - Prepared por Anddes Asociados - Julio 2014

- Ingeniería de Detalle de Depósitos de Relaves - Cuncurchaca Informe Civil e Hidráulico - Proyecto Ollachea - Preparado por: Anddes Asociados S.A.C. - Proyecto N°: TE-1306.10.02-200-004 Revisión 2 Diciembre 2014
- Ollachea Gold Project - Engineering Report - Document No. 650330-0000-3000-RPT-0008 Rev 1 - AMEC - 25-January-2013
- Ollachea Gold Project - NI 43-101 Technical Report on Feasibility Study - Submitted by: AMEC (Perú) S.A. Date: 29 November 2012. Project Number: 65033
- Ollachea Pre-Feasibility Study - Final Report - Submitted by: AMEC (Perú) S.A. Date: October 2011. Project Number: 166729
- Ollachea geological data capture: Compliance with NI 43-101; Prepared by Mining Plus; Dated 5 September 2016
- Ollachea Mining Optimization Study 1500 to 3000 Tones Per Day; Prepared by Mining Plus; Dated May 2017
- Kappes Cassidy and Associates – Summary of 2009 – 2010, 2013
- Ammtec Metallurgical Investigation 2010- 2011
- Gravity concentration tests carried out my Plenge, July 21, 2017
- SGS Concentracion Gravimetrica en Equipo Falcon, 19 August 2017
- Met-Solve Investigation MS1809 for Minera IRL S.A. October 12, 2017
- Met-Solve Investigation JO102- 107, 20 April, 2021

Metric measurements have been used throughout the Technical Report unless otherwise cited.

The PSAD56 and WGS84 coordinate systems have been used in the Technical Report.

2.3 Effective Dates

The effective date of the Technical Report is the 27th August 2021.

2.4 Site Visits and Scope of Personal Inspection

Qualified Persons' visits to the Property:

- Doug Corley has experience with the Ollachea Property since 2009 and was the 'qualified person' (QP) in terms of NI 43-101 reporting, for the July 2012 mineral resource estimate. Doug visited the Ollachea project site between the 21st and 22nd of June 2010, and an additional site visit was conducted from the 13th to 22nd of January 2014, to meet with the Lima based geological staff to help understand the changes to the geological / structural interpretation.
- Andrew Fowler visited the site between the 21st and 22nd of November 2016 where he inspected the drilling and QAQC procedures for the Minapampa Far East drilling program.
- Edgard Vilela visited the site between the 19th and the 20th of May 2021 and inspected the existing tunnel, proposed location of the treatment plant, proposed location of the tailings dam, water and energy intakes projected for the project.

- Donald Hickson visited the site between the 16th and 20th of January 2012 and inspected the proposed locations of the treatment plant, tunnel, and mine waste co-disposal facilities.

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QPs) responsible for this report have relied upon the following information provided to them by the issuer (Minera IRL, or MIRL) concerning, legal, political, environmental, and tax matters relevant to this technical report.

3.1 Land Tenure

The QPs have not reviewed the land tenure, nor independently verified the legal status, ownership of the Project area or underlying property agreements.

MIRL has provided information with respect to Land Tenure used in Section 4 of this Technical Report. The QPs have fully relied upon, and disclaim responsibility for information provided with respect to Land Tenure.

3.2 Surface Rights Agreements

All details related to surface rights agreements with the Ollachea Farming Community have been provided by MIRL.

MIRL has provided information with respect to surface rights agreements used in Section 4 and Section 20 of this Technical Report. The QPs have fully relied upon, and disclaim responsibility for information provided by MIRL with respect to surface rights agreements.

3.3 Environmental Liabilities and Permits Acquired

All details related to environmental liabilities and permits required to construct and operate the Ollachea Mine have been provided by MIRL.

MIRL has provided information with respect to environmental liabilities and permits used in Section 4 and Section 20 of this Technical Report. The QPs have fully relied upon, and disclaim responsibility for information provided with respect to environmental liabilities and permitting.

3.4 Environmental Studies, Permitting, and Social or Community Impact

All details related to environmental studies, permitting, and social or community impact have been provided by MIRL.

MIRL has provided information with respect to Environmental Studies, Permitting, and Social or Community Impact used in Section 4 and Section 20 of this Technical Report. The QPs have fully relied upon, and disclaim responsibility for, information provided by MIRL with respect to environmental studies, permitting, and social or community impact.

3.5 Depreciation, Taxes, and Royalties

All details related to depreciation, taxes, and royalties have been provided by MIRL.

Information pertaining to Depreciation, Taxes, and Royalties has been used in the cash flow model as documented in Section 21 and Section 22. These aspects of the cash flow model have been completed by Capia Servicios Financieros (a specialist investment banking firm). The QPs have fully relied upon, and disclaim responsibility for, information provided by MIRL and Capia with respect to depreciation, taxes, and royalties.

4 PROPERTY, DESCRIPTION AND LOCATION

The Property is located in southeast Peru and the Region of Puno, approximately equidistant from the cities of Cusco and Juliaca (Figure 4-1). The approximate centre of the Property is provided in (Table 4-1).

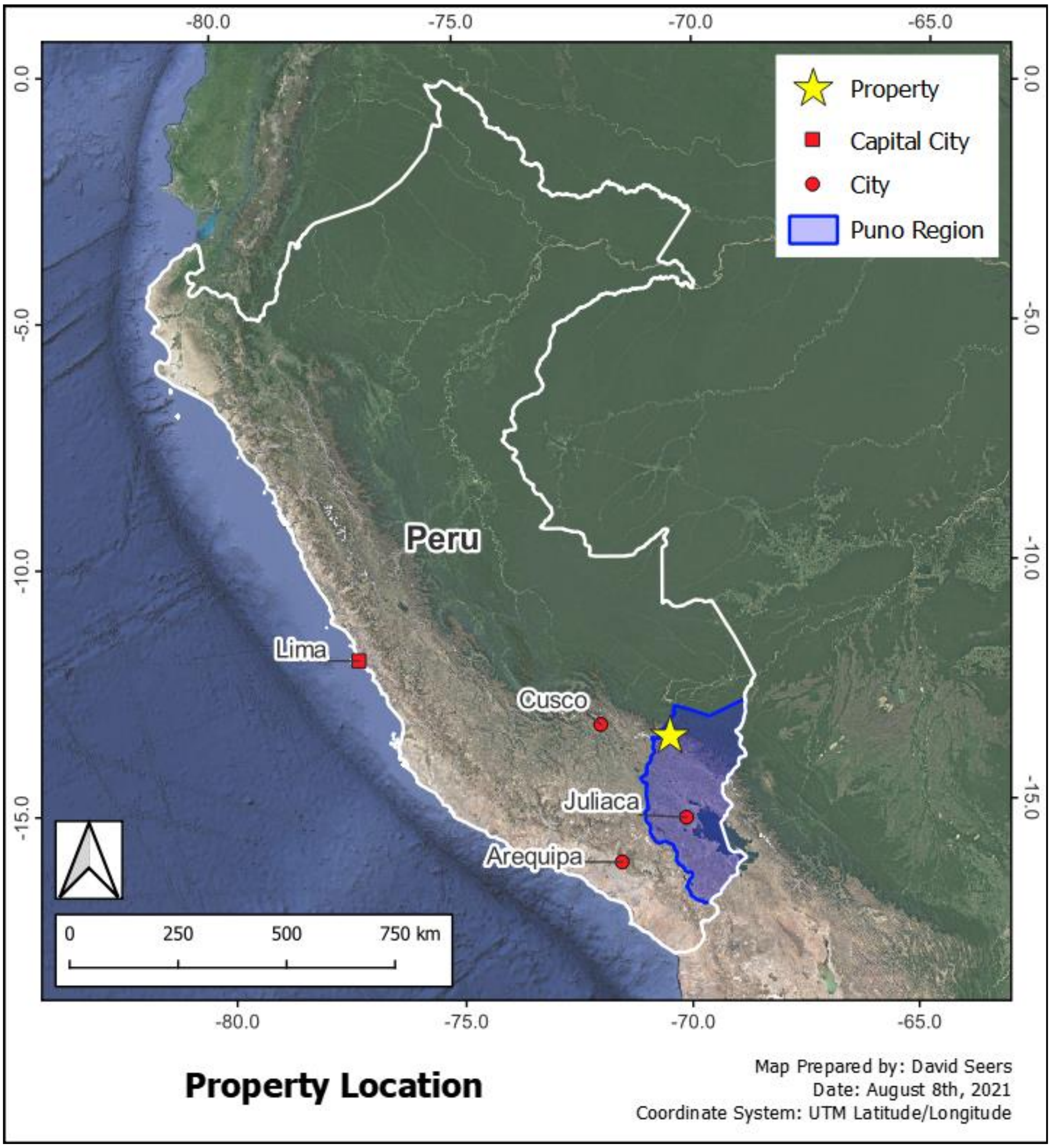

Figure 4-1: Property Location

Table 4-1: Approximate Centre Coordinates of the Property

Coordinate System	Easting / Longitude	Northing / Latitude
WGS 84, UTM Zone 18S	340544	8476187
UTM, Latitude/Longitude	-70.476	-13.783

The Property extends across the districts of Ayapata, and Ollachea, in the Carabaya Province of northern Puno (Figure 4-2).

The Property consists of 18 contiguous mining concessions ("Concession Minera"), some concessions partially overlap and a gap is recorded between Oyaechea [sic] concessions 1, 2, and 3 (Figure 4-2). Considering overlaps and gaps, the total footprint of the property is approximately 9899 hectares. Details for individual concessions are provided in Table 4-2.

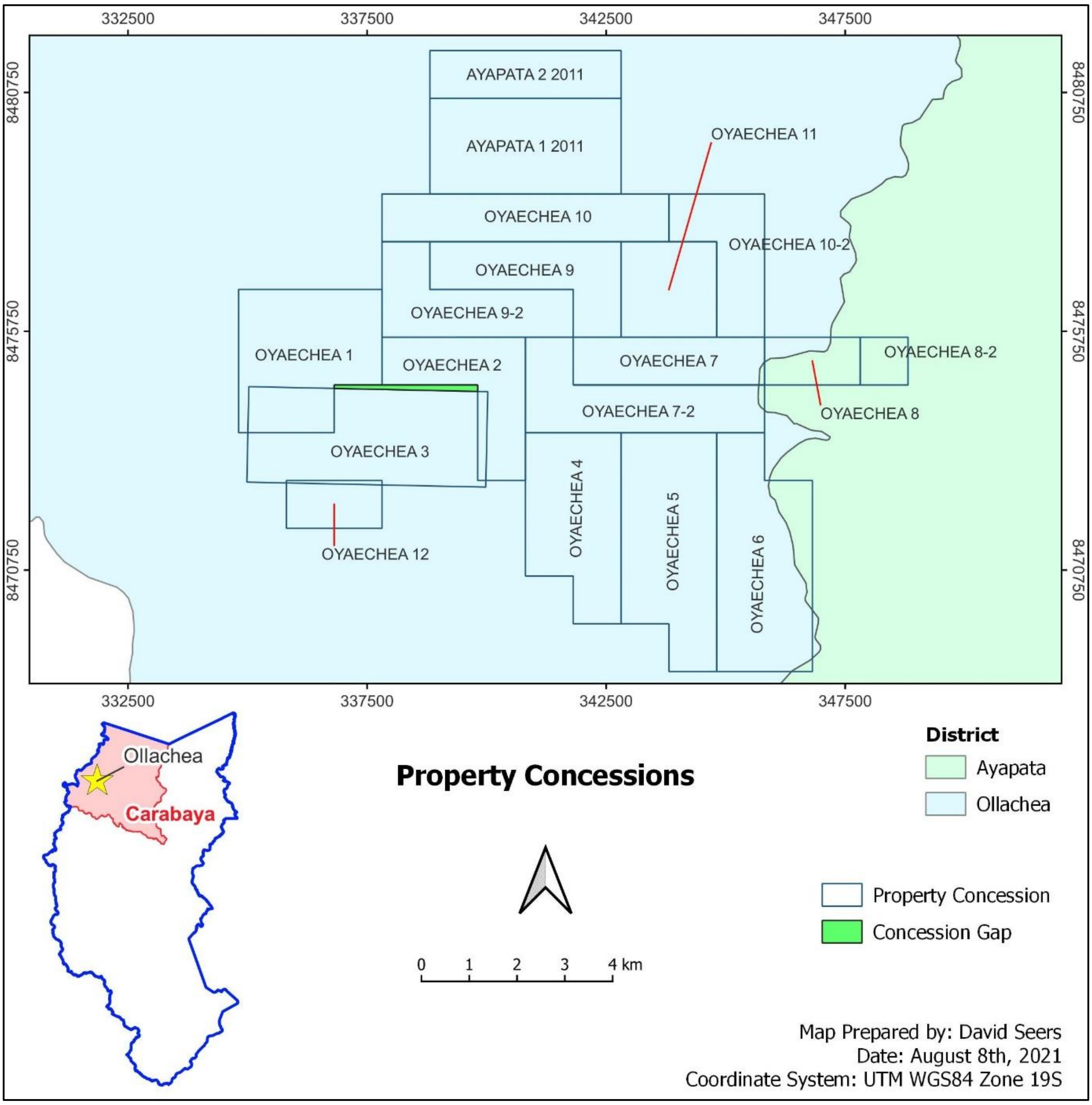

Figure 4-2: Property Concessions

Table 4-2: Property Concession Details

Concession Code	Concession Name	Title Holder	Date Registered	Approx. Area Hectares
010139909A	OYAECHEA 9-2	Compania Minera Kuri Kullu S.A.	29/04/2014	500
010140009A	OYAECHEA 10-2	Compania Minera Kuri Kullu S.A.	27/02/2014	400
010389807A	OYAECHEA 8-2	Compania Minera Kuri Kullu S.A.	27/02/2014	100
010218103	OYAECHEA 3	Compania Minera Kuri Kullu S.A.	30/06/2005	998.98
010215103	OYAECHEA 2	Compania Minera Kuri Kullu S.A.	08/11(2006	500
010215303	OYAECHEA 5	Compania Minera Kuri Kullu S.A.	06/02/2004	900
010139909	OYAECHEA 9	Compania Minera Kuri KulluS.A.	18/02/2010	500
010215003	OYAECHEA 1	Compania Minera Kuri Kullu S.A.	08/11/2006	800
010215203	OYAECHEA 4	Compania Minera Kuri Kullu S.A.	10/02/2004	700
010389907	OYAECHEA 7	Compania Minera Kuri Kullu S.A.	14/05/2009	400
010215403	OYAECHEA 6	Compania Minera Kuri Kullu S.A.	08/11/2006	900
010389807	OYAECHEA 8	Compania Minera Kuri Kullu S.A.	07/05/2009	200
010140109	OYAECHEA 11	Compania Minera Kuri Kullu S.A.	11/02/2010	400
010140009	OYAECHEA 10	Compania Minera Kuri Kullu S.A.	11/12/2010	600
010167809	OYAECHEA 12	Compania Minera Kuri Kullu S.A.	08/04/2010	200
010389907A	OYAECHEA 7-2	Compania Minera Kuri Kullu S.A.	27/02/2014	600
010165911	AYAPATA 2 2011	Compania Minera Kuri Kullu S.A.	07/05/2013	400
010165811	AYAPATA 1 2011	Compania Minera Kuri Kullu S.A.	07/05/2013	800

The concessions that makeup the Property are registered in the name of Compania Minera Kuri Kullu S.A. ("MKK"). MKK has negotiated 100% interest in the concessions that makeup the Property.

Pursuant to article 39 of the General Mining Law, titleholders of mining concessions should pay an Annual Maintenance Fee (derecho de vigencia). The Annual Maintenance Fee is due on June 30th of each year, is paid one year in advance and is calculated at a rate of US$3.00/ha. Failure to pay the Annual Maintenance Fee for two consecutive years causes the termination (caducidad) of the mining concession. However, according to article 59 of the General Mining Law, payment for one year may be delayed without penalty and the mining concessions remain in good standing.

If Annual Maintenance Fees are paid, and the concessions exploited, the titleholder can hold the concessions in perpetuity. Annual Maintenance Fees for the Property are up to date and the Property remains in good standing.

A mining concession does not grant the titleholder right of access. Right of access must be negotiated between the landowner(s) and concession holder.

Surface Rights

On November 25, 2007, MKK signed a surface rights agreement with the Community of Ollachea covering an area of approximately 6000 ha, which included the mining concessions called Oyaechea 1, Oyaechea 2, Oyaechea 3, Oyaechea 4, Oyaechea 5 and Oyaechea 6 (the main area of economic interest). The original term of this agreement was five years; however, it was extended for a period of 30 years on May 30, 2012.

Pursuant to the agreement, MKK will make payments for surface rights access amounting to PEN 100,000 each year in 2013 and 2014. In addition, MKK agreed to pay PEN 150,000 per year upon commencement of production, and throughout the term of the agreement. MKK also undertook to develop social responsibility programs for a total amount of PEN 3,960,000 between 2013 and 2014. This social commitment is 90% fulfilled, with only two remaining programs to be implemented during the construction phase. This agreement will be reviewed in terms of contributions to social and community programs once the mine is in operation, which was initially intended for 2015.

The agreement also includes a contribution for technical support to artisanal miners. In addition, MKK will grant a participation of 5% in the share capital of MKK to the Community of Ollachea upon the commencement of commercial production.

Agreements and Royalties

The following is summarized from the MIRL Annual Information Form for 2020 (MIRL, 2021) and is supported by Tong (2012).

On September 1, 2006, MIRL signed an agreement with Rio Tinto to acquire the original Ollachea concessions[1] (Oyaechea 1 to 6, Ayapata Uno 1 to 2, and Ayapata Dos 1 to 3). This entailed an initial payment of US$250,000 plus progressive payments totalling US$6,000,000 over four years, together with two additional payments in the event that Rio Tinto's clawback right under the agreement was not exercised. The option was conditional on MIRL successfully negotiating a surface rights agreement with the local community within 120 days.

On February 23, 2007, the Mining Transfer Agreement was entered into by Rio Tinto and MKK, whereby all concessions were transferred to MKK.

Rio Tinto's clawback right lapsed in 2009, and on December 15, 2009, Rio Tinto notified MIRL and MKK that MKK was to make the first additional payment allowing Rio Tinto a 1% net smelter return ("NSR") in exchange for payment of approximately US$3,807,000 million.

[1] The additional mining concessions have been acquired by MKK independently.

The Feasibility Study for the project was completed in 2012 (2012 FS), and in the third quarter of 2013, it was agreed that MIRL would pay a final amount of $21.5 million to Rio Tinto based on the results of the 2012 FS. Payment was originally scheduled to be made in three instalments, with the option to settle up to 80% of the payment in ordinary shares of MIRL. The amount outstanding would accrue interest at a rate of 7% per annum.

On January 28, 2014, 44,126,780 ordinary shares were issued to Rio Tinto in settlement of the First Instalment ($7.3 million), plus accrued interest for a total payment of $7.4 million. In addition, it was agreed that if Rio Tinto did not sell any ordinary shares for a period of one year, they would be entitled to a cash share hold incentive payment totalling $744,000. The Final Instalment representing the remaining 66% of the total amount payable ($14.2 million) was due in July 2016.

In June 2015, MIRL paid $12.0 million to Rio Tinto along with the $744,000 share hold incentive by using the proceeds from a bridge loan granted by Peruvian development bank Corporación Financiera de Desarrollo ("COFIDE"). A promissory note for the balance of $2.2 million was issued by MIRL to Rio Tinto. MIRL has repaid $700,000 of the principal plus interests. As of June 30, 2021, the outstanding balance payable to Rio Tinto is US$1.516M (inclusive of interest).

The Peruvian government currently levies a royalty based on gross profit per quarter from mining operations that ranges between 1% (for profits between 0% and 10%) and 12% (for profits greater than 80%).

Gold production from the Property is subject to three separate royalty agreements:

- 1.0% payable to Osisko (initially held by Rio Tinto and subsequently by BCKP)
- 1.0% payable to Macquarie
- 0.9% payable to Sherpa.

Permitting

An Environmental Impact Assessment ("EIA") for the Property has been approved by the Peruvian Ministry of Energy and Mines, and the National Environmental Certification Service for Sustainable Investment (SENACE). The EIA is valid for the life of the mining project.

Additional permits will be required to support Project development. Permitting is discussed in more detail in Section 20 of the Technical Report.

Environmental Liabilities

A physical, biological and socio-economic baseline has been established on the basis of ongoing social, environmental and archaeological baseline surveys carried out by MKK since

2007. Additional information on the Project environmental and social licences is contained in Section 20 of the Technical Report.

Environmental liabilities associated with waste dumps and tunnels generated by the artisanal mining activities on the property have been evaluated and are subject to ongoing monitoring as part of MKK's environmental baseline study work. Additional information on these activities and liabilities of the Project are included in Section 20 of the Technical Report.

Social Licence

The Ollachea Project has the approval and acceptance from the Ollachea Community. This has been formally ratified in an agreement signed on May 30, 2012, which increased the surface rights permit period to 30 years and strengthens the commitment between the project and the local community.

In MKK's opinion, "in most instances, local community groups see themselves as co-owners of the project and have a positive perspective of the project´s impact on their personal and family lives".

Most planned operational activities are underground and should not significantly affect the perceptions of environmental care that the community has about the project. Likewise, the local population has been productively incorporated into the mining activities and, in the specific case of artisanal miners; they have been incorporated by helping in the formalization of their activities. In addition, the social management implemented by MKK (in the form of support to vulnerable groups, food security projects, etc.) has been designed to mitigate negative impacts on the socioeconomic dynamic of the general population. In addition to providing employment opportunities in the planned mining operations, social initiatives consider the retraining of artisanal miners in activities that would improve their income.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Physiography

The Property is situated at the western margin of the Amazon basin and on the eastern slopes of the Andes Mountain range.

Elevation at the Property varies significantly between approximately 2350 meters above sea level in the San Gaban drainage and approximately 4800 meters above sea level. Topography is characterised by sharp ridges and steeply incised valleys (Figure 5-1).

Figure 5-1: Photography demonstrating physiography of the Property

Vegetation is variable, lower elevations are characterised by lush cloud forest, parts of which have been cleared for farmed crops. Upper elevations may be barren of vegetation; however, when vegetation is present, it is characterised by grasses and short shrubs.

5.2 Accessibility

A quality paved and maintained highway passes within a short distance of the planned location for the processing plant. The highway can be used to connect the Property with the significant cities of Puerto Maldonado, Cusco and Juliaca. These cities are served by daily flights from Peru’s capital city, Lima. From the highway, a series of 4x4 accessible unpaved roads offer access to various parts of Property.

The Property is connected by highway to the Pacific deep seaport of Matarani as shown in Figure 5-2.

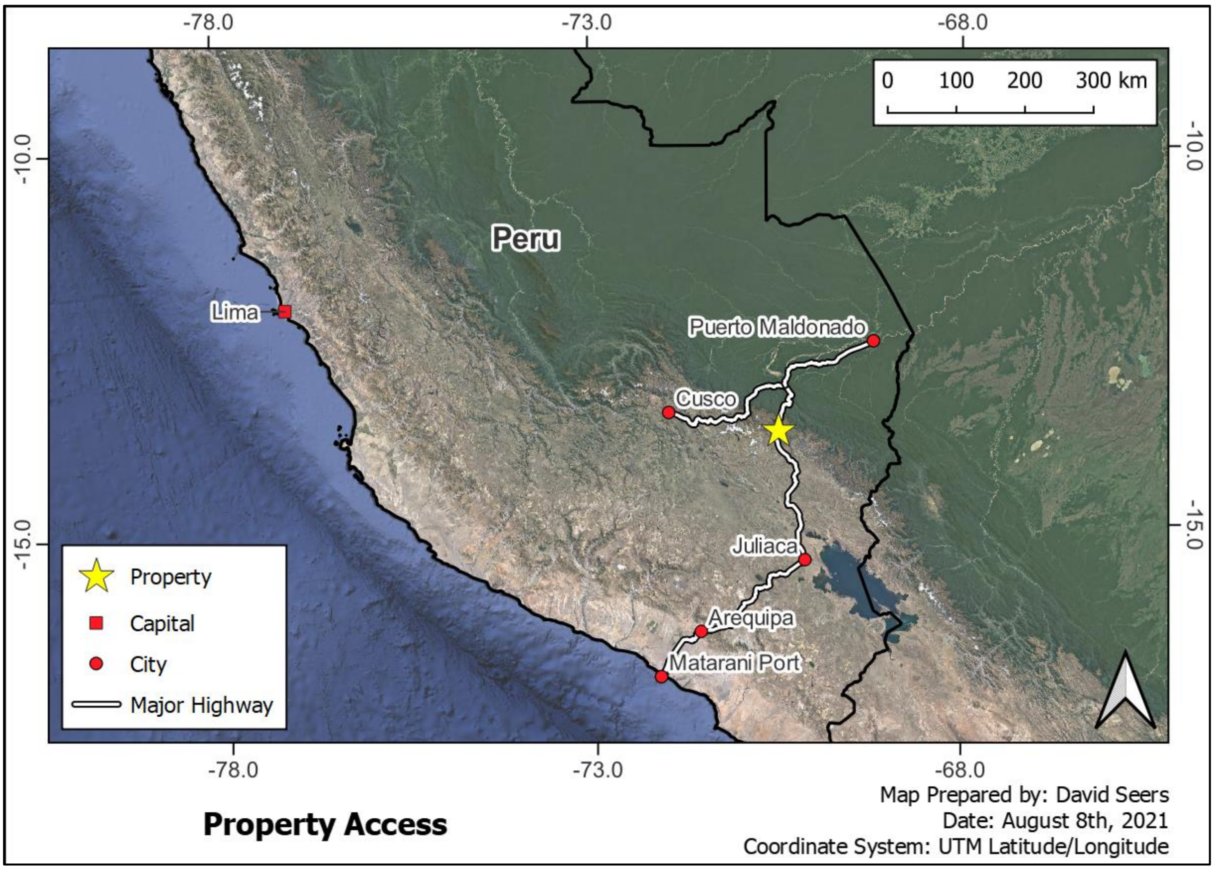

Figure 5-2: Property access

Approximate drive times (private vehicle) and distance between the Property and significant cities is provided in Table 5-1.

Table 5-1: Approximate drive times

Route	Approximate Distance	Approximate Drive Time
Juliaca to Property	260 km	4 hrs 5 mins
Puerto Maldonado to Property	305 km	5 hrs 30 mins
Cusco to Property	480 km	7 hrs 30 mins
Matarani to Property	630 km	10 hrs

5.3 Climate

The Project has a temperate sub-alpine climate with a pronounced rainy season and dry season. The rainy season extends from December to March, the dry season from May to August and the remaining months of April, September, October, and November are transition months. Based on historic data, average precipitation in the study area ranges from 20.9 mm

(June) to 228.7 mm (January). The maximum average monthly temperatures range from 12.8 °C to 14.6 °C from November to January. The minimum average monthly temperatures range from 10.6 °C to 12.3 °C between June and August. The predominant wind directions are northeast and northwest.

The moderate climate allows exploration activities to be carried out year-round and would also allow mine development and operation activities to be carried out year-round.

Access to the Property can be temporarily blocked by landslides triggered during heavy periods of rain. Because roads are maintained and alternative routes are available, landslides do not pose a significant threat to Property access.

5.4 Local Resources and Infrastructure

The Project is located immediately adjacent to the town of Ollachea which can provide basic commercial and labour support for exploration, development, and operational activities. The involvement of the community in the construction of the Interoceanic Highway and artisanal mining activities have served as training for the local workforce in basic construction and other support activities that will allow local workers to be involved in the development and operation of the Ollachea Project. The issuer has also invested in training artisanal miners, active in the Property, in aspects of mining and business.

The cities of Juliaca, Puno, and Puerto Maldonado offer access to a more sophisticated labour workforce with local university and college campuses, and commercial support for basic supplies including cement, aggregate, fuel, and food. It is expected that all additional labour, equipment and supplies required for the project can be procured nationally from the cities of Arequipa and Lima.

The San Gaban and San Gaban II hydroelectric generating stations are within 50 km of the Project and a number of other hydroelectric projects are proposed for the area. A major high-tension power line connecting the San Gaban II station with Azangaro runs through the Project.

6 HISTORY

Records of mining activities in the region date back to the 18th century and the time of the Spanish conquistadores. Informal mining activities targeting hard rock and placer gold continue to this day and likely date back much further than the 18th Century.

Modern exploration at the Property began in the late 20th century:

Peruvian Gold Ltd – 1998 to 1999

The publicly traded Canadian explorer drilled five diamond drill holes and identified low-grade gold mineralization.

Rio Tinto – 2003 to 2004

Following a program of regional stream sediment sampling, Rio Tinto explored the Property for gold with programs of surface sampling.

Rio Tinto formally ceased its interest in the Property in 2006, ceding control to Minera IRL.

Minera IRL – 2006 on-going

Under the national Act of Formalization and Promotion of the Small-scale and Artisanal Mining Industry, MKK worked with the Community of Ollachea to reach a mutually beneficially agreement that would accommodate MKK's exploration plans and allow the communities' artisanal mining activities at Minapampa.

The Community of Ollachea granted MKK permission to access and explore its' lands and MKK allowed for the continuation of surface mining activities in a specified area of Minapampa. Agreements were formalized in 2007. Small-scale mining continues to this day in the Minapampa area under the terms of the agreement.

The surface rights agreement was extended for a period of 30 years on May 20th, 2012.

MKK commenced exploration at the Property in early 2008:

- Geochemical sampling, mapping, and structural measurements were undertaken in areas identified from analysis and interpretation of ASTER (Advanced Spaceborne Thermal Emission and Reflection Radiometer) imagery
- By the end of September 2009, 71 diamond drill holes (26,026 m) had been completed at the Property
- A maiden mineral resource estimate was published in 2010
- Mineral IRL announced a Pre-feasibility Study (PFS) for the Property on July 18th, 2011. The PFS considered 120 drill holes and (46,404 m) in the Minapampa Zone

- A Feasibility Study considering an updated mineral resource estimate based on 151 drill holes (59,509 m) was announced by Minera IRL in 2012

6.1 Recent History

Mr Seers (QP) has summarised the recent history of the Property based on news releases by Minera IRL:

December 20, 2012 Environmental Impact Assessment (EIA) submitted to the Peruvian authorities

February 13, 2013 Completion of 1,234 m, (US$13.8M) Exploration Tunnel to the eastern extension of Minapampa (Minapampa Far East). The Exploration Tunnel was developed to facilitate underground drilling. Ground conditions were reported to be better than anticipated and ground water ingress was significantly less than had been expected.

May 22, 2013 Final community endorsement of Environmental and Social Impact Assessment announced. Community approval was unanimous.

September 26, 2013 Peruvian Ministry of Energy and Mines (MEM) approved the Environmental and Social Impact Assessment (ESIA) for the Ollachea Gold Project.

June 4, 2014 Positive results announced for post-Definitive Feasibility Study (DFS) mine optimization studies.

November 28, 2016 Results from 22-hole (5146.9 m) drill program from the Exploration Tunnel announced. Drilling confirmed the down-plunge extension of mineralization at Minapampa (Minapampa Far East).

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Geological, Mining and Metallurgical Institute ("INGEMMET") publishes 1:100k scale geological maps covering much of Peru. INGEMMET also publishes descriptive Bulletins (Boletín) detailing regional geology, lithological units, structure, and economic geology.

The area around the Property is covered by 1:100k map sheets 28U "Mapa Geológico del Cuadrángulo Coriani", and 28V "Mapa Geológico del Cuadrángulo Ayapata". Regional geology is described in Boletín 90 "Geológico del Cuadrángulo de Coriani y Ayapata".

David Seers (QP) has summarized the key regional geological features based on work published by INGEMMET (Figure 7-1):

- The Sandia Formation of Ordovician sediments has been thrust over the Ananea Formation of Devonian sediment along the prominent west-northwest trending Ollachea thrust fault (Ollachea Fault). Sandia Formation sediments have been metamorphosed at the thrust front.
- Intrusive plugs of Permian age have intruded the Sandia and Ananea Formation sediments across the thrust front. A Permian plug extends for over 20 km in a northeast direction and is a prominent feature to the north and west of the Property.
- The Mitu Group of Permian volcanics outcrop to the south and east of the Property and south of the Ollachea Fault.
- Jurassic plugs have been exposed through Permian volcanics.
- Cenozoic volcanics of the Quenamari Formation are exposed to the southeast of the Property and northeast to north-northeast normal faulting is recorded in the Quenamari Formation volcanics.
- Quaternary deposits have accumulated in drainages.
- The Property sits within Metallogenic Belt III, close to the thrust front and in the upper plate of the Ollachea Fault between Permian and Jurassic Intrusions. Metallogenic Belt III is recognized for hosting mineralization related to igneous activity during Permian and Triassic times.

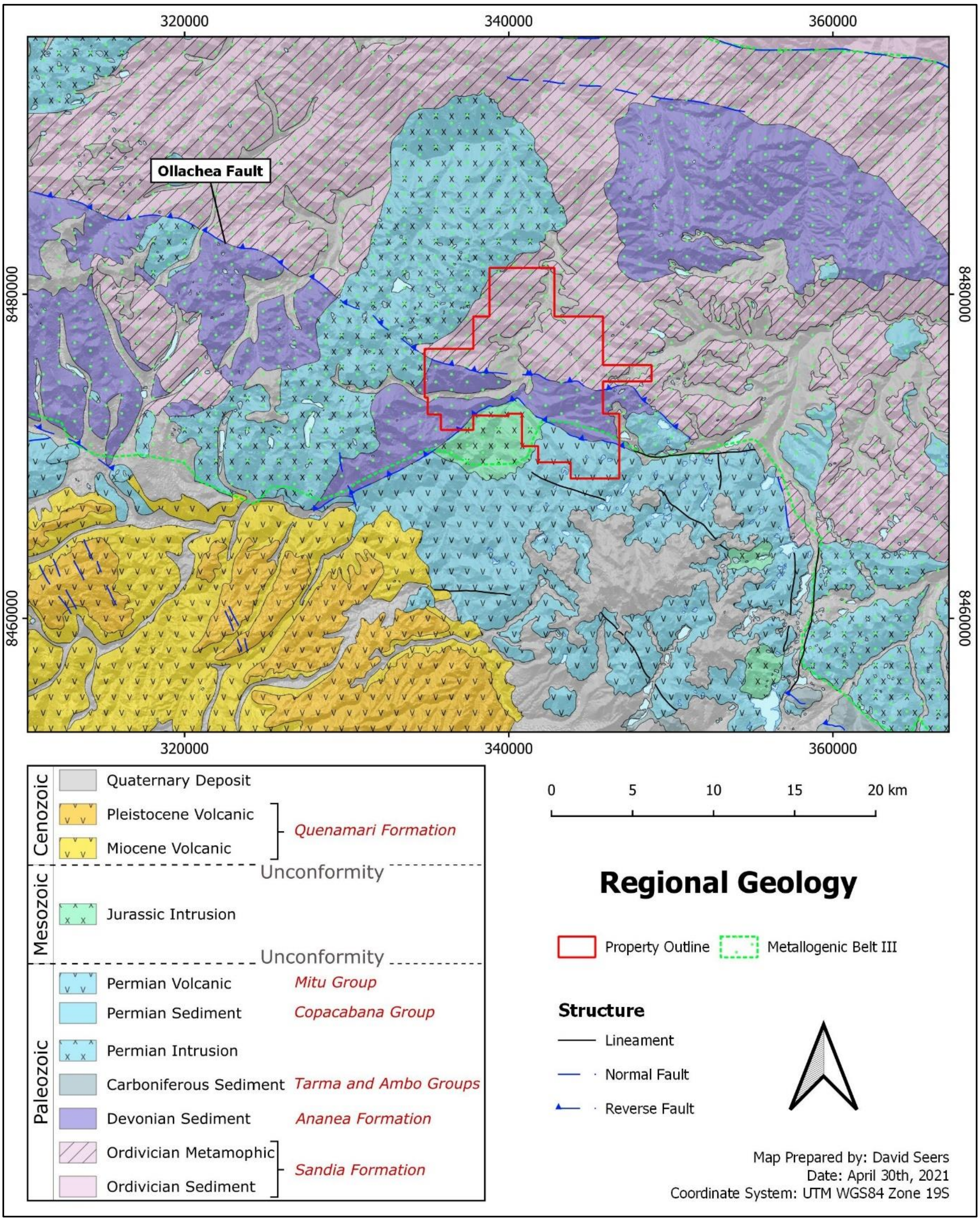

Figure 7-1: Regional Geology

7.2 Local Geology

Local geology is dominated by phyllites assigned to the Sandia Formation (Ordovician), lesser graphitic slates and shales of the younger Ananea Formation (Devonian) are also recorded. Zones of quartz veining are recorded in phyllites, veins trend approximately east west to east-northeast and dip moderately to the north. The distribution of quartz veining is most abundant between the Ollachea and Paquillusi faults (Figure 7-2).

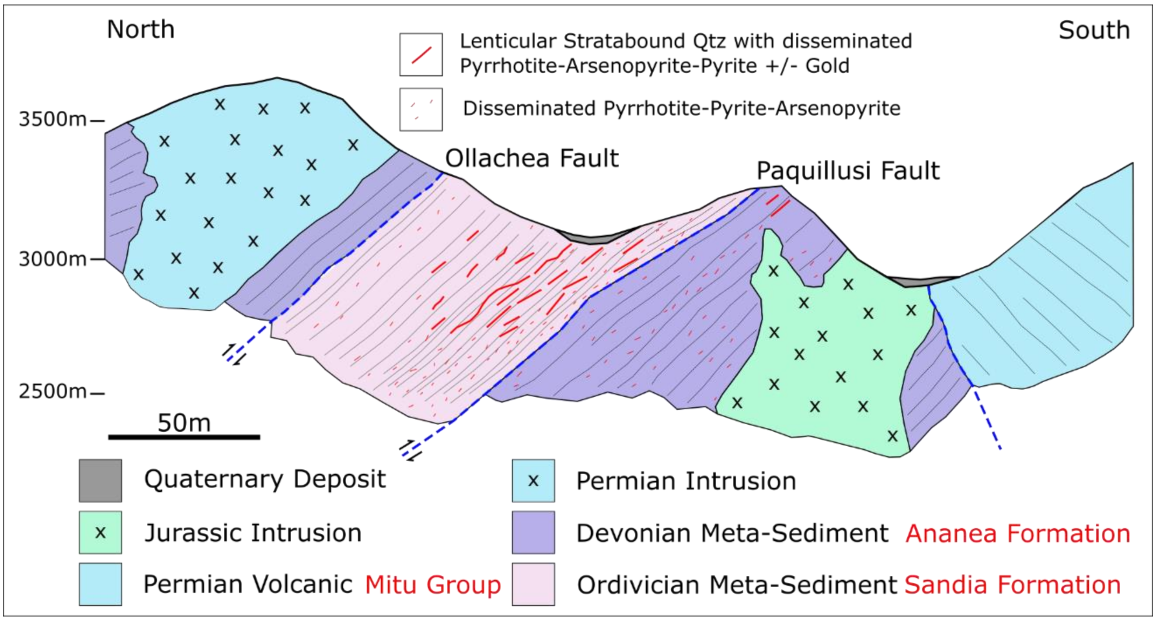

Figure 7-2: Schematic cross-section, north to south

7.3 Property Geology

Outcropping geology at the Property is dominated by sheared metasedimentary units, including phyllite, and graphitic slate and shale. Metasediments have a well-developed, east-northeast trending, and north dipping cleavage that approximately mirrors regional thrusting (Figure 7-3).

Intrusions of varied age and composition have interrupted metasedimentary units.

Metasedimentary units have been altered with weak but pervasive sericitization. Alteration is not apparently related to mineralization.

Gold mineralization at Minapampa is principally developed in phyllite between the Ollachea and Paquillusi thrust faults. The package of mineralized metasediments (Mineralized Package) is approximately 200m thick. Two zones of mineralization have been defined in the Mineralized package: Minapampa and Concurayoc (Figure 7-3).

Gold mineralization in the Mineralized Package is typically associated with pyrrhotite, arsenopyrite, pyrite, and minor chalcopyrite in quartz veins sub-parallel to cleavage, and as disseminations.

MKK has observed the following associations of sulfide with gold at Minapampa:

- When coarse pyrite occurs without other sulfides it is often indicative that gold is not present.
- Free gold is often observed in areas where coarse crystalline arsenopyrite is present.

Quartz veining in the Mineralized Package is not always auriferous, veins range from less than 5 mm to up to 40 cm across. Boudinage related to the thrust front has interrupted the continuity of quartz veining.

Mineralization at Minapampa has been traced at surface for 900 m along strike, and mineralization at the Concurayoc zone has been traced at surface for 400 m along strike. Drilling has traced mineralization to depths of up to 200 m from surface. The 900 m zone between Minapampa and Concurayoc is weakly mineralized.

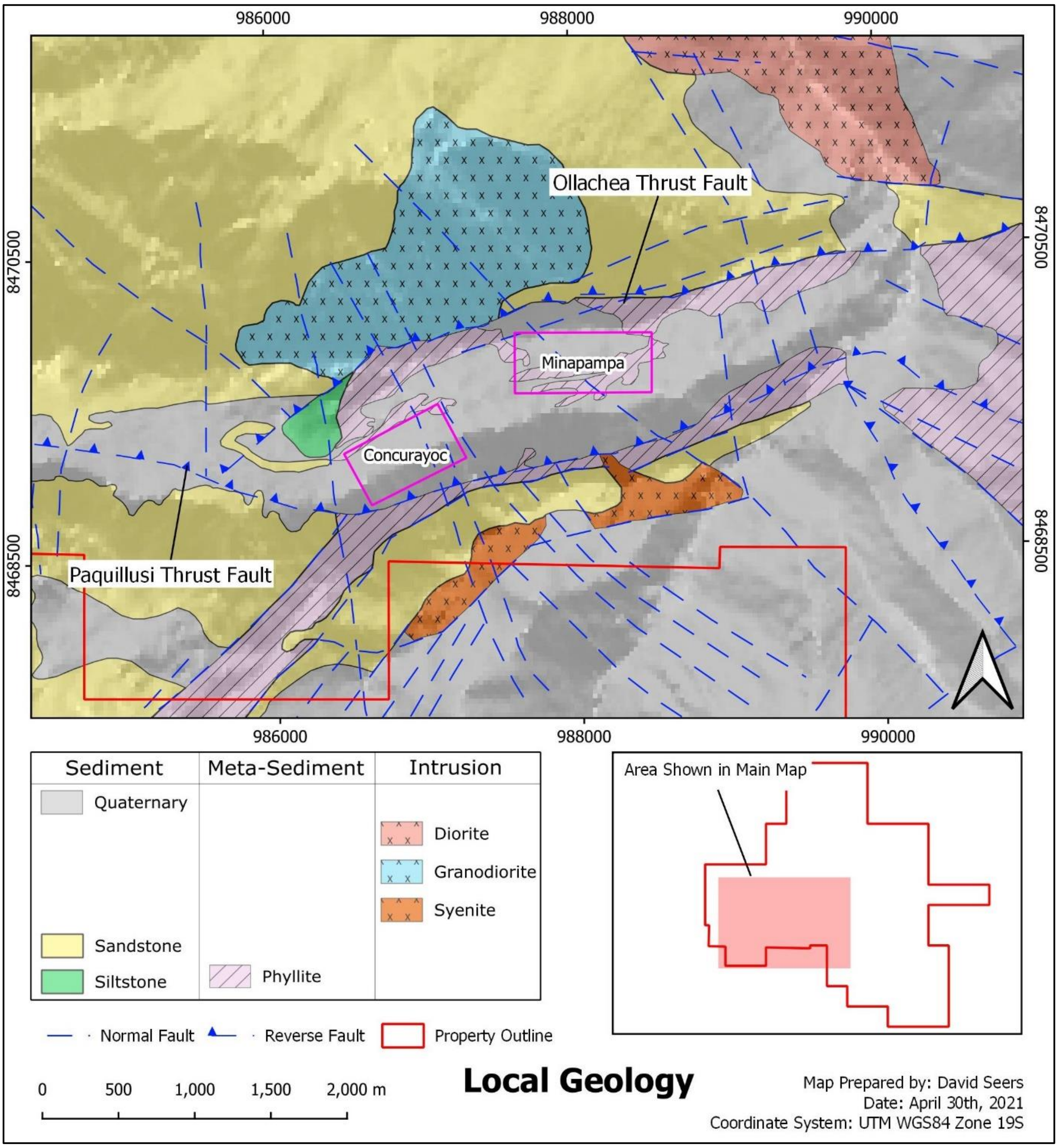

Figure 7-3: Property Geology

8 DEPOSIT TYPES

The Property hosts orogenic-style gold (Au) mineralization at the Property, and quartz veining in the Minapampa area is the principal area of interest.

Globally, orogenic deposits are a major source of gold and there are numerous examples of such deposits that host in excess of 10 Moz Au, such as Hollinger-McIntyre, Dome, Sigma Lamaque and Norseman. Orogenic gold deposits often form in clusters and are mined as both open-pit lower-grade high-tonnage mines and higher-grade lower-tonnage underground mines. Clusters of orogenic deposits frequently align along regionally significant structures.

Groves et al (1998) define Orogenic gold deposits as follows (Figure 8-1):

> *"Orogenic gold deposits are associated with regionally metamorphosed terranes of all ages. They form at convergent plate margins and are built by gold-bearing quartz veins, often with very simple mineralogy. They are characterized by a relatively high temperature and pressure of ore deposition which distinguishes them from a number of other types of gold deposits. Their fluids are also characteristic by increased CO2 content. In general, however, there is nogood single definition of these deposits."*

Orogenic Au deposits are often associated with greenschist to amphibolite grade metamorphism. Au is often concentrated in quartz lodes formed in brittle structures and can be associated with other elements such as Sb, As, Te and W.

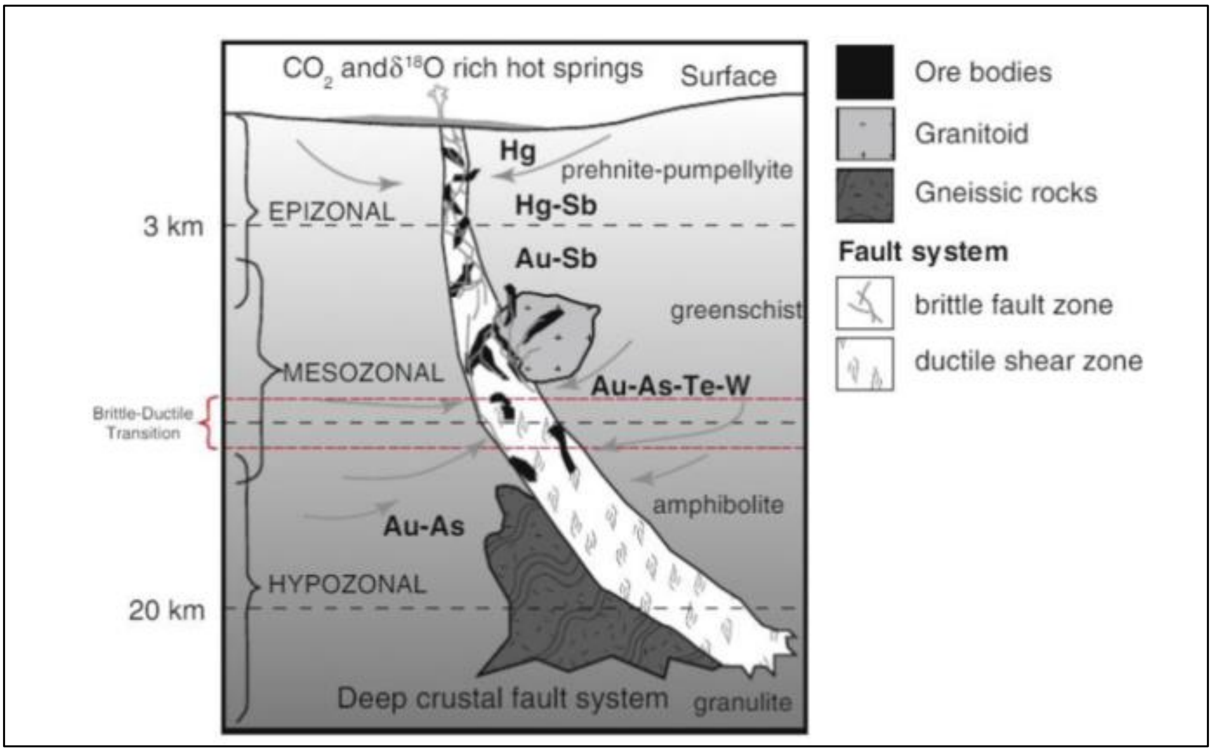

Figure 8-1: Typical cross-section of an Orogenic System (Groves et al. (1998)

9 EXPLORATION

The exploration methodologies employed by MKK at the Property encompass a range of industry standard techniques, including.

- Acquisition and structural interpretation of ASTER (Advanced Spaceborne Thermal Emission and Reflection Radiometer) imagery.
- Geological mapping over an area of approximately 784 hectares.
- Geochemical sampling; all samples have been submitted to an independent laboratory for fire assay (50g) gold, and multi-element (36) ICP analysis.
  - o 329 from artisanal mine workings, including, mineralization in outcrop, waste rock dumps, and tailings.
  - o 1312 grab and trench samples.
- 24.1 km ground magnetic survey (21 lines at 100m spacing) targeting pyrrhotite at depth in the Minapampa area.
- IP designed to indicate sulfide at depth has been tested at the Property but graphite content in sheared sediment masked the IP response to sulphide.

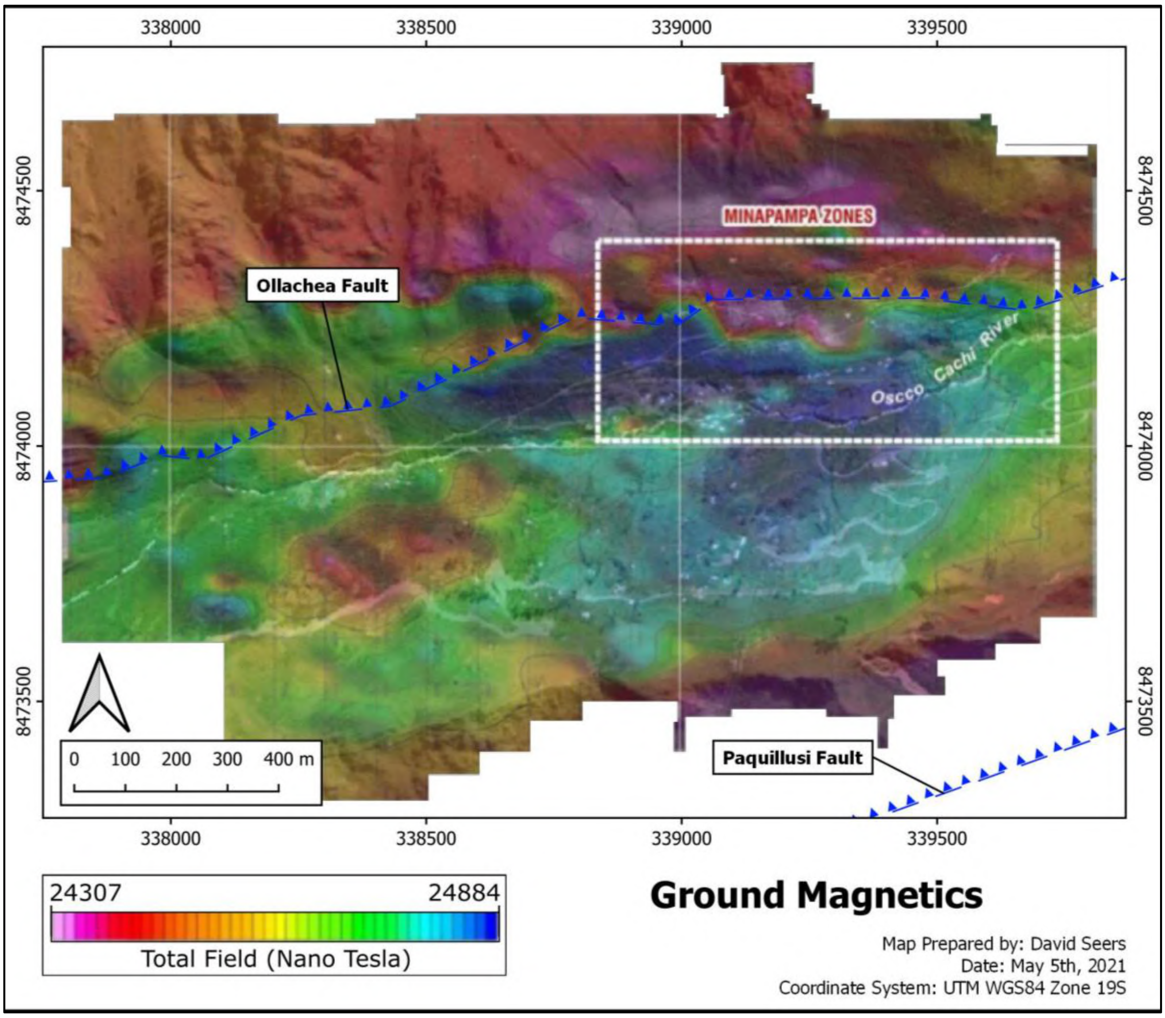

Figure 9-1: Ground Magnetic Survey at Minapampa Area

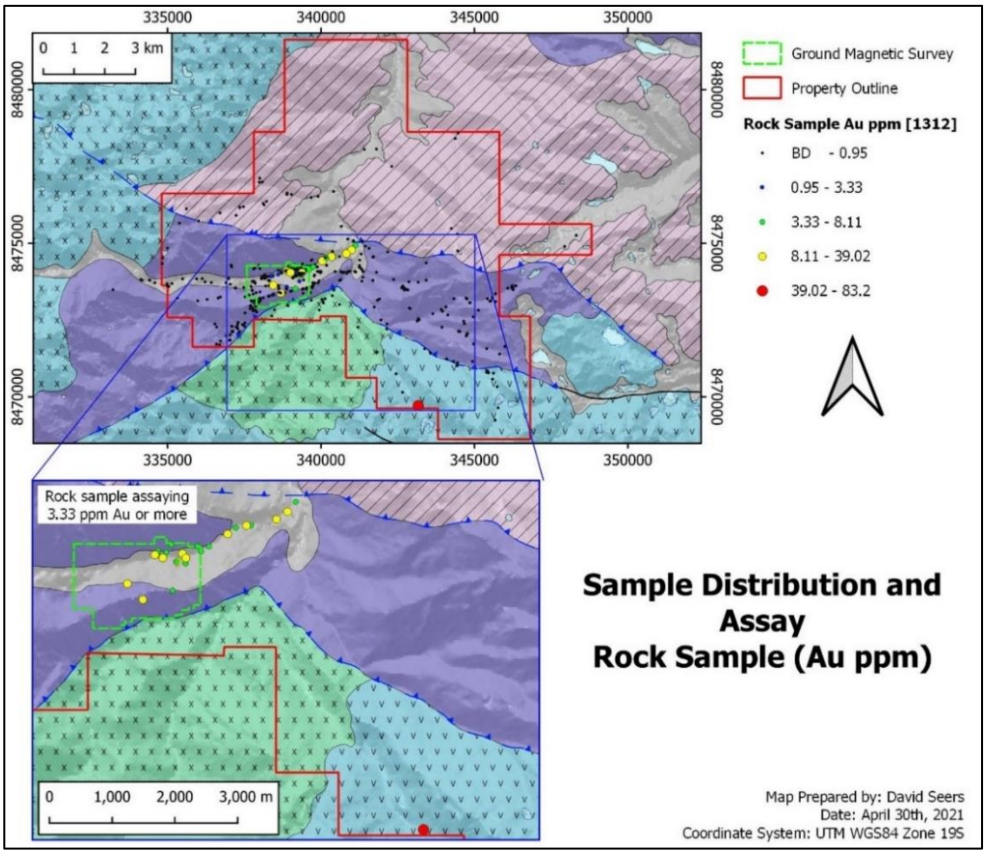

Figure 9-2: Grab and Trench Samples (Au ppm) shown with regional mapping

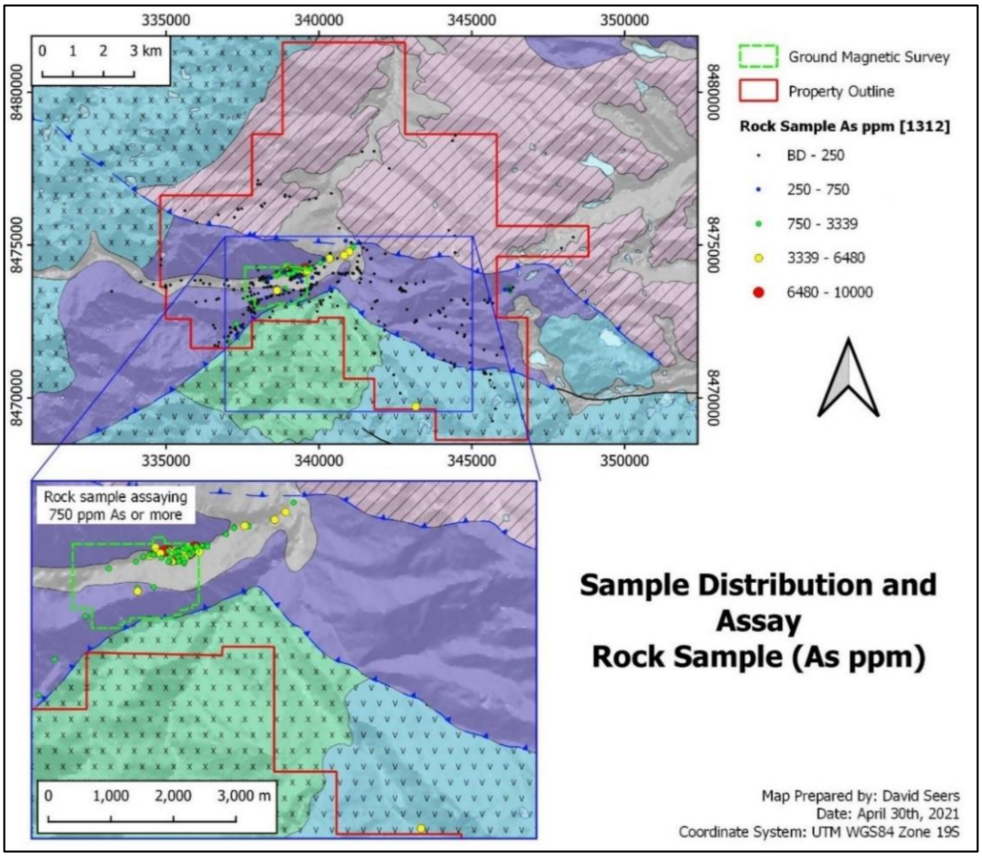

Figure 9-3: Grab and Trench Samples (As ppm) shown with regional mapping

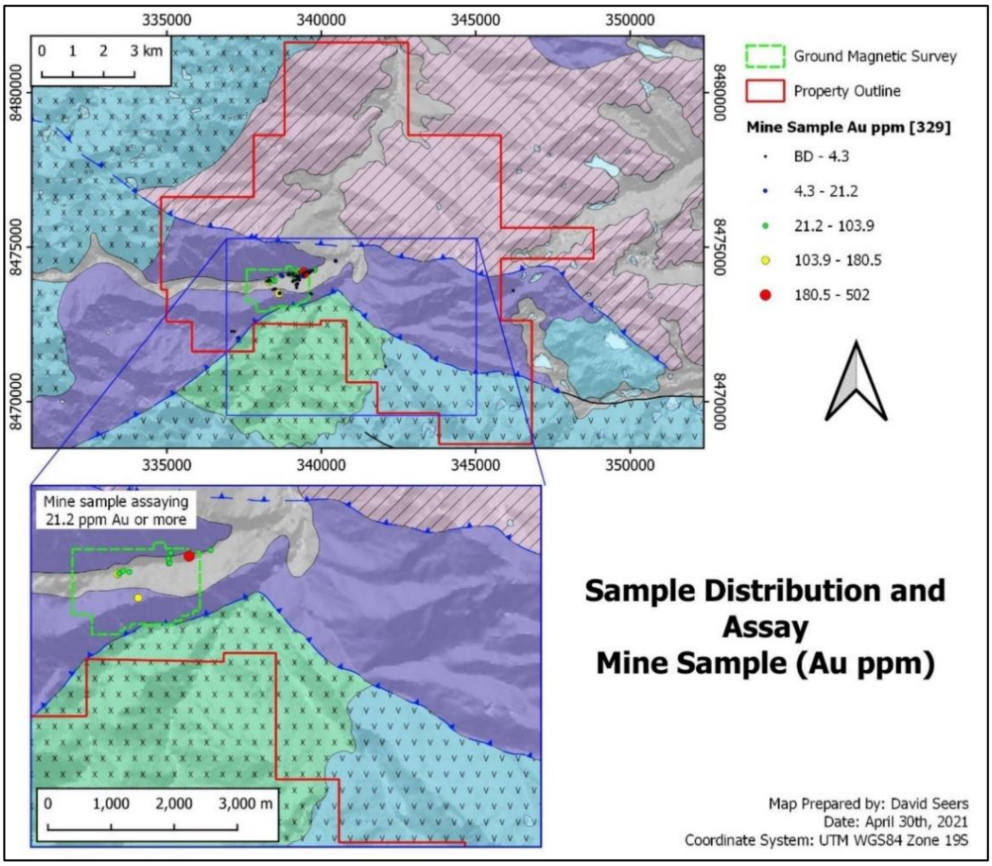

Figure 9-4: Samples from artisanal mine operations (Au ppm) shown with regional mapping

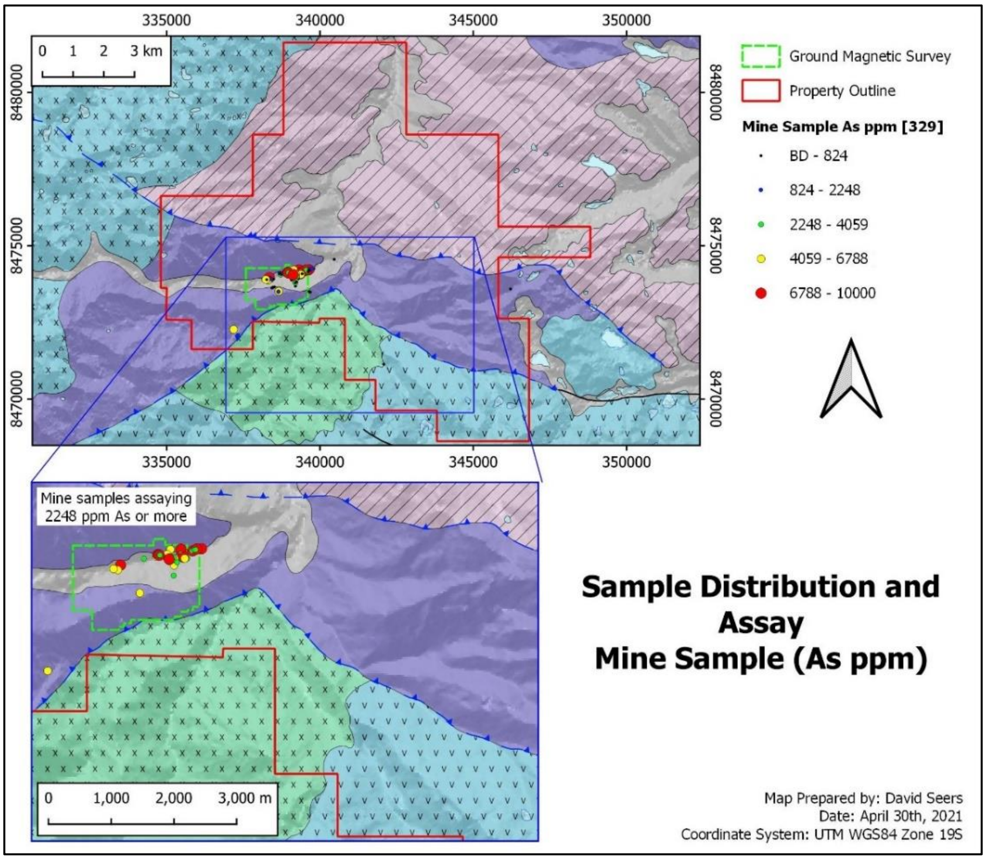

Figure 9-5: Samples from artisanal mine operations (As ppm) shown with regional mapping

Mr. Seers (QP) notes the following:

- The ground magnetic survey indicates that the Minapampa zone is spatially associated with an area with relatively high total field magnetism underlain by metasedimentary units (purple color).
- Assays from grab and trench sampling across the Property identified a northeast trending zone of gold and arsenic enrichment in metasedimentary units bound between two thrust faults. Enrichment of gold and arsenic is concentrated in the northeastern corner of the Minapampa Zone and extends beyond the zone further to the northeast.
- Samples from artisanal mine operations (including from waste rock dumps, tailings, and mineralized material) are concentrated in the Minapampa Zone. Enrichment of gold and arsenic from artisanal mine operations is coincident.
- All samples from artisanal mines operations are from areas underlain by metasediments.

10 DRILLING

10.1 Historical Drilling

Records indicate that Peruvian Gold drilled five diamond drill holes at the Property totalling 501 m between 1998 and 1999.

Details of these drill holes have been lost.

10.2 Drilling by the Issuer at Minapampa and Concurayoc

Doug Corley (QP) has reviewed records related to exploration diamond drilling by Minera Kuri Kullu S.A. (MKK) at the Minapampa and Concurayoc areas of the Property. Geotechnical and hydrological drilling completed for the 2012 Feasibility Study has not been reported here and has not been used in the resource estimate.

Drilling has been completed by a private drilling contractor.

Table 10-1 summarises drill hole count and meterage.

Table 10-1: Summary of MKK drilling at Minapampa and Concurayoc

Campaign	Hole Count	Meterage
2008	33	11,773.40
2009a	10	3,279.05
2009b	30	12,138.80
2010	52	20,636.55
2011	14	6,372.40
2012a	32	13,106.60
2012b	2	927.65
	174	**68,234.45**

MKK developed a geological and structural model (2014 Model) based on drilling data at Minapampa. The 2014 Model includes 64 approximately east-west trending, northerly dipping Au bearing lodes. Movement along the Oscco Cachi Shear ("OCS") has resulted in significant vertical offset of mineralized lodes.

Doug Corley (QP) has reviewed the 2014 Model, core logs, assay records and drill core and he is satisfied that the 2014 Model adequately reflects the geology of Minapampa.

Doug Corley (QP) is satisfied that the drilling contractor has used industry standard practises and that Issuer has adequately captured drilling data. Doug Corley (QP) does not believe that there are any drilling, sampling or recovery factors that could materially impact the accuracy and reliability of drill results.

10.2.1 Drilling Procedures

Drill holes are collared in HQ and reduced to NQ and BQ as conditions dictate.

Core recovery has been recorded for each core run and typically exceeds 95%. Recovery has been observed to reduce in zones of shearing, and zones of reduced recovery are typically less than 1m.

Collar location and mast orientation was recorded by MKK using total station instruments with a reported accuracy of +/- 0.5m. Based on survey data, MKK calculated collar orientation adjusted for magnetic declination.

Single shot and multi-shot downhole surveys were taken at approximately 20 m intervals using a REFLEX tool. The contractor supplied survey certificates to MKK, certificates for drill holes DDH08-01 and DDH08-02 have been lost. Based on survey data, MKK calculated downhole orientations adjusted for magnetic declination.

Whole core was routinely photographed.

MKK staff logged core for geological, geotechnical, and structural data.

Sample intervals were determined by trained geologists. Mineralized core was sampled in downhole intervals between 0.3 and 5 m, and non-mineralized core was sampled in downhole intervals between 2 to 5 m. The average sample interval recorded in the drill hole database is 1.33 m.

Prior to cutting by trained technicians using diamond core saw, sample intervals were marked on orientated core.

Drill holes typically intersect mineralization orthogonally and downhole intervals do not represent true mineral thickness. Downhole intervals can exceed true mineral thickness.

10.3 Drilling by the Issuer at Minapampa Far East

Dr. Andrew Fowler (QP) reviewed records related to exploration diamond drilling by MKK at the Minapampa Far East area of the Property.

Drilling has been completed by a private drilling contractor.

In 2013, MKK undertook a four (4) hole diamond core drilling in the Minapampa Far East area from the Ollachea Tunnel, completing 1202.5 m. Drilling was directed towards projected extensions of mineralization in the Minapampa area.

In 2016, MKK undertook a second program of twenty-two (22) diamond drill holes totalling 5146.9 m at the Minapampa Far East area from the Ollachea Tunnel. The drill program was designed to test projected extensions of mineralization at Minapampa.

Twenty-six (26) inclined drill holes totalling 6349.4 m have been completed from the Ollachea Tunnel (Table 10-2 and Figure 10-1).

Table 10-2: MKK Drilling (Minapampa Far East)

DDH ID	Year	Easting	Northing	Elevation	Length (m)
DDH13-T01	2013	339850.522	8474533.21	2776.701	407.05
DDH13-T02	2013	339850.189	8474533.55	2776.593	184
DDH13-T03	2013	339900.3	8474570.22	2775.82	290
DDH13-T04	2013	339850.276	8474539.43	2777.045	321.45
DDH16-T05	2016	339729.92	8474386.13	2780.309	170
DDH16-T06	2016	339817.75	8474478.42	2778.207	220
DDH16-T07	2016	339729.276	8474387.86	2779.729	190.75
DDH16-T08	2016	339817.75	8474478.42	2779.217	230
DDH16-T09	2016	339729.92	8474386.13	2781.159	170
DDH16-T10	2016	339817.408	8474479.36	2777.8	250.05
DDH16-T11	2016	339729.058	8474388.5	2779.729	178.1
DDH16-T12	2016	339775.618	8474435.98	2778.695	180
DDH16-T13	2016	339913.573	8474583.24	2775.658	264.8
DDH16-T14	2016	339775.58	8474436.08	2778.695	245.6
DDH16-T15	2016	339776.073	8474434.73	2780.159	200
DDH16-T16	2016	339913.792	8474582.46	2775.658	312.7
DDH16-T17	2016	339776.073	8474434.73	2779.029	170
DDH16-T18	2016	339868.025	8474532.96	2776.684	259.3
DDH16-T19	2016	339914.15	8474581.47	2775.658	264.4
DDH16-T20	2016	339868.305	8474532.19	2778.838	230.4
DDH16-T21	2016	339913.9	8474582.33	2775.658	252.3
DDH16-T22A	2016	339874.63	8474539.6	2776.137	210.3
DDH16-T23	2016	339950.269	8474618.61	2775.506	270.2
DDH16-T24	2016	340052.635	8474727.21	2772.2	290.1
DDH16-T26	2016	339949.721	8474620.11	2774.904	290.1
DDH16-T27	2016	340006.627	8474678.79	2773.522	297.8
					6349.4

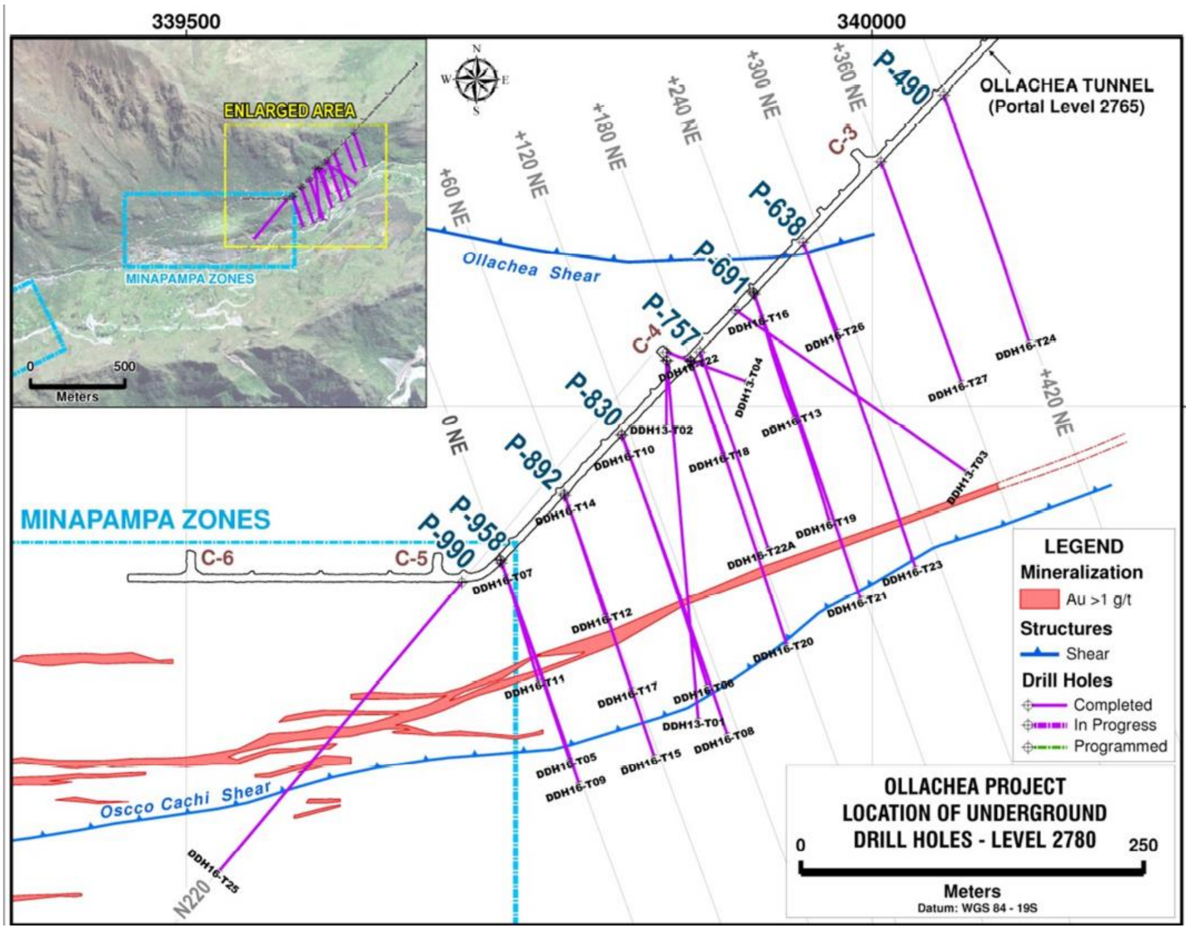

Figure 10-1: Plan view of drilling Minapampa Far East

Significant results from the 2016 Minapampa Far East drill program, included:

- DDH16-T06 8 m @ 3.69 g/t from 122 meters
- DDH16-T07 13 m @ 6.34 g/t from 143 meters
- DDH16-T10 4 m @ 11.23 g/t from 217 meters
- DDH16-T11 21 m @ 3.61 g/t from 96 meters
- DDH16-T12 18 m @ 3.4 g/t from 122 meters
- DDH16-T14 22 m @ 2.41 g/t from 176 meters
- DDH16-T16 4 m @ 16.8 g/t from 294 meters
- DDH16-T18 10 m @ 2.59 g/t from 186 meters
- DDH16-T18 10 m @ 2.65 g/t from 198 meters
- DDH16-T19 19 m @ 2.96 g/t from 188 meters
- DDH16-T24 18 m @ 2.1 g/t from 190 meters
- DDH16-T24 7 m @ 4.05 g/t from 213 meters

These results confirmed the continuation and projection of mineralization from Minapampa.

Core recovery was typically over 90%.

Dr. Andrew Fowler (QP) does not have any observations with respect to the drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of the results.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The following section on Sample Preparation, Analyses, and Security has been completed by Doug Corley (QP) and Dr. Andrew Fowler (QP), with the division of their respective responsibilities described below.

- Doug Corley (QP) has reviewed Sample Preparation, Analyses, and Security aspects relevant to exploration drilling at the Minapampa area (Section 11.1). Doug Corley's (QP) review includes samples from drill campaigns completed at Minapampa in 2008, 2009, 2010, 2011 and 2012.
- Dr. Andrew Fowler (QP) has reviewed Sample Preparation, Analyses, and Security aspects relevant to exploration drilling from Ollachea Tunnel in the Minapampa Far East area (Section 11.2). Dr. Fowler's (QP) review is relevant to drilling in 2013 and 2016.

Doug Corley (QP) is satisfied that Sample Preparation, Analyses, and Security procedures at Minapampa are adequate to ensure assay values suitable for use in the Mineral Resource Estimate ("MRE") reported in Section 14 of the Technical Report.

Dr. Andrew Fowler (QP) is satisfied that Sample Preparation, Analyses, and Security procedures at Minapampa Far East are adequate to ensure assay values suitable for use in the MRE reported in Section 14 of the Technical Report.

11.1 Minapampa

11.1.1 Core Sampling Methods

Sampling has been carried out using a series of different procedures since MKK began drilling at the Property. Sampling intervals have varied from fixed 2 m intervals within mineralized zones and fixed 5 m intervals outside mineralized zones to sampling intervals of a minimum of 0.5 m or 1.0 m with intervals determined by lithological contacts. In 2009, 2010 and 2012, re-sampling campaigns were undertaken such that all mineralized intervals were systematically sampled in intervals no longer than 2.0 m (2009), then intervals longer than 1.0 m have been re-sampled (2010, 2012). There is still a minor number of intervals longer than 1.0 m in the mineralized zone that are unable to be re-sampled as metallurgical sampling has used all remaining core.

The present sampling procedure requires that half-core samples of 1.0 m length be taken in mineralized zones recognized during the logging process. Core outside the 1.0 m sampling intervals but transitional to the visually identified mineralized zones, is half-core sampled on a 2.0 m sample length. Core interpreted to represent zones sterile of gold mineralization are quarter-sawn and sampled at 5.0 m intervals. Any intercept from the 2.0 m sampling which returns a greater than 0.5 g/t Au response, is re-sampled taking half-core samples, thus leaving no core remaining. If any assayed intercepts with greater than 0.5 g/t Au are encountered in the 5.0 m sampling intervals, these intervals are re-sampled by taking half-core samples at 1.0 m intervals, thus leaving quarter-core remaining.

Drill core is split using a diamond core saw. Samples are numbered and collected in individual plastic bags with sample tags inserted inside as well as being stapled to the outside of the bag.

The sampling is of industry standard and is considered adequate for use in the mineral resource estimate.

11.1.2 Laboratory Sample Preparation

MKK has used the independent Certimin (previously known as CIMM) Peru laboratories as the primary laboratories for preparation and assaying of drill core samples from Ollachea since the MKK 2008 drill campaign. Certimin Peru has the System of Quality Management ISO 9001:2008 certification "System Management Quality" and is accredited with NTP-ISO/IEC 17025:2006 certification "General Requirements for the Competence of Testing and Calibration Laboratories", for the preparation and assay of geochemical and metallurgical samples.

The Certimin sample preparation laboratory in Juliaca prepared the drill core samples for the Ollachea Project under the following procedure:

- Samples are sorted and dried in an electric oven at temperatures not exceeding 105˚C for at least four hours or until dried.
- Samples are crushed by two jaw crushers followed by a roll crusher to 2 mm. The full sample is riffle split to 500 g.
- A 500 g pulp is prepared in LM2 pulveriser bowls to 85% < 75 µm (200 mesh). 50 g pulps are submitted for chemical analysis.

11.1.3 Sample Analysis

Chemical analysis was conducted at the Certimin Lima laboratory and consisted of fire assay (FA) with atomic absorption spectrometry (AAS) finish on the 50 g pulp aliquot. A 32-element suite was also analyzed by ion-coupled plasma optical emission spectroscopy (ICP-OES) until the end of 2009 but was discontinued once sufficient analyses had been obtained from the initial nominal 100 m grid pattern.

Serious deficiencies with sample preparation practices at the Juliaca laboratory were identified by Smee (Smee 2009, and Smee 2011):

- The crushers were examined and both showed that the dust extraction pipe was connected directly to the rear of the crushers rather than the rear of the dust enclosure. This can create a sample bias.

- The pulveriser only handles 250 g at a time and 500 g is pulverized. These pulverisers need replacing.
- Sample drier has racks rather than wheel-in trolley access.

Between January 2012 through to March 2012, MKK sent all samples directly to Certimin Lima, for preparation and analysis. This practice ended once the Juliaca laboratory demonstrated it had corrected the serious deficiencies identified by Smee.

Reported improvements to the Juliaca laboratory included:

- Upgrading the pulverising unit to a COSAN TM, LM2 model.
- Pulveriser bowls have been upgraded to a Labtechessa B500 type, so they can handle the 500g pulverisation in one pass.
- Dust extraction unit; the pipe is no longer attached directly to the crusher (installed a plenum-style dust control system).
- Wheel-in trolley access sample drier has been installed.

MKK used three laboratories for secondary analysis during the 2008 to 2012 drilling campaigns.

- BSI Inspectorate laboratories, certified under ISO65 and certAll
- ALS Chemex Lima, certified under ISO 9001:2008, ISO 17025:2005 and IQNet
- Actlabs, Chile, certified under ISO 9001:2008, ISO 17025:2005.

11.1.4 Sample Security

Doug Corley (QP) considers that the sample preparation and security are adequate and appropriate for use in Mineral Resource estimation.

11.1.5 Adequacy of Procedures

Doug Corley (QP) considers that the procedures are adequate and appropriate for use in Mineral Resource estimation.

11.2 Minapampa Far East

Dr. Fowler (QP) reviewed sampling protocols used by MKK during the 2016 drilling campaign at Minapampa Far East. He noted that Sample Preparation, Analyses, and Security could be improved. Notwithstanding this, the procedures used by MKK during drilling at Minapampa Far East are adequate for use in Mineral Resource estimation and the NI 43-101 code.

11.2.1 Core Sampling Methods

Once logged, drill core was cut using diamond saw by trained technicians at MKK's core logging facility in Juliaca.

Half core samples were bagged for analysis at an independent laboratory. The remaining half core was stored in core boxes.

11.2.2 Laboratory Sample Preparation

Samples were dispatched to Certimin for analysis. Samples were prepared by Certimin at its sample preparation facility in Juliaca as described above.

11.2.3 Sample Analysis

Certimin dispatch prepared samples for analysis at its analytical laboratory in Lima as described above.

11.2.4 Sample Security

Samples prepared by MKK were kept under 24 hour a day security until delivery to Certimin.

Certimin has stated to Dr. Fowler (QP) that they have never experienced any issues with lost or damaged samples in transit between laboratories.

12 DATA VERIFICATION

Verification of data used in the Mineral Resource Estimate ("MRE") (Section 14) has been completed by Doug Corley (QP) and Dr. Andrew Fowler (QP). Division of their responsibilities has been described below.

- Doug Corley (QP) has verified data related to the Minapampa area (as reported in Section 12.1 and 12.2 of the Technical Report), his verification included:
    - Review of the database to compare recorded data against original certificates for collar surveys, downhole surveys, and assays.
    - Comparison of drill core with drill logs.
    - Visiting drill collars with handheld GPS and cross checking against database records.
- Dr. Andrew Fowler (QP) has verified data generating and recording practices related to the Minapampa Far East area (as reported in Section 12.3 of the Technical Report), his verification included:
    - Review of written protocols covering aspects from core drilling to sample dispatch.
    - Discussion of written protocols with MKK technicians and geologists.
    - Review of MKK's core logging facilities in Juliaca and the Certimin sample preparation laboratory in Juliaca.

Doug Corley (QP) is satisfied that data used in the MRE meets the requirements for NI 43-101.

Dr. Andrew Fowler (QP) is satisfied that data used in the MRE meets the requirements for NI 43-101.

12.1 Minapampa

Doug Corley (QP) conducted the data verification for Minapampa.

12.1.1 Database and Certificates

Collar Records

The following validation was undertaken on all Diamond Drill Holes ("DDH") used for the resource estimate:

- All DDH names within the collar, survey and assay tables were reviewed and cross-checked to ensure the same naming convention was used throughout the database.
- The collar data was reviewed for missing easting, northing, reduced level ("RL") or total depth entries for each drill hole and to verify that each drill hole had survey and sample data associated with it.

- The collar data supplied was compared with the collar data used in previous model. This was to check for missing and / or new drill holes, and if there were any significant differences in collar location and depth of hole.
- The collar RL's were compared to the topographic survey file provided, to check for any discrepancies.

Downhole Survey

After filtering the supplied survey data to ensure exclusion of irrelevant drill hole data, the resulting data was reviewed and validated. The following steps were undertaken as part of the review and validation process.

Sixty-three (63) downhole surveys (Table 12-1) recorded in the database have been cross-checked against pdf records of reflex downhole survey. These 63 DDH consisted of the first 56 DDH drilled (not including DDH08-01 and 02, which do not have downhole survey data) and 7 other DDH selected at random. The validation included the following:

- Verification of depth, azimuth, and dip and that the azimuth was appropriately corrected for magnetic declination
- If a survey reading was excluded, it was cross-checked the data with the supplied downhole survey validation report completed by MDH SAC (report 07.0412.01) for MKK, to validate the reasons for the exclusion.

Downhole survey data supplied was also compared to survey data used in the AMEC 2012 Mineral Resource estimate. This was to check for missing and / or new downhole surveys, and if there were any significant differences in downhole survey data.

Table 12-1: Downhole Survey - Verified Drill Holes

DDH ID	DDH ID	DDH ID
DDH08-03	DDH09-26	DDH09-48
DDH08-04	DDH09-27	DDH09-49
DDH08-05	DDH09-28	DDH09-50
DDH08-07	DDH09-29	DDH09-51
DDH08-08	DDH09-30	DDH09-52
DDH08-10	DDH09-31	DDH09-53
DDH08-12	DDH09-32	DDH09-54
DDH08-16	DDH09-33	DDH09-55
DDH08-17	DDH09-34	DDH09-56
DDH08-18	DDH09-35	DDH09-57
DDH08-21	DDH09-36	DDH09-58
DDH08-22	DDH09-37	DDH09-59
DDH08-23	DDH09-38	DDH09-60
DDH09-06	DDH09-40	DDH09-61
DDH09-09	DDH09-41	DDH09-73
DDH09-11	DDH09-42	DDH10-112
DDH09-13	DDH09-43	DDH10-82
DDH09-19	DDH09-44	DDH10-85
DDH09-20	DDH09-45	DDH11-153
DDH09-24	DDH09-46	DDH11-192
DDH09-25	DDH09-47	DDH12-203

Assay

The supplied assay data was filtered to exclude the holes that were included in the Mineral Resource Estimate reported in Section 14 of the Technical Report. Validation of the 100 selected samples comprised cross-checking the supplied laboratory certificates to ensure that the Au grade in the database matched with the reported laboratory results.

The assay data was also cross-checked with the assay data used in the previous resource estimate to check for any material changes or discrepancies between the data sets. Table 12-2 provides a list of samples that have been validated.

Table 12-2: Data Validation - Verified Sample Data

DDH ID	Sample ID
DDH08-01	G-08-97482
DDH08-01	G-08-90305
DDH08-01	G-08-87532
DDH08-01	G-08-98404
DDH08-01	G-08-81875
DDH08-01	G-08-98922
DDH08-01	G-08-96563
DDH08-01	G-08-72672
DDH08-01	G-08-86289
DDH08-01	G-08-81084
DDH08-01	G-08-86957
DDH08-01	G-08-95979
DDH08-01	G-08-71361
DDH08-01	G-08-85398
DDH08-01	G-08-90929
DDH08-01	G-08-89778
DDH08-01	G-08-70487
DDH08-01	G-08-84995
DDH08-01	G-08-95502
DDH08-01	G-08-94288
DDH08-01	G-08-81779
DDH08-01	G-08-88722
DDH08-01	G-08-72320
DDH08-01	G-08-80551
DDH08-01	G-08-99349
DDH08-01	G-08-90696
DDH08-01	G-08-71572
DDH08-01	G-08-83278
DDH08-01	G-08-96067
DDH08-01	G-08-88421
DDH08-01	G-08-97414
DDH08-01	76374
DDH08-01	76375
DDH08-01	76376
DDH08-01	76377
DDH08-01	76378
DDH08-01	76379
DDH08-01	76381
DDH08-01	76382
DDH08-01	76383
DDH08-01	76384
DDH08-01	G-08-77750
DDH08-01	76385
DDH08-01	76386
DDH08-01	G-08-80454
DDH08-01	G-08-70355
DDH08-01	G-08-81210
DDH08-01	G-08-77937
DDH08-01	G-08-80761
DDH08-01	G-08-80818
DDH08-10	G-08-1728302
DDH08-22	G-08-71820
DDH09-06	G-09-6794748
DDH09-19	75605
DDH09-25	G-09-3620093
DDH09-27	G-09-10255447
DDH09-28	76586
DDH09-30	G-09-10543658
DDH09-30	G-09-11258768
DDH09-31	74818
DDH09-37	G-09-13854834
DDH09-46	47728
DDH09-52	49630
DDH09-56	51635
DDH09-62	53456
DDH09-78	58985
DDH10-103	65440
DDH10-109	67038
DDH10-115	67651
DDH10-120R	4291
DDH10-133	70367
DDH10-82	74300
DDH10-85	74629
DDH10-89	62126
DDH10-91	62325
DDH10-94	63662
DDH10-96	63780
DDH10-99	64166
DDH11-140	73001
DDH11-146	77551
DDH11-151	78317
DDH11-175	83600
DDH11-176	85133
DDH11-182	85714
DDH11-186	86261
DDH11-186	86263
DDH11-190	86671
DDH11-192	87334
DDH12-194	87807
DDH12-197	89171
DDH12-198	88553
DDH12-199	89548
DDH12-200	90130
DDH12-202	89786
DDH12-204	91010
DDH12-204	91122
DDH12-206	91430
DDH12-207	90944
DDH12-208	91595
DDH12-208	91789

The review of the data undertaken on all filtered assay data consisted of the following:

- Ensure that there were no overlapping sample intervals.
- Check for missing sample intervals. If missing sample intervals were located, a sample interval was created for the hole that covered the missing extent. For each interval added, the following rules were applied:
  - Sample ID = NS
  - Au grade = -999

- Verify that sample intervals are correct.
- Check that sample depth is not greater than total hole depth.

Validation of the 100 selected samples comprised cross-checking the supplied laboratory certificates to ensure that the Au grade in the database matched with the reported laboratory results.

The assay data was also cross-checked with the assay data used in the previous resource estimate to check for any material changes or discrepancies between the data sets.

During modelling, it was noted that DDH12-195 contained six sample intervals that had sample ID's but no assay results. This information was cross-checked with the original supplied data; a previous database titled "Consulta1-assay-ddh.xlsx" supplied by client; and with the database used for previous Mineral Resource estimate. It was found that these six sample intervals were not included in "Consulta1-assay-ddh.xlsx" nor were the sample ID's allocated to other sample intervals. However, the database used previously had the sample intervals relating to the missing assay results as not sampled. The list of sample intervals and associated sample ID's is provided in Table 12-3.

Table 12-3: Sample ID with Missing Assay Results

Hole ID	From (m)	To (m)	Sample ID
DDH12-195	130	135	88875
DDH12-195	170	175	88884
DDH12-195	196	198	88891
DDH12-195	200	202	88893
DDH12-195	244	245	88930
DDH12-195	245	246	88931

No material errors were identified by Doug Corley (QP) in the data included in the validation; however, some minor discrepancies between the previous resource estimate data and the current data were identified. The high-level checks indicate that the drilling database provided appears suitable for use in the Mineral Resource estimate.

12.1.2 Field Review

The following checks were undertaken by Doug Corley (QP) during his site visit:

Collar

Several drill collars have been visited in the field and their location reviewed using hand-held GPS. All visited drill holes checked are within +/- 5 m of the reported survey location (within the accuracy limits of the device).

Geological Logging

Randomly selected core has been compared against geological logs, and reviewed drill logs adequately reflect drill core. No significant discrepancies or inconsistencies have been identified.

12.2 Minapampa Far East

Dr. Andrew Fowler (QP) conducted the data verification for Minapampa Far East.

In September 2016, Dr. Andrew Fowler (QP) undertook a site visit to review geological data capture practices used by MKK for drilling at Minapampa Far East. This drilling was completed from the Ollachea Tunnel.

Dr. Fowler (QP), notes the following:

- Written protocols have been prepared for many exploration tasks, from drilling until sample dispatch to the laboratory. Protocols are aligned to "CIM Exploration Best Practice Guidelines" and "CIM Estimation of Mineral Resources and Mineral Reserve Best Practice Guidelines"
- MKK Technicians and Geologists are familiar with and follow established protocols
- MKK Technicians and Geologists have sufficient training and experience in the capture and interpretation of geological information related to orogenic/shear hosted mineralization
- Geologists demonstrate a good understanding of mineralizing controls, and numerous visual indicators of Au mineralization are recognized, for example: increased vein frequency, grey quartz, presence of arsenopyrite, folding, and rock type
- A core library of rock types and established descriptions is not available at the Property
- Drill holes are not cemented at the Property
- To reduce oxidation of core, the most significant drilled intercepts are stored in refrigerated containers. Remaining core is stored according to standard industry practices
- Samples are guarded 24 hours a day until delivery to the laboratory
- QAQC protocols are consistent with "CIM Exploration Best Practice Guidelines" and "CIM Estimation of Mineral Resources and Mineral Reserve Best Practice Guidelines". Notwithstanding this, Dr. Fowler (QP) notes that neither the re-testing of the pulps nor the submission of duplicates to an umpire laboratory are included in the QAQC protocols
- Certimin's office in Juliaca is clean and orderly and their operational procedures follow industry standards
- Certimin has established security controls at the Juliaca (preparation) and Lima (analysis) laboratories. Certimin states that they have never had any issue with lost or damaged samples
- MKK does not undertake random inspections of the Certimin laboratories

- Dr. Fowler was unable to establish the following:
    - Who at Certimin was the responsible for chain of custody of results
    - Who at Certimin was responsible for continual Quality Control
    - Who at MKK was responsible for the database and its validation.

Dr. Fowler (QP) recommended the following improvements:

- Additional written protocols should be developed for:
    - Core photography
    - Recording of chain of custody for samples submitted to the laboratory
    - Continual assay control
    - Back up and validation of the master database
- Descriptions of rock type, alteration, mineralization, and all other characteristics should be documented and illustrated with photos. These descriptions should be printed and clearly visible in core logging areas. Doing this will ensure that all geologists have same non-ambiguous mental image of the important characteristics of mineralization, and this will help deliver more consistent standardized logging
- A “Mineralization Index” should be developed to help predict gold grade based on various visible mineralization indicators
- Random checks should be carried out at the Certimin preparatory laboratory in Juliaca, and analytical laboratory in Lima. Conducting random checks should encourage Certimin to maintain standards as they are aware that they are being monitored and could be inspected at any moment
- Once complete, drill holes should be cemented to avoid potential future mining dangers. Abandoned drill holes should be cemented too. This is particularly important for drill holes located at topographic low points that may increase the risk of the mine flooding, or drill holes that could eject waste during blasting
- The QAQC protocol could be improved by including the re-analysis of pulps and using an umpire laboratory for the analysis of duplicate samples. Notwithstanding this, the QAQC protocol ensures assays were adequate.

Dr. Fowler is unaware if these recommendations were acted on after his visit; nonetheless, the geological data were still considered of sufficient quality for use in Mineral Resource estimation in accordance with NI 43-101.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

John Thomas (QP) is responsible for the content of Section 13 of the Technical Report.

13.1 Historical Test Work

2009 - Kappes Cassidy and Associates

Test work has been carried out in a series of programs, starting with an investigation in 2009-2010 by Kappes Cassidy and Associates ("KCA 2009"). KCA 2009 demonstrated that the mineralized material was refractory to direct cyanide leaching due to preg-robbing by organic carbon in the ore. Initial gravity concentration tests showed that some gold could be recovered, and flotation also recovered gold, but the mass pull of flotation concentrate was high, and the preg-robbing carbon reported to the flotation concentrate. Neither of these processes gave a recovery of gold considered to be attractive. Leaching in the presence of activated carbon (CIL) was shown to be reasonably successful.

2011 Ammtec

Ammtec undertook further test work on a mineralized material composite in 2010 and 2011 ("Ammtec 2011"). Ammtec 2011 investigated the removal of carbon by flotation with low gold loss and flotation followed by leaching the flotation concentrate gave overall recoveries lower than whole mineralized material CIL leaching; this approach was considered unsuccessful. Optimization of the CIL process showed that addition of kerosene in an attempt to render the organic carbon less active in the CIL gave some increase in recovery. The work also showed that a grind to a P_{80} of 75 microns was required to achieve recoveries greater than 90%.

Additional tests carried out on individual samples taken from around the mineralized area showed that lower extractions were obtained from other samples with an average of only 80%, and that in some cases a leach time of 48 hours was required.

A further program of test work was carried out by Ammtec using a composite made up of drill core from three mineralized zones; the composite graded 3.3 g/t gold. A mineralogical analysis showed that pyrrhotite was the predominant sulfide present, with the gold occurring with this sulfide and lower amounts of pyrite, arsenopyrite and arsenopyrite/pyrrhotite.

A slightly coarser grind was found to still be effective, stated to be optimal conditions, and have been reported in Table 13-1.

Table 13-1: DFS Optimisation Test work Optimal Conditions

Parameter	Unit	Value
Grind Size P80	µm	106
Pulp Density	% w/w	42
pH	-	10.5
Carbon Population	g/L	10
Source of Oxygen	-	Oxygen Injection
Kerosene conditioning	kg/t	0.1
Temperature	°C	Ambient
Cyanide concentration - Initial	% w/v	0.1
Cyanide concentration - Maintained for 36 h	% w/v	0.1

The flow sheet used incorporated low mass pull gravity recovery and gravity extractions of 40 – 45% were obtained. This concentrate was upgraded on a shaking table and a smeltable concentrate was made with a 90% recovery. Leaching for 48 hours with kerosene additions raised the overall extraction to 80 – 85% with cyanide consumptions between 1 and 1.5 kg/t. The use of elevated temperatures was tested, and higher extractions were obtained but at very high cyanide consumption.

An alternate flow sheet was proposed where a low mass pull gravity concentration would be used to recover the free gold, followed by a high mass pull gravity concentration ("HMPG") which would concentrate the sulfides and gold. The HMPG concentrate would contain little preg-robbing native carbon and should leach well. Finally, the tailings from the HMPG would be leached using a CIL circuit with the addition of kerosene. Some improvement in overall extraction was found but cyanide consumption was higher than that found with the standard CIL flow sheet. Subsequent tests using a pre-aeration treatment of the leach slurry to lower the level of soluble iron compounds was shown to decrease cyanide consumption.

Variability testing using this flow sheet showed that extractions as high as 95% could be obtained when the head grade was high (4-5 g/t Au) but some samples with grades of 0.9 – 2 g/t Au gave overall extractions in the range 66-90%. Over the whole range of head grades, from 0.9 to 10.7 g/t Au, for 27 samples, the recovery to the two gravity concentrates was 71%. CIL processing of the gravity concentrate was stated to result in a final leach residue analysing 0.3 g/t Au. Using the results of the variability test work, the relationship between organic carbon content (the preg-robbing component) and final leach residue grade was established as shown in Figure 13-1. This shows that lowering the carbon content will result in higher leach recoveries.

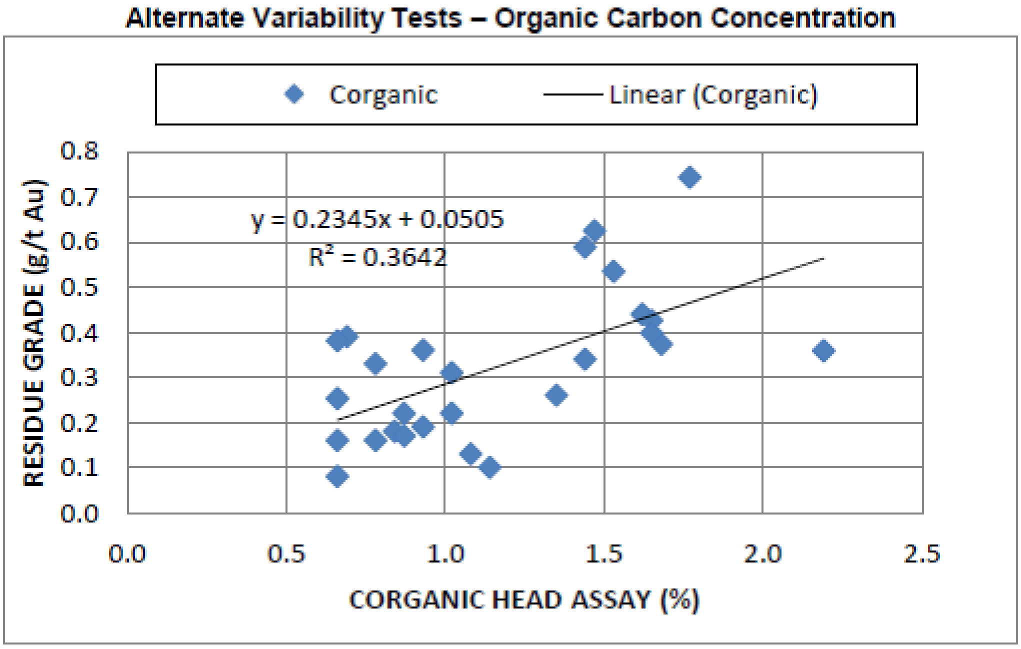

Figure 13-1: Relationship between organic carbon content (corganic) and final leach residue grade

Test work on effluent treatment used the SO_2 air process for cyanide destruction. The high levels of iron cyanide compounds (105 mg/l iron in treated solution) would give a high total cyanide concentration, which was calculated to be 299 mg/l. The Peruvian regulations call for a level of <0.8 mg/l total cyanide, so this would not allow discharge to the environment. Subsequent treatment with zinc sulfate was shown to precipitate the iron very effectively and an acceptable level of total cyanide was obtained, although it appeared that removal of residual zinc may be necessary.

Comminution Test Work

Bond Ball, Rod and Abrasion indices were measured for a master composite and several variability samples. The results are shown in Table 13-2.

Table 13-2: Historical Comminution Test Work

	DFS			Historic (KCA and PFS)		
	BWI (kWh/t)	RWI (kWh/t)	Ai	BWI (kWh/t)	RWI (kWh/t)	Ai
Samples Tested	6	2	6	5	5	1
Min	18.6	22.1	0.009	18.0	22.0	0.012
Max	22.7	22.5	0.022	19.9	25.1	0.012
Average	20.2	22.3	0.017	18.8	23.4	0.012
Standard Deviation	1.7	0.3	0.006	0.9	1.3	-
80th Percentile	20.6	23.7	-	19.5	24.1	0.012

2013 Outotec

Outotec undertook a program to investigate continuous thickening, filtration, and rheology in 2013 ("Outotec 2013"). Outotec 2013 undertook continuous thickening tests that indicated a flux of 0.75 t/h/m^2 could be obtained with a 15 g/t flocculent addition with an underflow concentration of 58% solids. It was noted that the overflow was of poor clarity and a pH of 11.1 was necessary (with lime addition) to obtain a clear overflow. These results were obtained using the master composite leach residue, which had a P_{80} of 106 microns. Earlier work with a P_{80} of 75 microns showed a lower flux of 0.25/t/h/m^2.

Pressure filtration tests gave a rate of 228 kg/h/m^2 with a cake moisture of 14.8%.

Rheology test work showed that for slurries at 9 and 10.5 pH, no constraints should be found for slurries in the 40 – 60% w/w range, and at 40% solids the viscosity was less than 60 cPS.

13.2 Gravity Concentration Test Work

Although previous investigation indicated relatively good recoveries, it was decided to investigate the possibility of using a gravity only process. A gravity only process would decrease overall capital and operating costs. Particular emphasis was put on mineralized material from the Minapampa high-grade zone, with an estimated grade of 4.65 g/t Au, as this could be mined first. Several laboratories carried out the work and each one is described below.

13.2.1 Plenge Laboratory - (Peru)

In July 2017, Plenge Laboratories in Lima undertook a gravity concentration test on a 43.5 kg composite sample considered with a grade of 4.96 g/t Au (Plenge 2017).

Plenge 2017 performed the following tests in sequential order:

1. Chemical characterization of the sample
2. Gravimetric concentration
3. Grinding.

Plenge 2017 results are summarized as follows:

- The ROM grade was 4.96 g/t Au with 0.6 g/t Ag, total carbon of 0.78% and organic carbon of 0.73 %. It also highlighted the value of arsenic with 857 ppm.
- Gravimetric concentration was performed with a Falcon SB40 device, with a water flow of 2.0 gal/min and a frequency of 70 Hz.
- Six passes were made through the concentrator, applied for a constant granulometry of P_{80} of 74 µm where a total recovery of 81.3% of Au and 50.5% of Ag was observed.
- The calculated ROM grade was 4.97 g/t Au with a tailings grade of 0.96 g/t Au.
- The grinding time to reach a P_{80} of 74 µm was 23 minutes.

Table 13-3 shows a summary of the test results of the gravimetric concentration.

Table 13-3: Summary of the Gravimetric Test conducted by PLENGE during 2017

Product	Weight %	Assay		Distribution %	
		Ag g/t	Au g/t	Ag	Au
Gravimetric concentrate 1	0.50	20.0	571.1	17.1	57.6
Gravimetric concentrate 2	0.45	14.8	121.1	11.3	11.0
Gravimetric concentrate 3	0.45	10.7	57.5	8.2	5.2
Gravimetric concentrate 4	0.48	7.7	34.5	6.3	3.3
Gravimetric concentrate 5	0.49	4.7	23.1	4.0	2.3
Gravimetric concentrate 6	0.51	4.2	18.3	3.6	1.9
Rougher concentrate	2.89	10.3	139.9	50.5	81.3
Gravimetric tailing	97.1	0.3	0.96	49.5	18.7
Head (calculated)	102.9	0.6	4.97	100	100
Head (assayed)		0.6	4.96		

Source: Investigación Metalúrgica No. 18251 - 18251 KURI KULLU - ANEXO-PLENGE

13.2.2 SGS Peru Test Work

In August 2017, the SGS laboratory in Peru carried out a series of tests on another sample from Ollachea, which was analyzed to contain 5.36 g/t Au and 0.65 g/t Ag ("SGS 2017").

SGS 2017 reported that mineralized material was ground to two P_{80}'s 106 and 74 microns. Each of the ground samples were passed through a laboratory scale Falcon concentrator six times and the concentrate from each pass was weighed and analyzed.

The overall results of the two tests are shown in Table 13-4:

Table 13-4: SGS Peru - Test Results

Test #	Grind P80	Concentrate Grade g/t Au	Gold Recovery	Concentrate Grade g/t Ag	Silver Recovery	% of Feed to Concentrate
1	106	74.1	75.00%	10.8	68.10%	4.90%
2	74	86.6	81.80%	10.5	71.90%	4.70%

SGS 2017 reported that a finer grind resulted in a significantly higher recovery to concentrate, with 81.8% of the gold reporting to the accumulated concentrate, and with the concentrate mass being 4.7% of the initial feed. Silver analysis was also carried out and although the recoveries to concentrate were relatively high, the grade of silver in the gravity concentrate was low.

A more detailed analysis of the gold recovery obtained with the finer grind (P_{80} 74 microns) is shown in Table 13-5. The final (sixth) stage of concentration still recovered some gold (1.3%) but with 5 stages, resulting in a concentrate mass of 4.07% of the feed, the gold recovery was still 80.6%.

Table 13-5: Test 2 - Details

Concentrate from Stage	Gold Grade g/t		Gold Recovery %		% Of Feed To Concentrate	
	In Stage	Accumulated	In Stage	Accumulated	In Stage	Accumulated
1	331.4	331.4	61.6	61.6	0.93	0.93
2	64.1	200.2	11.5	73.1	0.9	1.83
3	17.0	142.2	2.9	76.0	0.84	2.67
4	18.1	115.2	2.7	78.7	0.74	3.41
5	14.6	98.9	1.9	80.6	0.66	4.07
6	9.71	86.6	1.3	81.9	0.65	4.72
Tailings	0.95	0.95	-	18.1	-	95.3

13.2.3 Met-Solve Test Work (Affiliated with Sepro Mineral Systems)

In October 2017, Met-Solve Laboratories, located in British Columbia, Canada, carried out a test work program on a sample of Ollachea ore, which was analyzed to contain 3.29 g/t Au (“Met-Solve 2017”).

Met-Solve 2017 investigated three stages of gravity concentration were carried out on the ore, starting with a coarse grind (P_{80} 1290 microns) then after further grinding of the gravity tailings to a P_{80} of 319 microns and finally on the tailings from the first two concentration stages with a grind of P_{80} 70 microns. The results are shown in Table 13-6:

Table 13-6: Sepro Test Work - Three stage gravity concentration results

Grind Size (P80 in µm)	Product	Weight		Au	
		(g)	(%)	g/t	Distribution %
1290	Stage 1 Concentrate	114.8	0.6	93.1	16.2
319	Stage 2 Concentrate	89.2	0.5	228.7	31.0
	Stage 1 + 2 Concentrate	**204.0**	**1.0**	**152.4**	**47.2**
70	Stage 3 Concentrate	85.4	0.4	190.8	24.7
	Total Concentrate	**289.4**	**1.5**	**163.7**	**72.0**
70	Final Tailings	19,710.6	98.6	0.9	28.0
Calculated Head		**20,000.0**	**100.0**	**3.3**	**100.0**
Assayed Head		**-**	**-**	**2.7**	**-**

Met-Solve 2017 reported that the three stages of gravity concentration yielded a concentrate grading 163.7 g/t Au with a “mass pull” (mass of concentrate as a % of feed mass) of 1.45% and a gold recovery of 72%. The large increase in recovery obtained at the finer grind suggested that a further increase in mass pull would yield a higher recovery and the tailings were treated with a high mass pull gravity concentration which recovered a further 18.7% of the gold into a concentrate with a mass of 14.7% of the feed. This concentrate was re-ground to a P_{80} of 28 microns and passed through a further stage of gravity concentration, which resulted in a concentrate with a mass of 0.81% of the initial feed with a grade of 31.86 g/t Au. When added to the first concentrates, this gave an overall gold recovery of 79.7% with a mass of concentrate of 2.25% of the feed mass. The detailed results are shown in Table 13-7, together with the flow sheet and mass balance (Figure 13-2).

Table 13-7: Sepro Test Work – Results of P80 regrind with additional stage of gravity concentration

Products	Weight		Assay (g/t)	Distribution (%)
	(g)	(%)	Au	Au
D-GRG Concentrates	289.4	1.45	163.70	71.9
Scavenger Cleaner Concentrate	161.3	0.81	31.86	7.8
Combined Concentrate Products	**450.7**	**2.25**	**116.51**	**79.7**
Scavenger Cleaner Tails	2779.4	13.9	2.59	10.9
Final Gravity Tails	16,770	83.8	0.37	9.4
Calculated Head	**20,000**	**100.0**	**3.30**	**100.0**
Direct Assayed Head			**2.67**	

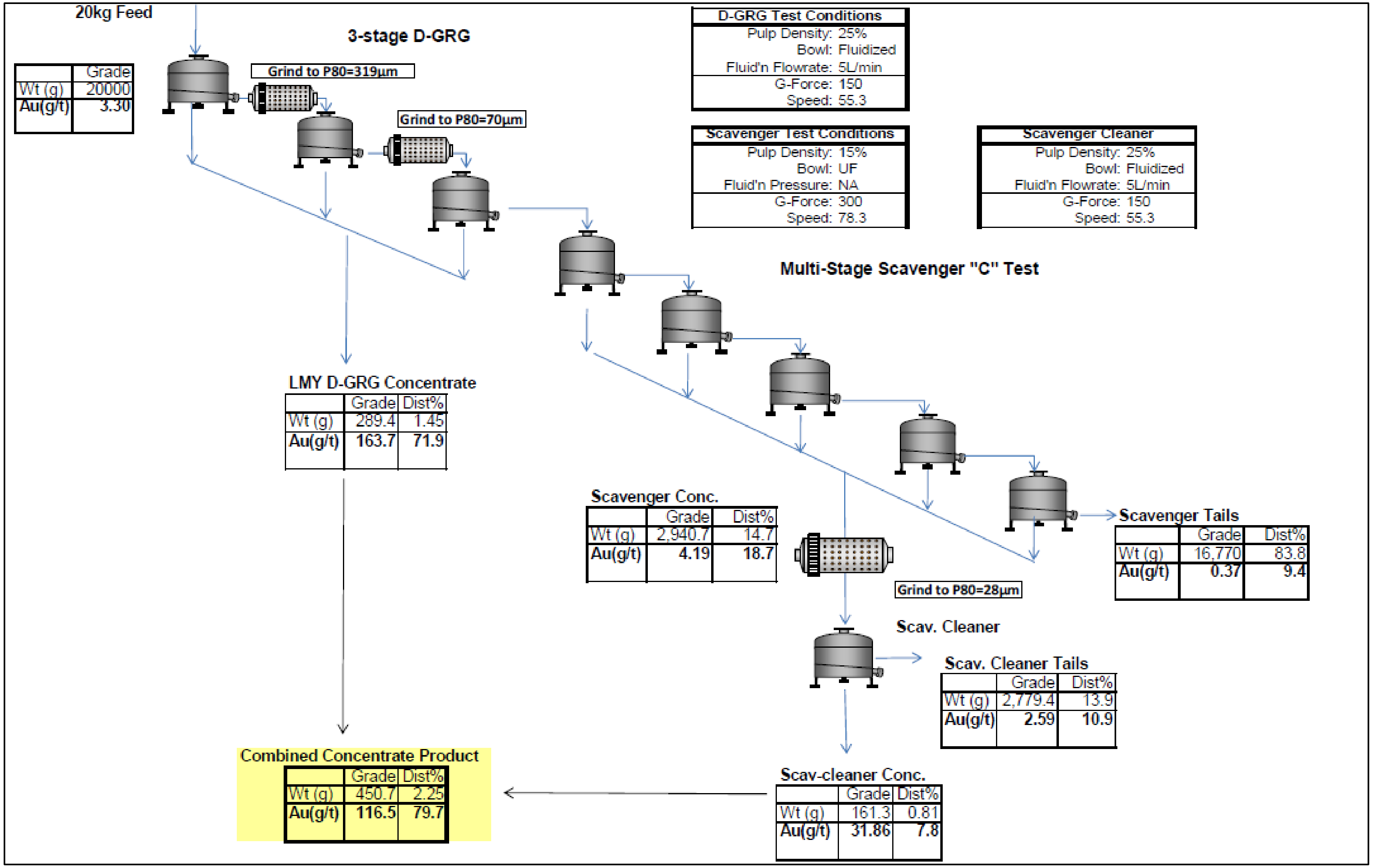

Figure 13-2: Sepro Test Work - flow sheet and mass balance

If all the concentrates are treated by leaching, this would result in 90.6% of the gold reporting to the leach circuit.

13.2.4 Met-Solve Confirmatory Work

In April 2021, it was decided to confirm the results obtained by Met-Solve using a higher-grade sample representative of the Minapampa higher grade zone (“Met-Solve 2021”). The grade of the sample assayed 4.35 g/t Au, which is considered representative of the first three years of the planned production presented in the Technical Report (PEA). Met-Solve 2021 results are shown in flow sheet form in Figure 13-3.

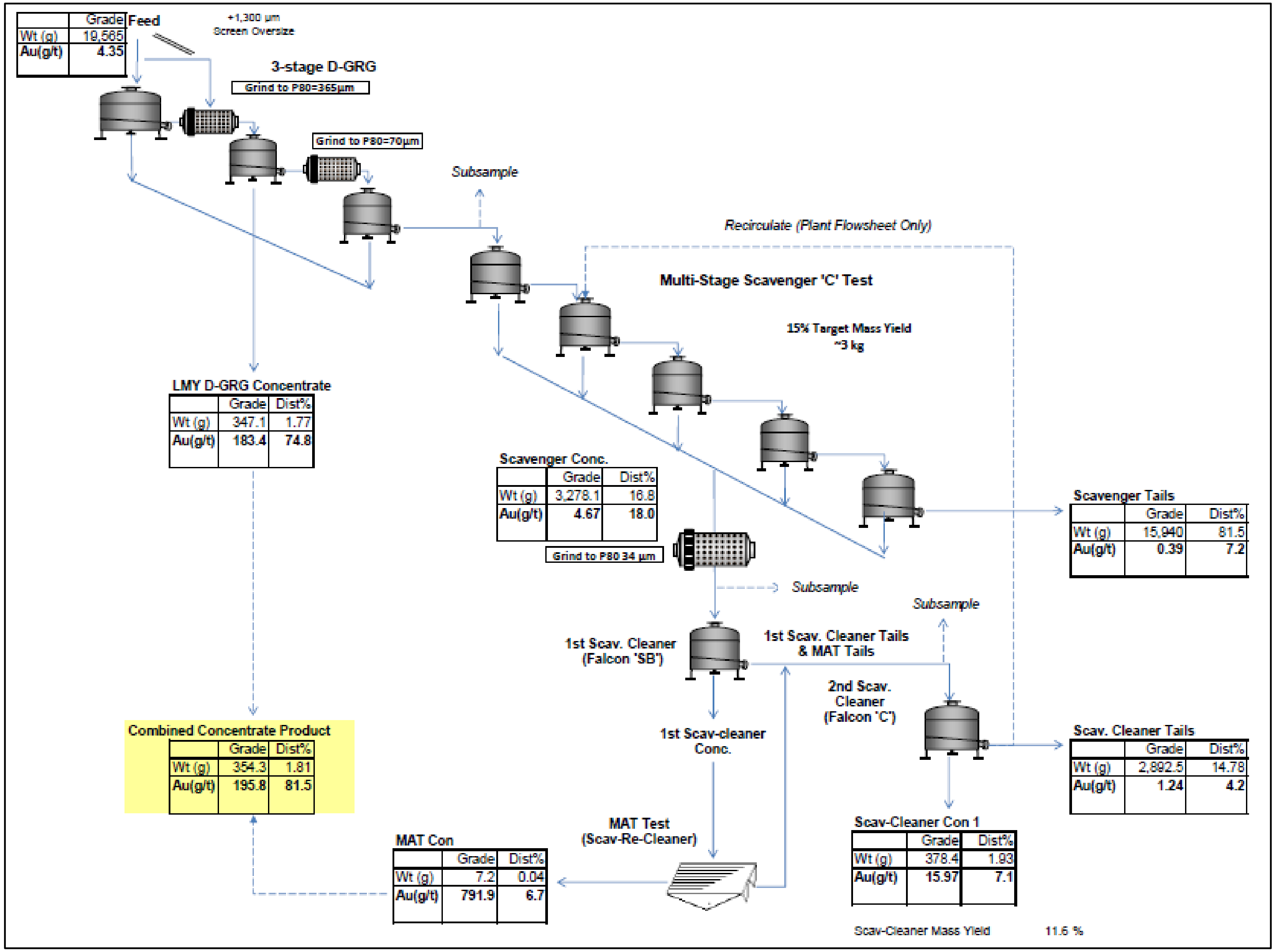

Figure 13-3: Confirmatory Work - results and flow sheet

Met-Solve 2021 reported very similar results to those reported by Met-Solve 2017. Met-Solve 2021 indicated a slightly higher gold recovery to high-grade concentrate and a final tailings grade of 0.39 g/t Au compared with 0.37 g/t Au in Met-Solve 2017. This results in 92.8% of the gold reporting to 18.5% of the mass as various concentrates. It was proposed that an integrated plant would include the leaching of the combined concentrates, after a smeltable gold concentrate had been produced by further concentration of the high-grade concentrate using a shaking table. The organic carbon content of the high mass pull concentrate was 0.51%.

To obtain reliable results, the high-grade concentrates were assayed to extinction, but the final scavenger tailings were leached with cyanide. Unfortunately, the pH was allowed to drop to below 9 and most of the test was carried out with no free cyanide present. The tailings grade was 0.85 g/t Au, with only a 71.6% gold recovery. Excessive cyanide consumption was reported in Ammtec 2013, and re-grinding was found not to improve leach extraction. However, extensive test work reported in Ammtec 2013 has established that the leach residue grade was a function of organic carbon content (see Figure 13-1) and using the regression equation generated, the expect leach residue would assay only 0.17 g/t Au.

13.2.5 Summary of Gravity Concentration Test Work

All gravity concentration test work (Plenge 2017, SGS 2017 and Met-Solve 2017) concluded that recovery to concentrate was improved by a finer grind, and a P_{80} of 74 microns appears to be reasonable level that can be readily obtained in a single stage of grinding. The application of a low mass pull gravity concentration in the primary grinding circuit gives a gold recovery to a high-grade concentrate of 70–75%. The treatment of cyclone overflow from grinding circuit using continuous concentrators gives additional gold recovery to a concentrate with a mass pull of 15-20% of the feed mass, and produces tailings containing 7–10% of the gold with 90–93% reporting to the various concentrates. Met-Solve 2017 and 2021 reported that re-grinding of concentrate allows further gravity recovery to be obtained, and direct CIL leaching this concentrate should give high gold recovery as the organic carbon content was low, 0.51%.

A summary of all gravity concentration test work is presented in Table 13-8.

Table 13-8: Summary of gravity concentrate test work

Laboratory	Head Grade g/t Au	Grind P80 microns	Recovery to concentrate	Mass pull % of feed mass	Notes
Plenge	4.96	74	81.3	2.90%	6 low mass pull concentrates combined
SGS	5.36	74	81.90%	4.70%	6 low mass pull concentrates combined
SGS	5.36	106	74.10%	4.90%	
Met-Solve	3.29	70*	72%	1.45%	* Sequential grinding to 70 microns
Met-Solve 2017		28	79.70%	2.25%	Regrind of high mass pull scavenger concentrate
Met-Solve 2017			90.60%	16.20%	Overall recovery to concentrates
Met-Solve 2021	4.35	70*	74.80%	1.77%	* Sequential grinding to 70 microns
Met-Solve 2021		34	92.80%	18.50%	Overall recovery to concentrates

13.3 Overall Gold Recovery

Using the results of the two gravity concentration tests reported by Met-Solve 2017 and 2021, with head grades of 3.29 and 4.35 g/t Au, respectively, with CIL leaching of all the tailings from the re-grind circuit, predicted overall recoveries of gold are which presented in Table 13-9.

Table 13-9: Summary of overall gold recovery

Head Grade g/t Au	3.29	4.35
Gold Recovery	86.2%	90.3%

The assumptions used are:

- Recovery of gold from high grade concentrates using a shaking table is 50%
- Tailings grade after recovery of a high mass pull concentrate (15%) is 0.4 g/t Au
- Tailings grade after CIL leaching (Ammtec 2013) is 0.3 g/t Au
- Overall process losses in smelting, solution losses in CIL is 1.0%.

Note:

The tailings grade obtained after recovery of a high mass pull concentrate was 0.37 and 0.39 g/t Au, 0.4 g/t Au has been used in calculating the overall gold recovery expected.

With an organic carbon content of 0.51%, the regression equation developed by Ammtec (Ammtec 2013) gives a leach tailings grade of 0.17 g/t Au, the value of 0.3 g/t Au has been used in calculating the overall recovery expected.

14 MINERAL RESOURCE ESTIMATES

14.1 Summary

Dr. Andrew Fowler (QP), Principal Geologist and full-time employee of Mining Plus, is responsible for the Mineral Resource Estimate ("MRE") reported in Section 14 of the Technical Report for the Ollachea Property. The MRE has an effective date of June 30, 2021.

The MRE relates to the Minapampa Zone, and Minapampa Far East Zone ("MFE") of the Property. These zones are within the Ollachea 3 mining concession and are entirely covered by the community agreement reported in Section 4. It is important to note that mineralization extends beyond the Ollachea 3 mining concession into an area held by a third party. This portion of mineralization has been excluded from the MRE Statement.

The MRE has been based on a subset of the drilling data (the drill hole database) reported in Section 10 of the Technical Report. Drill holes not in the Minapampa or Minapampa Far East zones of the Property, and drill holes without downhole survey data have been excluded from the MRE. The subset of drilling data includes 192 diamond drill holes (166 in Minapampa, and 26 DDH in Minapampa Far East) totalling 70,151.75 m of drilled core.

Verification of drill data is summarised in Section 12 of the Technical Report. Dr. Fowler is satisfied that drill data was collected in alignment with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Exploration Best Practice Guidelines (CIM, 2018) and Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019), and that it is suitable for use in the MRE.

Minera Kuri Kullu ("MKK") interpreted the geological wireframe model and structural interpretation in 2014 ("2014 Model") following a program to relog 61 km of core from the Minapampa Zone. The 2014 Model recognised significant vertical displacement of mineralization along the Oscco Cachi Shear ("OCS") and a steepening of mineralized lodes north of the shear. Historical MRE's for the Property considered a different geological model and structural interpretation.

Based on the drill hole database and the 2014 Model, a single block model was generated in Datamine software under Dr. Andrew Fowler's (QP) supervision.

A statistical study of the gold grade distribution and behaviour has been undertaken to inform grade interpolation in the block model. Gold grades were estimated using Ordinary Kriging (OK) and bias was reviewed using a Nearest Neighbour estimate (NN). Drill hole intervals have been composited to a length of 1 m, which is the average sample length in the mineralized zone. Grade capping has been applied to composited grade intervals on a case-by-case basis within each mineralized and host rock domain.

Dry bulk density applied to the model is based on measurements from 777 core samples. Mineralized material has been assigned a dry bulk density of 2.83 t/m^3 and host rock has been assigned a dry bulk density of 2.80 t/m^3.

The MRE for Ollachea, with an effective date of June 30, 2021, has been constrained by optimised underground stope shapes and is reported at a cut-off grade of 1.4 g/t Au. The MRE has been categorized in accordance with the CIM Definition Standards (CIM, 2014) and comprises an Indicated and Inferred Mineral Resource as summarised in Table 14-1.

Table 14-1: Mineral Resource Estimate for the Ollachea Project by classification and Zone

Mineral Resource Estimate for the Ollachea Project - June 30, 2021						
Zone	Indicated			Inferred		
	Tonnes (Mt)	Au g/t	Au Ounces (Moz)	Tonnes (Mt)	Au g/t	Au Ounces (Moz)
Minapampa	10.7	3.28	1.13	1.8	3.0	0.2
Minapampa Far East	-	-	-	5.5	2.6	0.5
Total	10.7	3.28	1.13	7.3	2.7	0.6

1. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
2. All figures are rounded to reflect the relative accuracy of the estimates.
3. The Mineral Resource was estimated by Ms. Muñoz and supervised by Dr. A. Fowler, MAusIMM, CP(Geo), Independent Qualified Person under NI 43-101., of Mining Plus Consultants who takes responsibility for it.
4. Composite gold grades were capped where appropriate.
5. Mineral Resources are diluted and are reported within optimized underground stope shapes.
6. The stope shapes were optimized at a gold cut-off value of 1.4 grams per tonne, considering metal prices of US$1700 per ounce of gold, and assuming metal recovery of 87% for gold, and total operating costs of $61.18/t.
7. Tonnages reported are metric tonnes and ounces of contained gold are troy ounces.
8. Mining Plus is not aware of any environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues that could materially affect the potential development of the Mineral Resource Estimate.

14.2 Drill Data

The MRE has been based on a subset of the drill hole database reported in Section 10 of the Technical Report. Drill holes outside the Minapampa or Minapampa Far East zones of the Property, and drill holes without survey data have been excluded from the MRE. The subset of drilling data includes 192 diamond drill holes, which consist of 166 in Minapampa (65,004.85 m) and 26 in Minapampa Far East (5146.90 m) totalling 70,151.75 m of drilled core. Collar details of the drill holes used in the MRE have been summarized in Table 14-2.

Table 14-2: Drill Collar Data Used in the MRE

DDH ID	Easting (m)	Northing (m)	Elevation (m)	Length (m)	Resource Zone
DDH08-01	339317.899	8474215.34	3059.32	304.9	Minapampa
DDH08-02	339074.185	8474179.7	3062.78	324.85	Minapampa
DDH08-03	339008.398	8474161.89	3065.4	303.85	Minapampa
DDH08-04	339209.286	8474181.5	3056.3	267.7	Minapampa
DDH08-05	339371.373	8474236.15	3053.53	373.5	Minapampa
DDH08-07	339584.272	8474304.63	3037.01	416.3	Minapampa

DDH ID	Easting (m)	Northing (m)	Elevation (m)	Length (m)	Resource Zone
DDH08-08	338853.26	8474134.16	3087.6	353.2	Minapampa
DDH08-10	338687.268	8474051.56	3082.68	224.8	Minapampa
DDH08-12	338470.127	8473991.48	3121.92	308.4	Minapampa
DDH08-16	339027.236	8473994.87	3048.55	196.2	Minapampa
DDH08-17	339161.14	8473986.47	3046.47	198.25	Minapampa
DDH08-18	339289.219	8473943.43	3048.99	172.45	Minapampa
DDH08-21	339640.563	8474312.55	3022.22	393.4	Minapampa
DDH08-22	339149.468	8474194.57	3059.12	301.8	Minapampa
DDH08-23	339452.751	8474264.79	3044.88	372.4	Minapampa
DDH09-06	339069.911	8474278.06	3106.31	427.05	Minapampa
DDH09-09	339011.035	8474253.68	3114.9	383.2	Minapampa
DDH09-11	338927.415	8474242.7	3122.51	432.7	Minapampa
DDH09-13	338847.858	8474209.61	3133.2	424.1	Minapampa
DDH09-19	339201.229	8474283.55	3082.15	389.1	Minapampa
DDH09-20	339141.387	8474291.38	3096.48	453.3	Minapampa
DDH09-24	339141.592	8474290.55	3096.48	425.05	Minapampa
DDH09-25	339292.365	8474305.05	3075.65	453.8	Minapampa
DDH09-26	339262.659	8474196.85	3060.27	306.9	Minapampa
DDH09-27	339292.365	8474305.05	3075.65	430.7	Minapampa
DDH09-28	339377.309	8474333.54	3079.06	435.4	Minapampa
DDH09-29	339201.229	8474283.55	3082.15	393.9	Minapampa
DDH09-30	339377.309	8474333.54	3079.06	457.55	Minapampa
DDH09-31	339069.838	8474278.31	3106.31	407.05	Minapampa
DDH09-32	339281.251	8474200.49	3060.14	333.8	Minapampa
DDH09-33	339301.328	8474205.56	3059.98	352.7	Minapampa
DDH09-34	339259.63	8474177.32	3057.51	311.1	Minapampa
DDH09-35	339271.111	8474225.38	3061.96	322.2	Minapampa
DDH09-36	339279.951	8474257.55	3065.65	319.8	Minapampa
DDH09-37	339292.632	8474278.45	3068.82	341.1	Minapampa
DDH09-38	339370.122	8474283.38	3059.68	411.35	Minapampa
DDH09-39	339201.229	8474283.55	3082.15	51.95	Minapampa
DDH09-40	339011.035	8474253.68	3114.92	449.4	Minapampa
DDH09-41	339377.309	8474333.54	3079.06	462.95	Minapampa
DDH09-42	339008.315	8474162.23	3065.35	300.2	Minapampa
DDH09-43	339371.677	8474235.5	3053.53	300	Minapampa
DDH09-44	339074.28	8474178.9	3062.78	275	Minapampa
DDH09-45	339149.513	8474193.84	3059.17	250.55	Minapampa
DDH09-46	339301.328	8474205.56	3060	248.7	Minapampa
DDH09-47	339209.555	8474180.53	3056.3	222.15	Minapampa
DDH09-48	339262.824	8474196.04	3060.24	253.3	Minapampa
DDH09-49	339452.817	8474263.71	3044.88	270.7	Minapampa
DDH09-50	339456.042	8474360.73	3081.44	495.5	Minapampa
DDH09-51	339292.365	8474305.05	3075.65	378.4	Minapampa
DDH09-52	339456.042	8474360.73	3081.45	516.1	Minapampa
DDH09-53	339292.365	8474305.05	3075.65	368.1	Minapampa
DDH09-54	339456.042	8474360.73	3081.33	452.4	Minapampa
DDH09-55	338927.368	8474242.12	3122.51	425.85	Minapampa
DDH09-56	339456.042	8474360.73	3081.31	433.35	Minapampa
DDH09-57	338927.368	8474242.12	3122.5	382.5	Minapampa
DDH09-58	338687.204	8474051.54	3082.68	253.15	Minapampa
DDH09-59	338830.502	8473974.98	3071.82	282.7	Minapampa
DDH09-60	338847.804	8474209.96	3133.23	451.8	Minapampa
DDH09-61	339108.385	8474288.77	3104.16	407.3	Minapampa
DDH09-62	339517.218	8474405.9	3093.33	431.25	Minapampa
DDH09-64	339584.697	8474306.29	3037.01	394.1	Minapampa
DDH09-65	339517.135	8474406.24	3093.32	524.5	Minapampa
DDH09-67	339640.58	8474313.19	3022.33	373.1	Minapampa
DDH09-68	338929.105	8474398.22	3208.34	160.1	Minapampa
DDH09-70	339600.498	8474454.35	3093.52	558	Minapampa
DDH09-71	339642.766	8474463.86	3092.934	600.3	Minapampa
DDH09-73	339642.766	8474463.86	3092.934	533.3	Minapampa
DDH09-76	339642.766	8474463.86	3092.934	562.35	Minapampa
DDH09-78	338679.884	8474207.78	3193	596.1	Minapampa
DDH09-79	338578.884	8474191.79	3199.431	470.5	Minapampa

DDH ID	Easting (m)	Northing (m)	Elevation (m)	Length (m)	Resource Zone
DDH10-100	339204.226	8474234.16	3067.356	360.7	Minapampa
DDH10-101	339107.545	8474192.86	3062.922	239.3	Minapampa
DDH10-102	339204.146	8474235.01	3067.272	300.7	Minapampa
DDH10-103	339107.59	8474193.24	3062.91	273.3	Minapampa
DDH10-104	338978.395	8474254.44	3120.775	346.05	Minapampa
DDH10-105	338904.635	8474235.34	3128.514	437.1	Minapampa
DDH10-106	339069.885	8474278.21	3106.813	369.75	Minapampa
DDH10-107	339035.215	8474265.62	3114.85	302.8	Minapampa
DDH10-108	339334.341	8474302.43	3068.412	441.9	Minapampa
DDH10-109	339141.405	8474290.99	3096.984	400.1	Minapampa
DDH10-110	339164.675	8474250.5	3076.595	400.65	Minapampa
DDH10-111	339141.405	8474290.99	3096.984	366	Minapampa
DDH10-112	339165.455	8474249.81	3076.431	340	Minapampa
DDH10-113	338904.635	8474235.34	3128.514	409.9	Minapampa
DDH10-114	339204.146	8474235.01	3067.272	383.8	Minapampa
DDH10-115	338948.345	8474297.01	3162.117	490.75	Minapampa
DDH10-116	339172.145	8474195.99	3059.343	280	Minapampa
DDH10-117	339224.446	8474190	3059.107	313.8	Minapampa
DDH10-118	339292.826	8474280.79	3069.472	419.9	Minapampa
DDH10-119	339238.081	8474239.01	3065.883	320.2	Minapampa
DDH10-120	339642.769	8474463.86	3092.934	66.6	Minapampa
DDH10-120R	339658.416	8474394.97	3045.144	475.15	Minapampa
DDH10-121	339600.196	8474453.44	3094.019	470.35	Minapampa
DDH10-122	339516.936	8474406.29	3093.864	422.5	Minapampa
DDH10-123	339600.616	8474453.63	3094.248	500	Minapampa
DDH10-124	339516.936	8474406.29	3093.864	437.6	Minapampa
DDH10-125	339642.766	8474463.86	3092.934	539.45	Minapampa
DDH10-133	339481.866	8474417.96	3111.662	647.1	Minapampa
DDH10-136	339502.246	8474381.87	3091.402	401.1	Minapampa
DDH10-138	339517.206	8474406.11	3093.822	456.7	Minapampa
DDH10-80	338852.955	8474134.47	3088.097	336.1	Minapampa
DDH10-81	338488.104	8474191.18	3227.487	562.6	Minapampa
DDH10-82	338927.025	8474242.71	3123.103	431.2	Minapampa
DDH10-83	339035.215	8474265.62	3114.85	360.35	Minapampa
DDH10-84	339399.936	8474345.48	3083.848	446.3	Minapampa
DDH10-85	339035.215	8474265.62	3114.85	400.1	Minapampa
DDH10-86	339335.716	8474301.16	3068.441	376.9	Minapampa
DDH10-87	339165.455	8474249.81	3076.431	322.9	Minapampa
DDH10-88	339335.826	8474300.19	3068.322	410.5	Minapampa
DDH10-89	339165.505	8474249.27	3076.546	360	Minapampa
DDH10-90	339414.206	8474324.25	3068.499	362.3	Minapampa
DDH10-91	338980.695	8474166.91	3073.39	310.7	Minapampa
DDH10-92	339493.596	8474288.06	3044.207	360	Minapampa
DDH10-93	339517.206	8474406.11	3093.822	480.6	Minapampa
DDH10-94	338941.015	8474166.1	3080.322	330.5	Minapampa
DDH10-95	338941.105	8474165.28	3080.34	304.8	Minapampa
DDH10-96	339415.536	8474387.41	3114.314	527.5	Minapampa
DDH10-97	338902.675	8474153.33	3083.229	312.95	Minapampa
DDH10-98	339481.926	8474417.06	3111.634	516	Minapampa
DDH10-99	338902.665	8474153.63	3083.274	303.9	Minapampa
DDH11-140	339549.656	8474424.79	3089.728	511.5	Minapampa
DDH11-142	339549.066	8474424.35	3089.721	523.35	Minapampa
DDH11-143	339556.176	8474292.12	3039.287	434.7	Minapampa
DDH11-144	339605.906	8474359.58	3048.235	364.7	Minapampa
DDH11-146	339605.906	8474359.58	3048.235	369.15	Minapampa
DDH11-148	339655.116	8474399.03	3045.165	459.7	Minapampa
DDH11-150	339677.066	8474480.7	3099.052	515.8	Minapampa
DDH11-151	339644.206	8474465.05	3093.132	513.6	Minapampa
DDH11-152	339685.036	8474366.36	3019.817	376.75	Minapampa
DDH11-153	339685.036	8474366.36	3019.817	323.1	Minapampa
DDH11-175	339605.906	8474359.58	3048.235	354.35	Minapampa
DDH11-176	339552.18	8474293.38	3038.956	298	Minapampa
DDH11-177	339605.906	8474359.58	3048.235	379.9	Minapampa
DDH11-178	339685.036	8474366.36	3019.82	305.3	Minapampa

DDH ID	Easting (m)	Northing (m)	Elevation (m)	Length (m)	Resource Zone
DDH11-179	339603.52	8474362.65	3047.557	40.1	Minapampa
DDH11-179R	339603.52	8474362.65	3047.557	46.7	Minapampa
DDH11-179T	339603.52	8474362.65	3047.557	440.5	Minapampa
DDH11-180	339685.036	8474366.36	3019.817	302.2	Minapampa
DDH11-181	339685.036	8474366.36	3019.817	365.15	Minapampa
DDH11-182	339599.73	8474455.65	3093.297	266.2	Minapampa
DDH11-183	339682.55	8474367.95	3019.056	416.8	Minapampa
DDH11-184	339549.656	8474424.79	3089.728	525.65	Minapampa
DDH11-185	339645.57	8474438.05	3071.704	530.6	Minapampa
DDH11-186	339549.656	8474424.79	3089.728	533.7	Minapampa
DDH11-187	339590.856	8474426.29	3077.42	534.65	Minapampa
DDH11-188	339228.88	8474192.74	3057.885	375.6	Minapampa
DDH11-189	339593.04	8474432.02	3078.622	544.5	Minapampa
DDH11-190	339170.21	8474274.59	3082.62	350.2	Minapampa
DDH11-192	339146.33	8474198.28	3059.24	340.5	Minapampa
DDH11-193	339165.455	8474249.81	3076.431	283.9	Minapampa
DDH12-191	339136.91	8474280.13	3088.752	484.2	Minapampa
DDH12-194	339162.16	8474251.39	3075.79	328	Minapampa
DDH12-195	339100.42	8474290.07	3104.345	402.4	Minapampa
DDH12-196	339129.44	8474201.46	3061.343	341.5	Minapampa
DDH12-197	339069.29	8474279.51	3106.148	445.05	Minapampa
DDH12-198	339411.75	8474265.01	3050.254	407.5	Minapampa
DDH12-199	339035.215	8474265.62	3114.85	424.9	Minapampa
DDH12-200	339334.33	8474302.19	3067.546	382.2	Minapampa
DDH12-201	339032.53	8474268.41	3113.91	437	Minapampa
DDH12-202	339376.58	8474335.07	3078.643	437.8	Minapampa
DDH12-203	339413.93	8474389.83	3113.789	521.7	Minapampa
DDH12-204	338966.11	8474256.15	3119.91	438.55	Minapampa
DDH12-205	339548.79	8474428.22	3089.5	497.5	Minapampa
DDH12-206	338879.21	8474220.05	3122.866	410.6	Minapampa
DDH12-207	338877.963	8474218.74	3124.8	440.5	Minapampa
DDH12-208	339279.334	8474302.76	3075.92	487.15	Minapampa
DDH13-T01	339850.522	8474533.21	2776.701	407.05	Minapampa Far East
DDH13-T02	339850.189	8474533.55	2776.593	184	Minapampa Far East
DDH13-T03	339900.3	8474570.22	2775.82	290	Minapampa Far East
DDH13-T04	339850.276	8474539.43	2777.045	321.45	Minapampa Far East
DDH16-T05	339729.92	8474386.13	2780.309	170	Minapampa Far East
DDH16-T06	339817.75	8474478.42	2778.207	220	Minapampa Far East
DDH16-T07	339729.276	8474387.86	2779.729	190.75	Minapampa Far East
DDH16-T08	339817.75	8474478.42	2779.217	230	Minapampa Far East
DDH16-T09	339729.92	8474386.13	2781.159	170	Minapampa Far East
DDH16-T10	339817.408	8474479.36	2777.8	250.05	Minapampa Far East
DDH16-T11	339729.058	8474388.5	2779.729	178.1	Minapampa Far East
DDH16-T12	339775.618	8474435.98	2778.695	180	Minapampa Far East
DDH16-T13	339913.573	8474583.24	2775.658	264.8	Minapampa Far East
DDH16-T14	339775.58	8474436.08	2778.695	245.6	Minapampa Far East
DDH16-T15	339776.073	8474434.73	2780.159	200	Minapampa Far East
DDH16-T16	339913.792	8474582.46	2775.658	312.7	Minapampa Far East
DDH16-T17	339776.073	8474434.73	2779.029	170	Minapampa Far East
DDH16-T18	339868.025	8474532.96	2776.684	259.3	Minapampa Far East
DDH16-T19	339914.15	8474581.47	2775.658	264.4	Minapampa Far East
DDH16-T20	339868.305	8474532.19	2778.838	230.4	Minapampa Far East
DDH16-T21	339913.9	8474582.33	2775.658	252.3	Minapampa Far East
DDH16-T22A	339874.63	8474539.6	2776.137	210.3	Minapampa Far East
DDH16-T23	339950.269	8474618.61	2775.506	270.2	Minapampa Far East
DDH16-T24	340052.635	8474727.21	2772.2	290.1	Minapampa Far East
DDH16-T26	339949.721	8474620.11	2774.904	290.1	Minapampa Far East
DDH16-T27	340006.627	8474678.79	2773.522	297.8	Minapampa Far East

14.3 Geological / Structural Model

The MRE reported in the Technical Report (June 2021 MRE) is based on the geological and structural model developed by MKK following a program to relog 61 km of core from the Minapampa Zone in 2014 (“2014 Model”), and this model was updated in 2016 to incorporate drilling at Minapampa Far East. The 2014 Model recognised significant vertical displacement of mineralization along the Oscco Cachi Shear (“OCS”).

Dr. Fowler (QP) notes that historical MRE’s did not reflect vertical displacement of mineralization along the OCS or the steepening of mineralized lodes north of the shear.

The 2014 model reflected an easterly-striking mineralized corridor (1km x 250m) consisting of stacked mineralized lodes. Individual lodes within the corridor dip northwards between 44 and 65 degrees, and 1 to 25 m thick. Sixty-four (64) discreet lodes have been modelled within the corridor.

Mineralized lodes (LODES) occur as discrete zones by depth and have locally been segregated into horizons (MZONES).

Sixty-four (64) individual mineralized lodes (58 from Minapampa and 6 from Minapampa Far East) were interpreted in 2014 by MKK, based on the drill hole sample data using a notional grade threshold of approximately 1.0 g/t Au to define coherent zones of mineralization. The three-dimensional interpretation wireframe was constructed using conventional sectional strings and manual wireframe linking. Wireframe triangle vertices were snapped to drill hole sample boundaries on sections orthogonal to the strike direction. Figure 14-1 is a plan view of the mineralized lodes, there is a clear separation of lodes to the north and south of the OCS. Figure 14-2 shows a perspective view of the lodes, looking west.

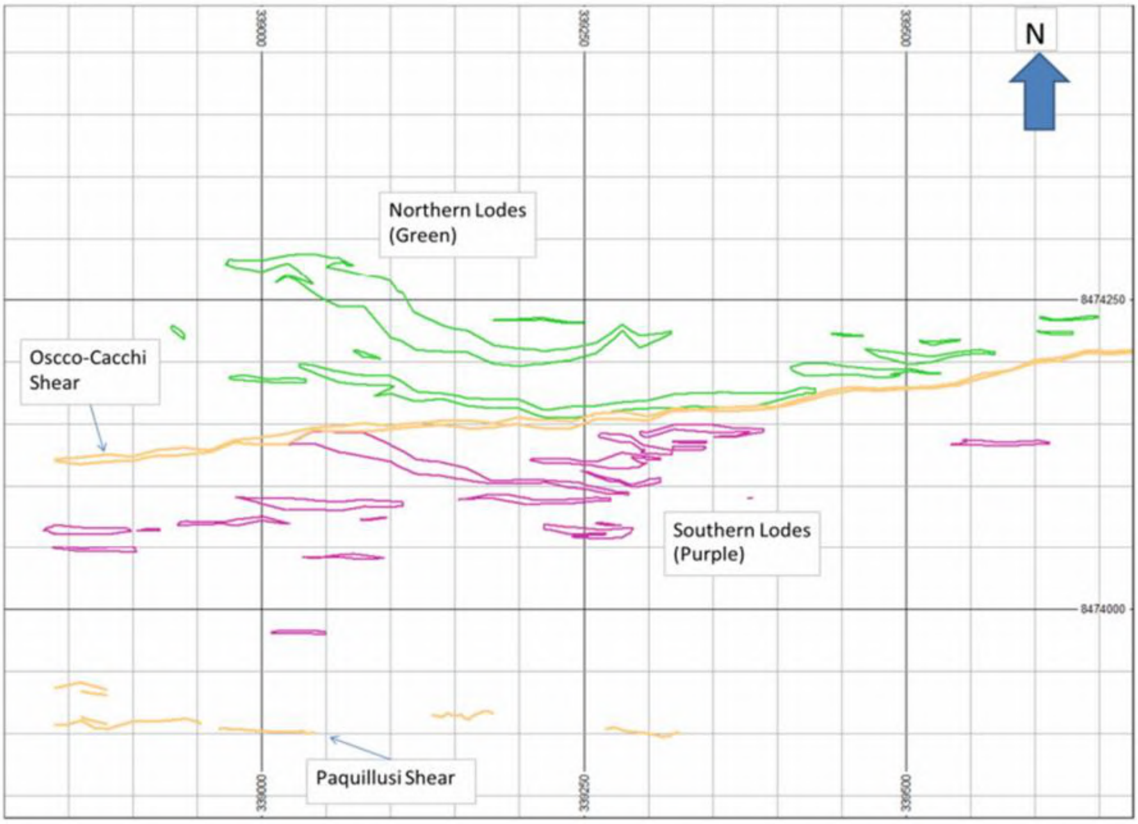

Figure 14-1: 2014 Structural and Geological Model - Plan view (2875 level) of mineralized lodes and the OCS Fault

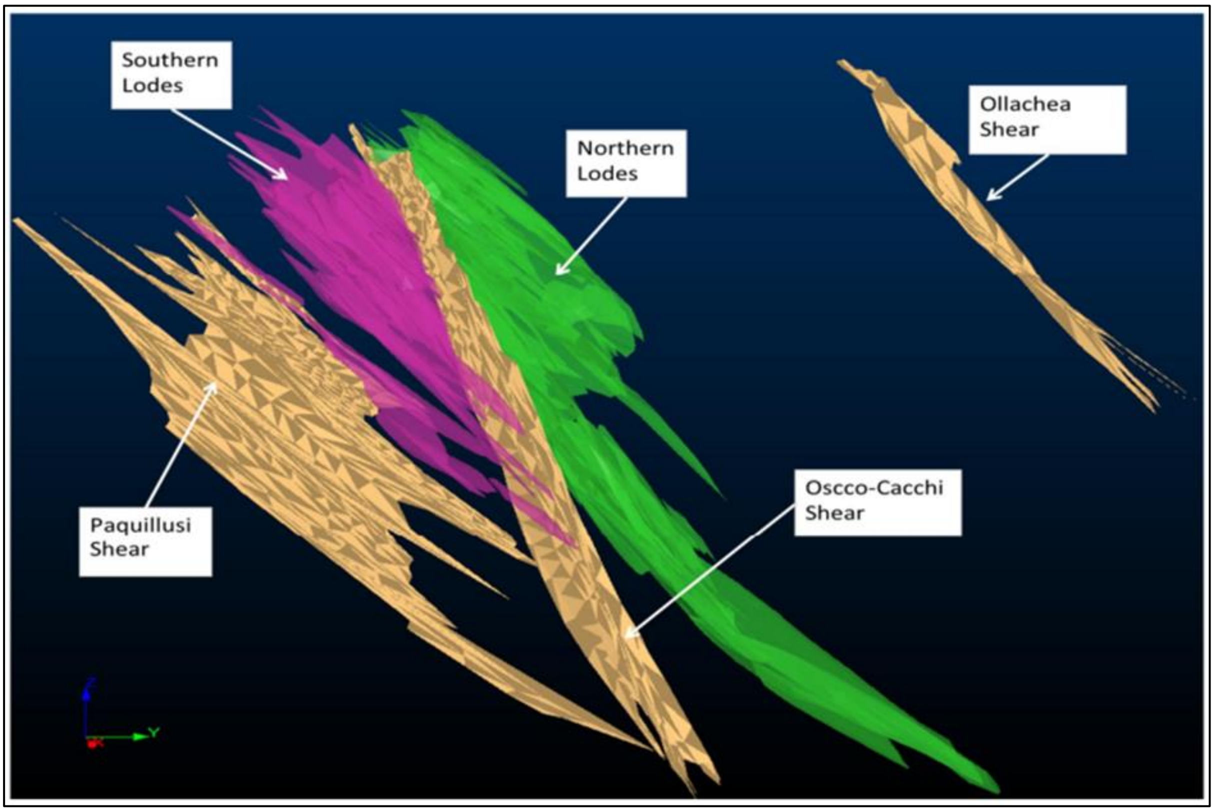

Figure 14-2: Perspective view looking west of the 2014 Model - Mineralization is clearly interrupted either side of the OCS.

The 2014 Model included sixty-four (64) mineralized lodes (LODE) that have been grouped into nine mineralized horizons (MZONE) and subdivided north and south of the OCS (Table 14-3 and Table 14-4).

Table 14-3: 2014 Model (Minapampa) - Coded mineralized lodes (LODE) and horizons (MZONE)

LODE	North or South of OCS	MZONE
101	North	1
112		
113		
116		
117		
118		
120		
121		
123		
124		
125		
126		
151	South	
152		
153		
154		
155		
156		
201	North	2
202		
203		
206		
207		
208		
251	South	
252		
253		
254		
255		
256		
257		
303	North	3
305		
308		
345		
351	South	
354		
357		
358		
361		
408	North	4
407		
410		
411		
412		
413		
451	South	
453		
454		
456		
457		
461		
462		
463		

LODE	North or South of OCS	MZONE
502	North	5
555	South	
556		
601	North	6

Table 14-4: 2014 Model (Minapampa Far East) - Modelled mineralized lodes and horizons

LODE	North or South of OCS	MZONE
101	North	7
102		
301		
201		8
202		
203		9

In 2016, Dr. Fowler considered an additional high-grade domain should be interpreted. The high-grade domain has been modelled by Mining Plus using conventional sectional strings and manual wireframe linking methods at a nominal grade threshold of >=4 g/t Au (Figure 14-3). Wireframe triangle vertices were snapped to drill hole sample boundaries on sections orthogonal to the strike direction. The >=4 g/t Au High-Grade domain has been modelled in the central Minapampa Zone. Dr. Fowler considers that this zone has sufficient continuity between drill hole sections at >=4 g/t Au and is the only portion of the deposit that could potentially support an elevated mining cut-off grade for a reasonable period of time (3 - 4 years) at the proposed initial mining rate of 1,500 tpd.

Additionally, Dr Fowler notes that the newer MFE Zone mineralization interpretation overlaps part of the MRE that supported the 2012 Feasibility Study. As the MFE Zone has been interpreted from closer-spaced drilling and improved geological understanding, it has replaced the Minapampa Zone interpretation where the two overlapped. That overlapping part of the Minapampa Zone, comprising approximately 20,000 ounces of gold, has been effectively subtracted from the Minapampa Zone and now forms part of the MFE Zone, with no change in the Mineral Resource classification.

A plan view of mineralized domains (Minapampa, Minapampa Far East, and High-grade) can be seen in Figure 14-3.

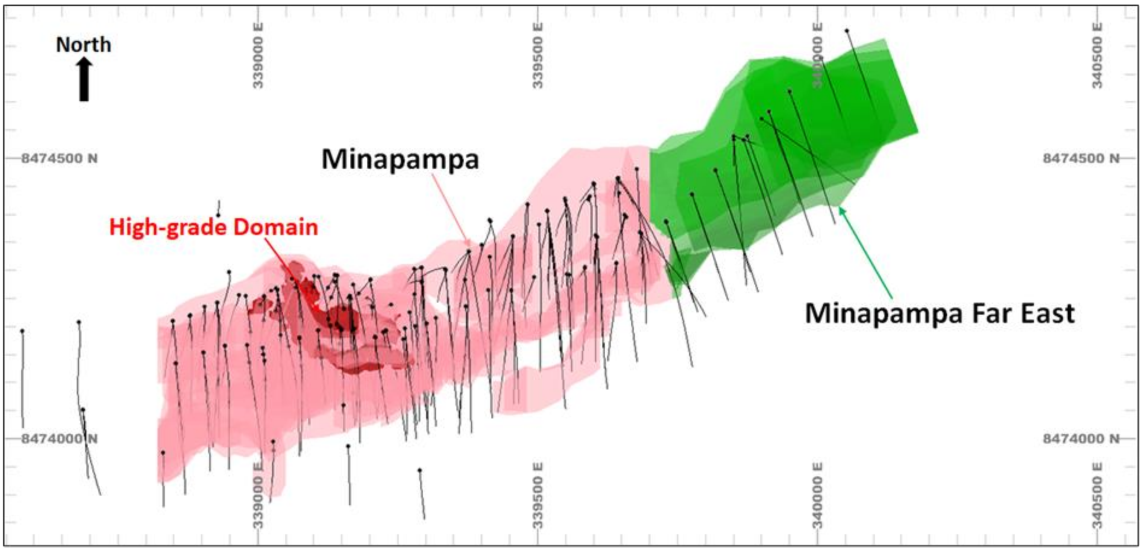

Figure 14-3: 2014 model - Plan view of mineralized domains (Minapampa, Minapampa Far East, and High-grade)

To account for mining dilution, a low-grade envelope in the host rock to the mineralization (MZONE 99) has been modelled around the lodes Figure 14-4.

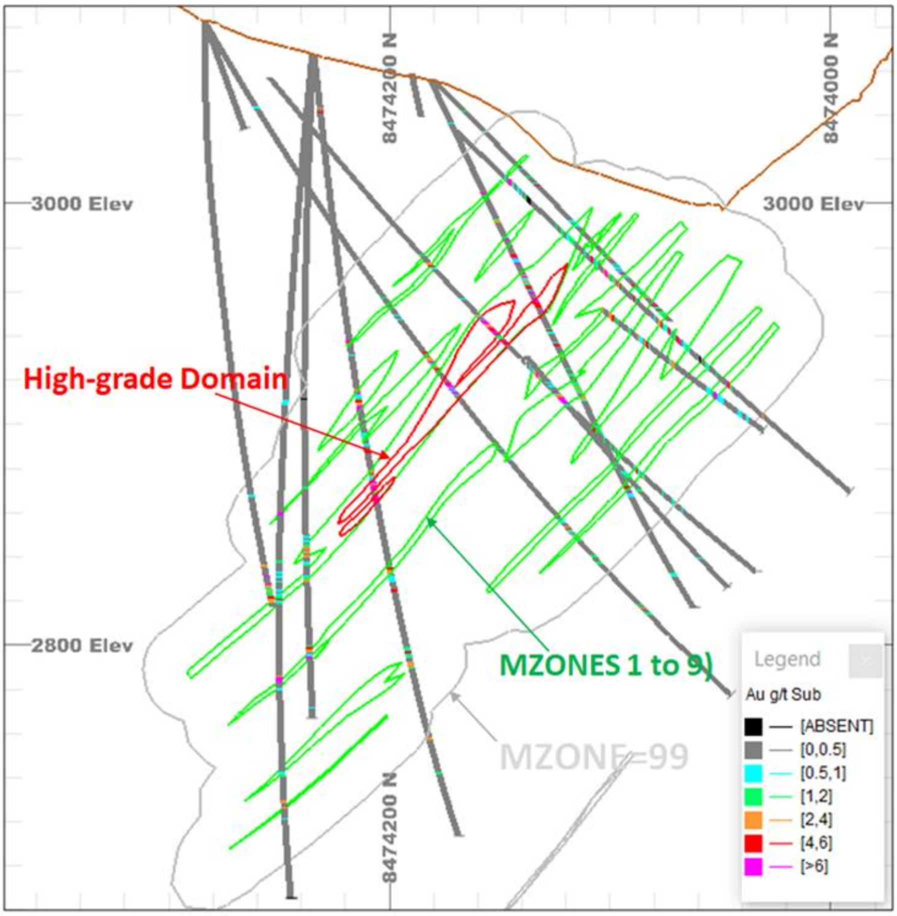

Figure 14-4: 2014 model - Cross-section 339,200 mE showing low-grade host rock domain (MZONE 99) modelled around mineralized lodes

Estimation domains are based on the MZONE grouping defined during wireframe construction and have also been divided according to grade and location relative to the OCS. Domain codes are summarized in (Table 14-5).

Table 14-5: Estimation domains and Codes

MZONE		CODE
99 (Host rock)		10990
1 (North)	Low Grade	10011
	High Grade	20011
2 (North)	Low Grade	10021
3 (North)	Low Grade	10031
4 (North)	Low Grade	10041
5 (North)	Low Grade	10051
6 (North)	Low Grade	10061
1 (South)	Low Grade	10012
2 (South)	Low Grade	10022
3 (South)	Low Grade	10032
4 (South)	Low Grade	10042
5 (South)	Low Grade	10052
7 (North)	Low Grade	10070
8 (North)	Low Grade	10080
9 (North)	Low Grade	10090

14.4 Oxidation State

Based on a core logging, oxidation at the Property is poorly developed and therefore, the MRE has treated all material as fresh.

14.5 Treatment of Missing / Absent Samples

Unsampled intervals of drill core have been assigned grades according to the following rules:

- Unsampled intervals within MZONE 1 to 9 have been assumed to reflect to poor core recovery and have been assigned a grade of “Null”
- Unsampled intervals within the low-grade envelope (MZONE 99) modelled around mineralized lodes have been assigned a grade of 0.005 g/t Au
- Unsampled intervals from surface (0 m) have been assumed to reflect colluvial cover and have been assigned a grade of 0.005 g/t Au
- Unsampled intervals >5 m have been assigned a grade of 0.005 g/t Au.

14.6 Compositing

Considering common raw sampling intervals and to achieve uniform sample support, the drill hole database has been coded with the mineralized horizons (MZONE 1 to 9) and host rock (MZONE 99) to 1 m down hole composite intervals.

A residual retention routine has been used where residuals are added back to the next adjacent interval. For the 1 m composites the majority of composite intervals are 1 m, with a small number of composite intervals ranging from 0.5 to 1.5 m. Mean composite intervals are 1 m.

Summary statistics for raw (un-composited) and composited sample intervals in mineralized material and host rock are presented in Figure 14-5 and Figure 14-6.

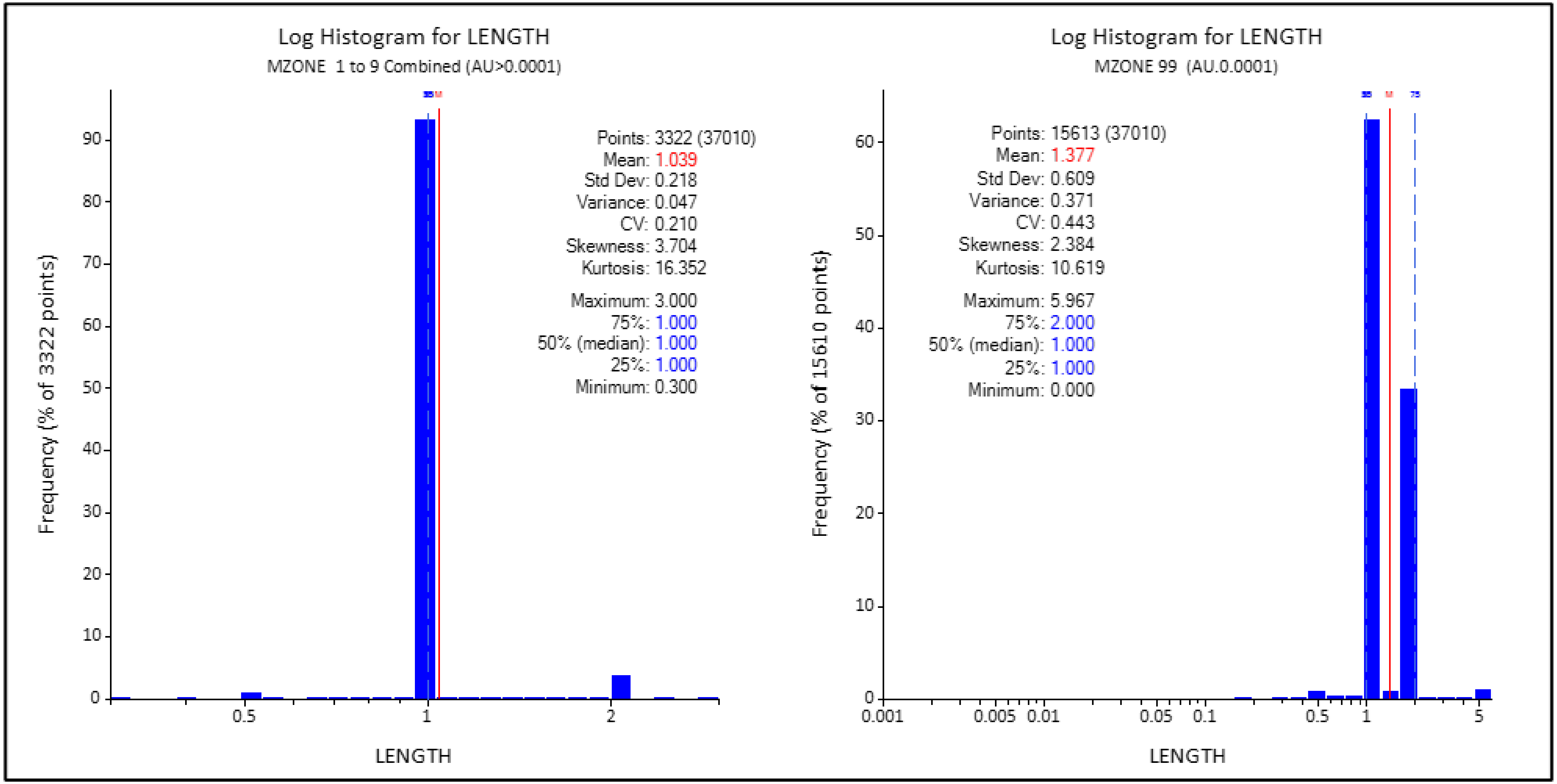

Figure 14-5: Un-composited Sample Data - Samples length within Mineralized Lodes (left) and Host rock (right).

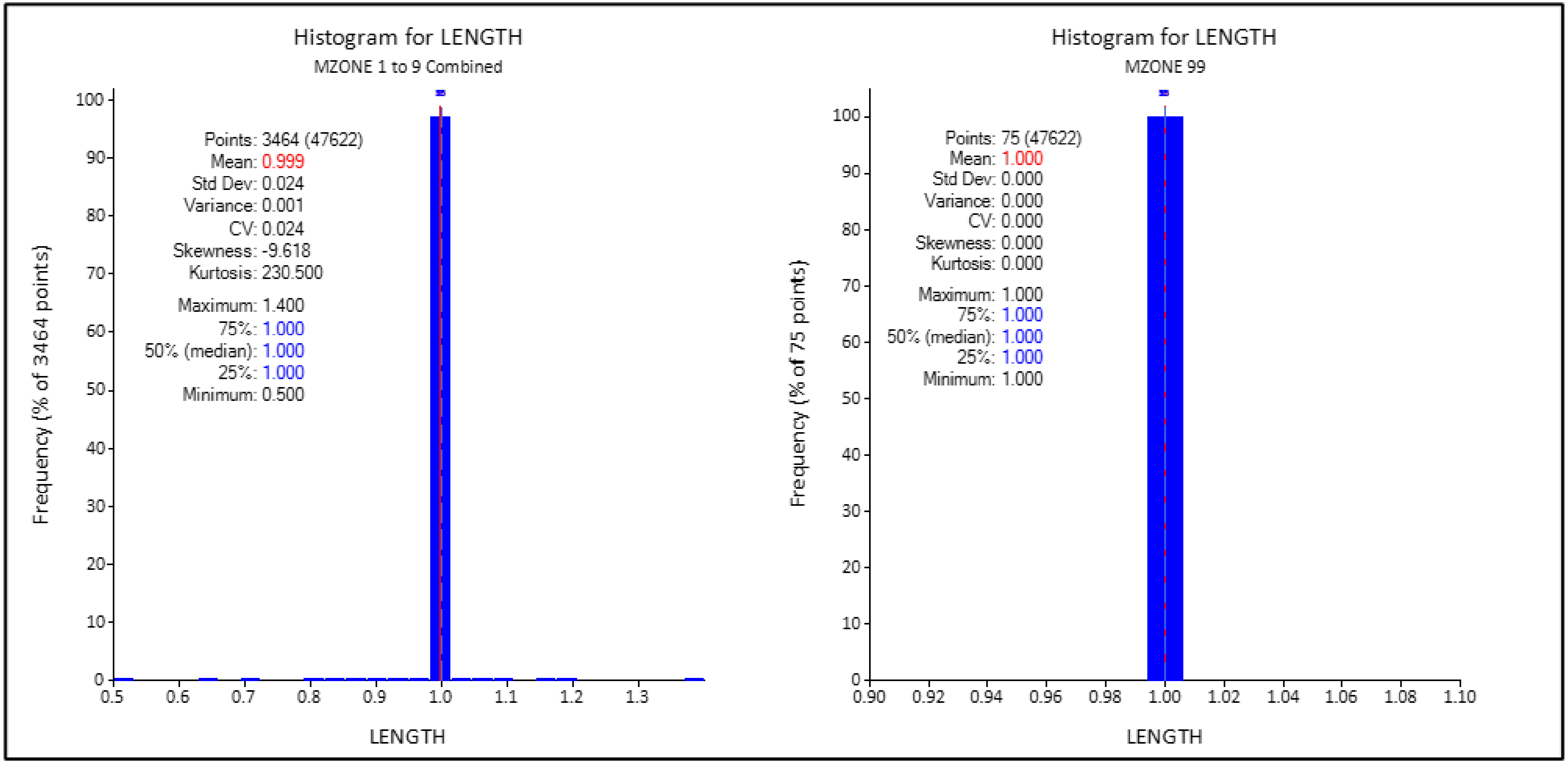

Figure 14-6: 1 m Composite Data - Sample intervals within Mineralized Lodes (left) and Host rock (right).

14.7 Statistical Analysis

Statistical analysis of Au grade has been undertaken on the raw (un-composited) assay values and the 1 m composites.

Estimation domains were used as primary subdivisions for the analysis.

Summary statistics for each estimation domain based on raw assay values have been prepared and compared with 1 m composite data in Table 14-6.

Table 14-6: 1 m Composite data Au g/t - Summary statistics by domain

Domain	Number of Samples		Mean Grade			Std Dev		CV	
	Raw	Composite	Raw	Composite	% Diff	Raw	Composite	Raw	Composite
10011	998	1038	3.68	3.68	0%	8.19	7.82	2.22	2.12
10012	89	90	5.8	5.78	0%	7.7	7.62	1.33	1.32
10021	138	139	2.43	2.43	0%	2.23	2.2	0.92	0.91
10022	423	440	3.68	3.67	0%	6.02	6.01	1.64	1.64
10031	505	519	4.21	4.21	0%	9.19	9.18	2.18	2.18
10032	227	240	3.35	3.35	0%	4.5	4.49	1.34	1.34
10041	109	110	3.9	3.9	0%	8.31	8.32	2.13	2.13
10042	208	210	2.88	2.89	0%	4.09	3.94	1.42	1.36
10051	10	10	2.59	2.59	0%	1.58	1.58	0.61	0.61
10052	12	12	13.51	13.51	0%	32.49	32.49	2.4	2.4
10061	3	3	13.14	13.14	0%	9.81	9.81	0.75	0.75
10070	57	67	2.97	2.97	0%	7.81	7.81	2.63	2.63
10080	276	307	2.8	2.8	0%	3.42	3.42	1.22	1.22
10090	70	75	2.96	2.96	0%	7	7	2.36	2.36
10990	15613	21505	0.2	0.2	0%	1.06	0.94	5.19	4.61
20011	197	196	9.36	9.35	0%	8.49	8.47	0.91	0.91

Log histogram plots for the same data filtered by estimation domain are presented in Figure 14-7 to Figure 14-10.

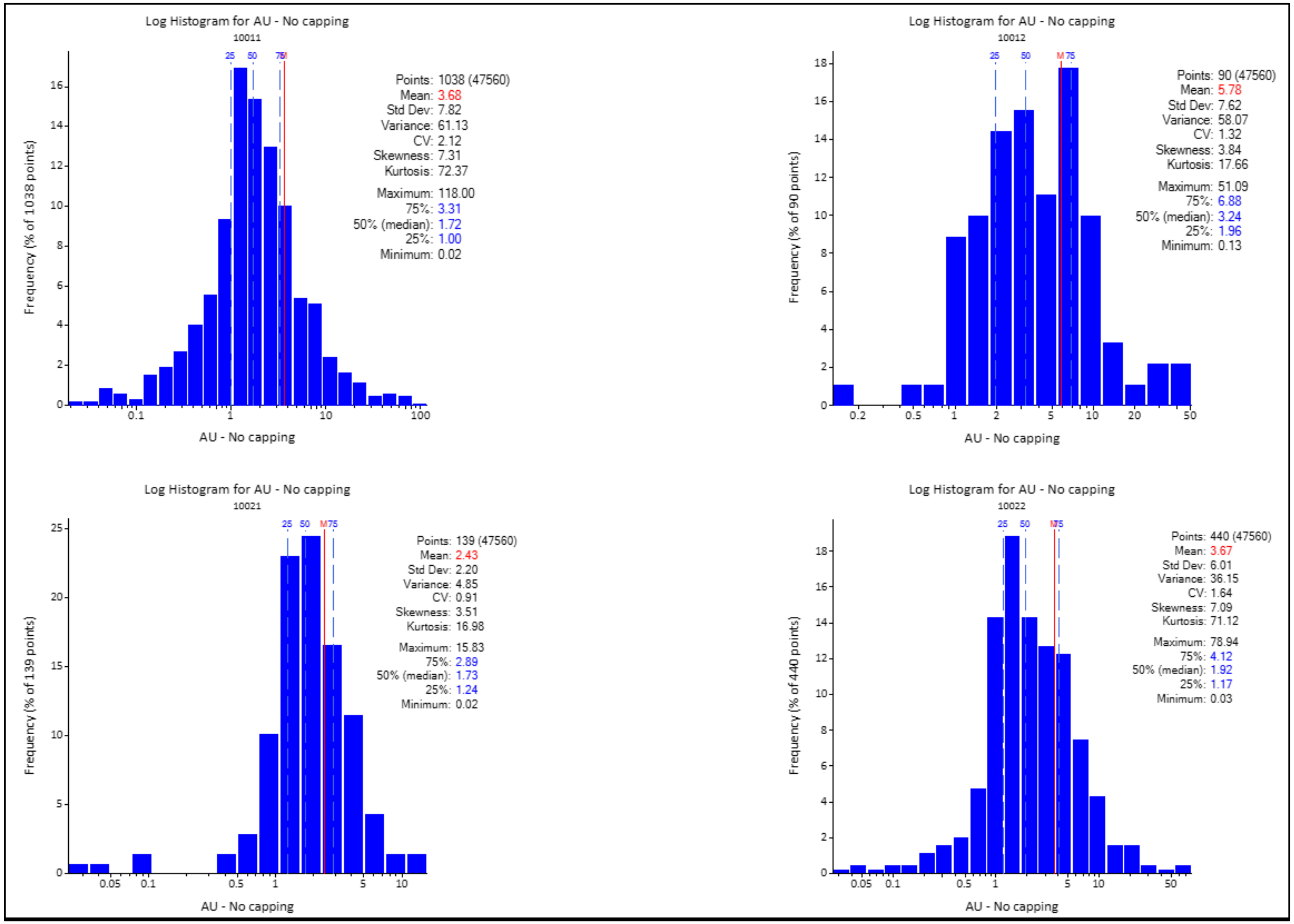

Figure 14-7: 1 m Composite Data - Log Histogram Plots - 10011, 10012, 10021, and 10022.

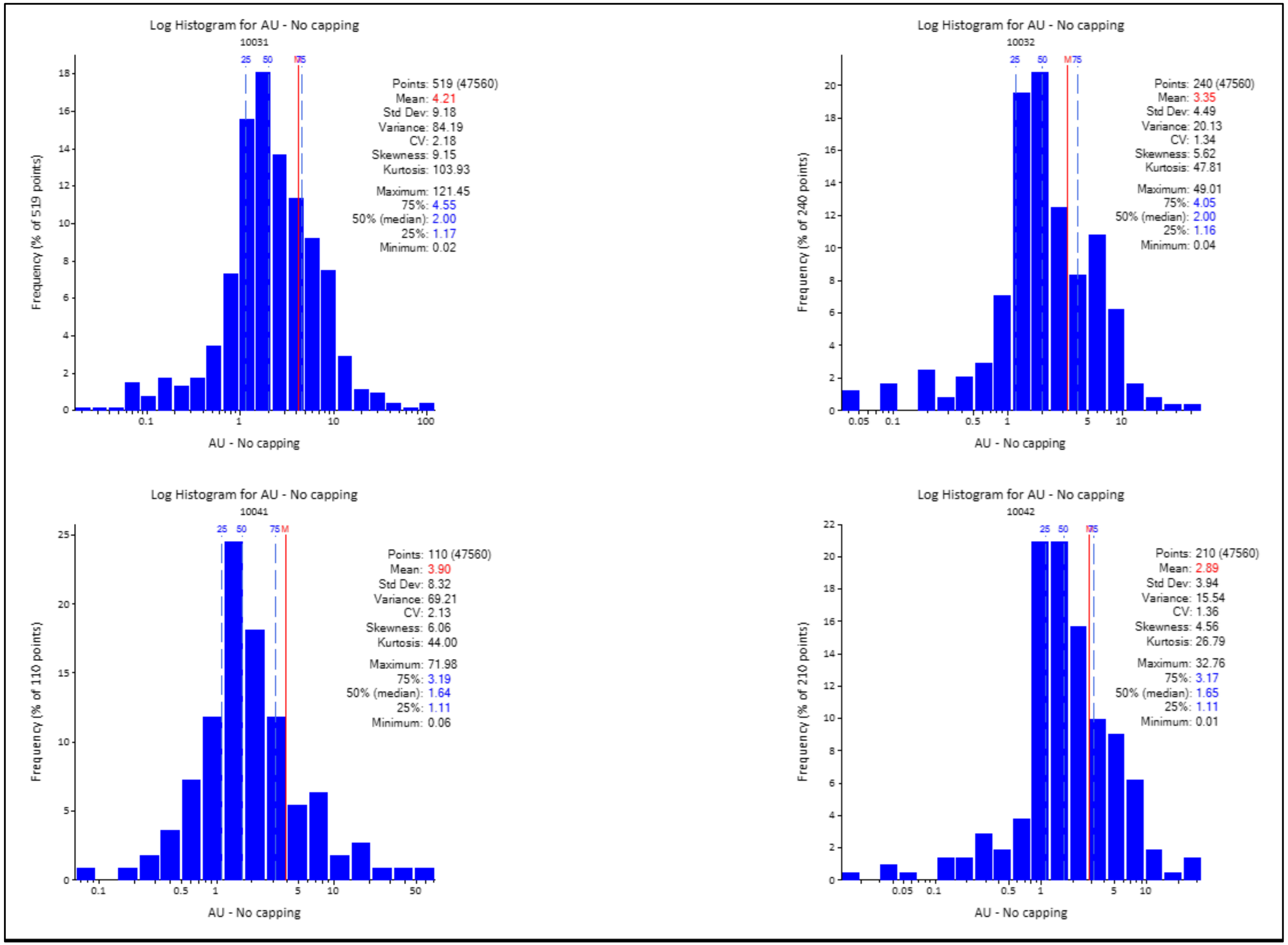

Figure 14-8: 1 m Composite Data - Log Histogram Plots - Estimation domains 10031, 10032, 10041, 10042.

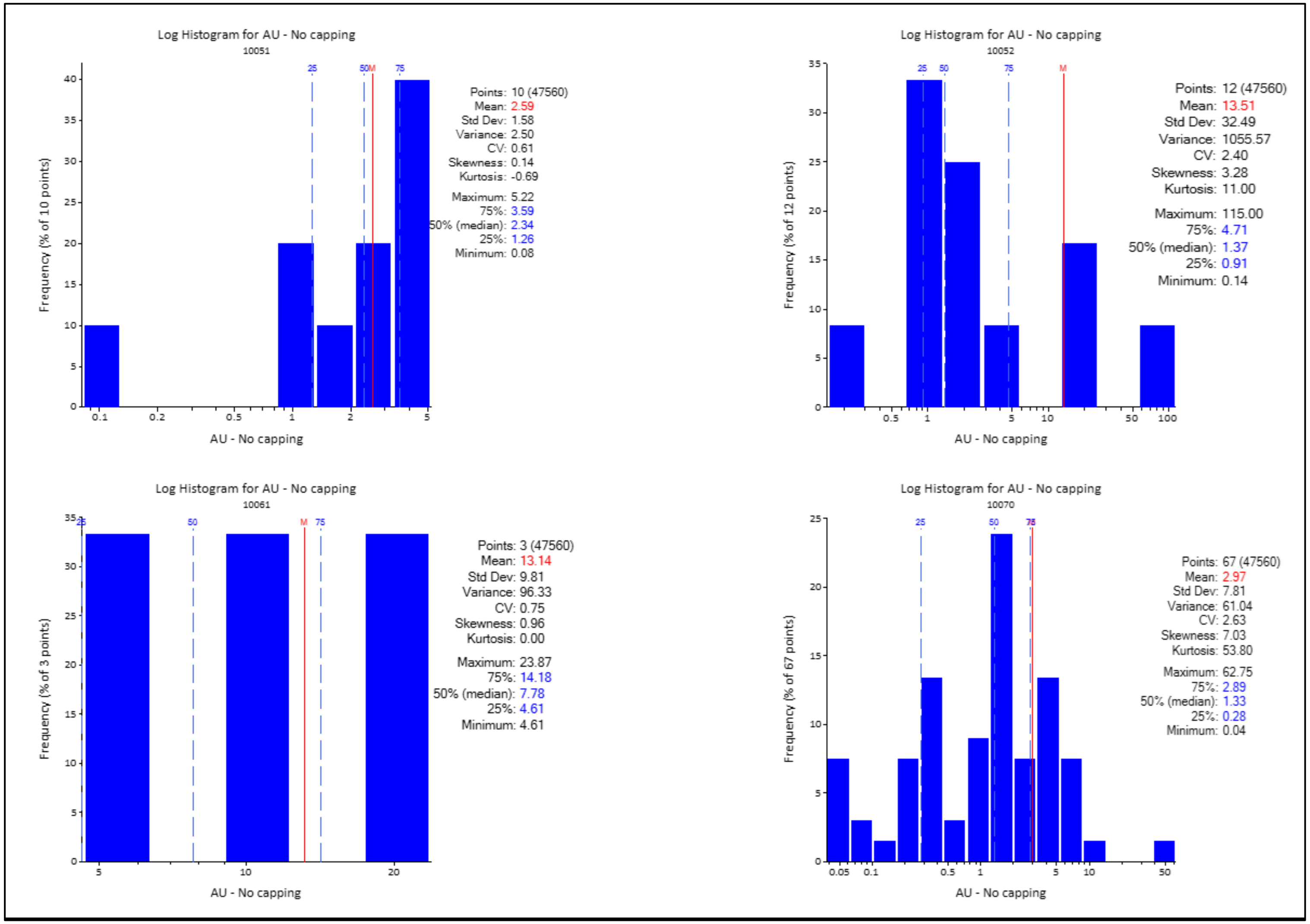

Figure 14-9: 1 m Composite Data - Log Histogram Plots - Estimation domains 10051, 10052, 10061, 10070.

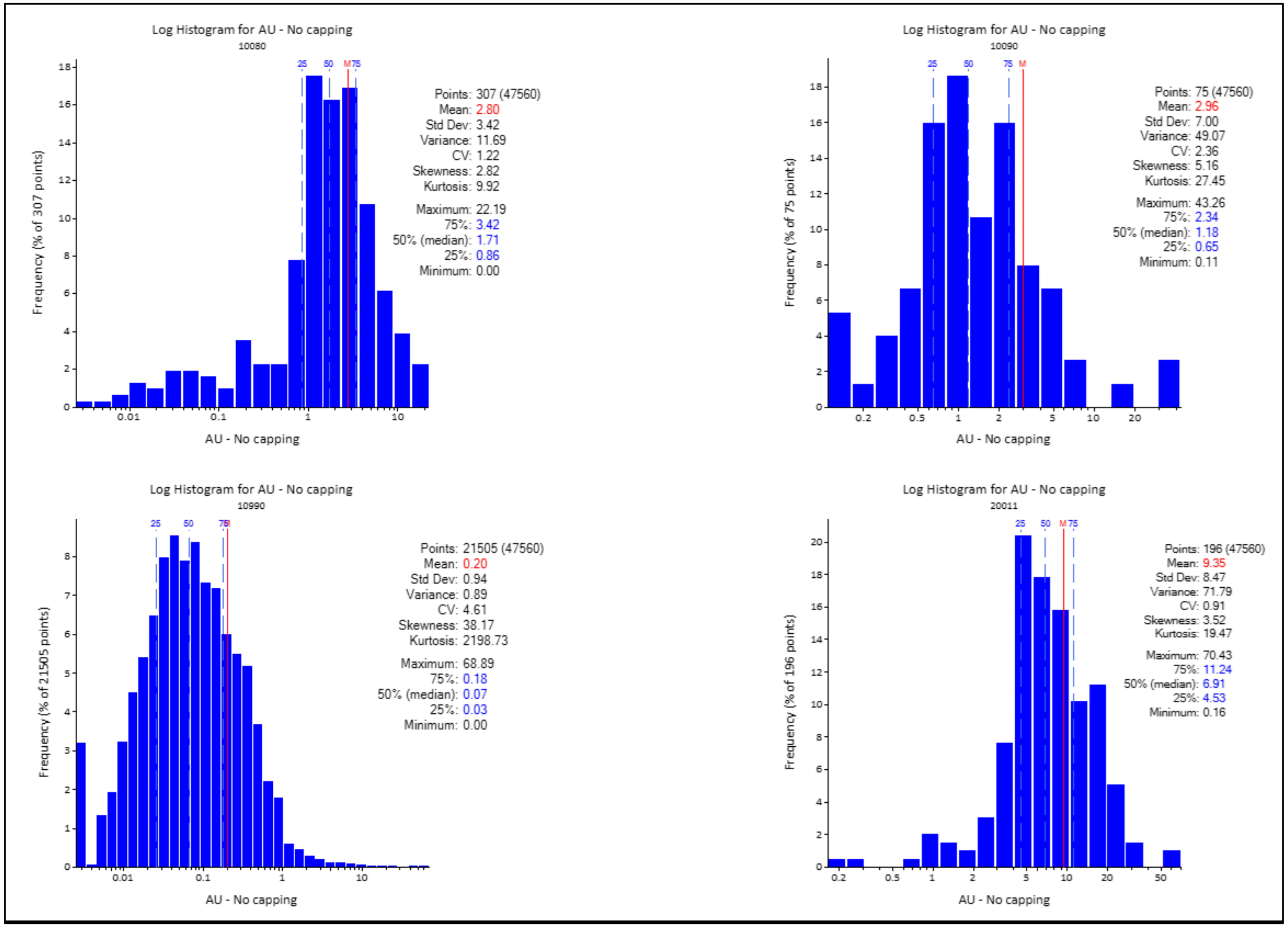

Figure 14-10: 1 m Composite Data - Log Histogram Plots - Estimation domains 10080, 10090, 20011 and dilution domain 10990.

14.8 Analysis of Mineralized Lodes of Minapampa (North and South Lodes)

A Quantile–Quantile (QQ) Plot was made of the 1 m composite data of the north and south mineralized lodes, to compare the distribution of the two data sets. The QQ plot of two similar distributions will be distributed along the first bisector of the graphic. If the two distributions differ, the QQ plot will move away from the straight line.

Figure 14-11 shows the QQ Plot of the north and south regions of the mineralized lodes. There is a good correlation between the two data sets. This correlation is confirmed in the Probability-Probability (PP) Plot (which represents the distributions for a given set of probabilities, between 0 and 1), shown in Figure 14-12.

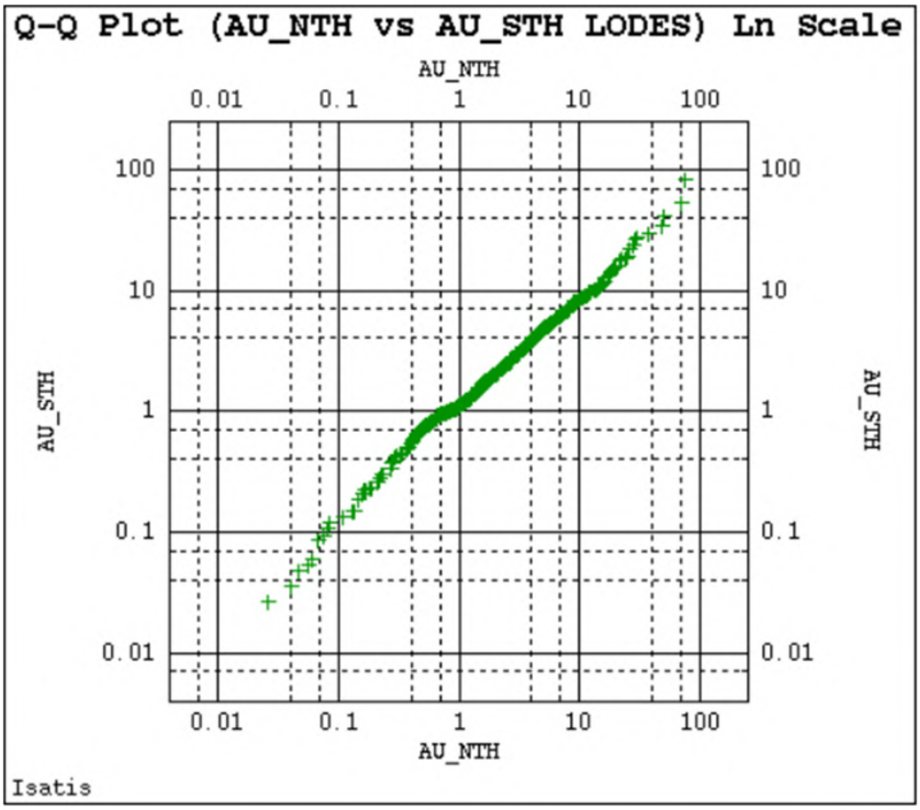

Figure 14-11: QQ Plot (Log Normal scale) - Mineralized Lodes North of the OCS versus Mineralized Lodes South of the OCS

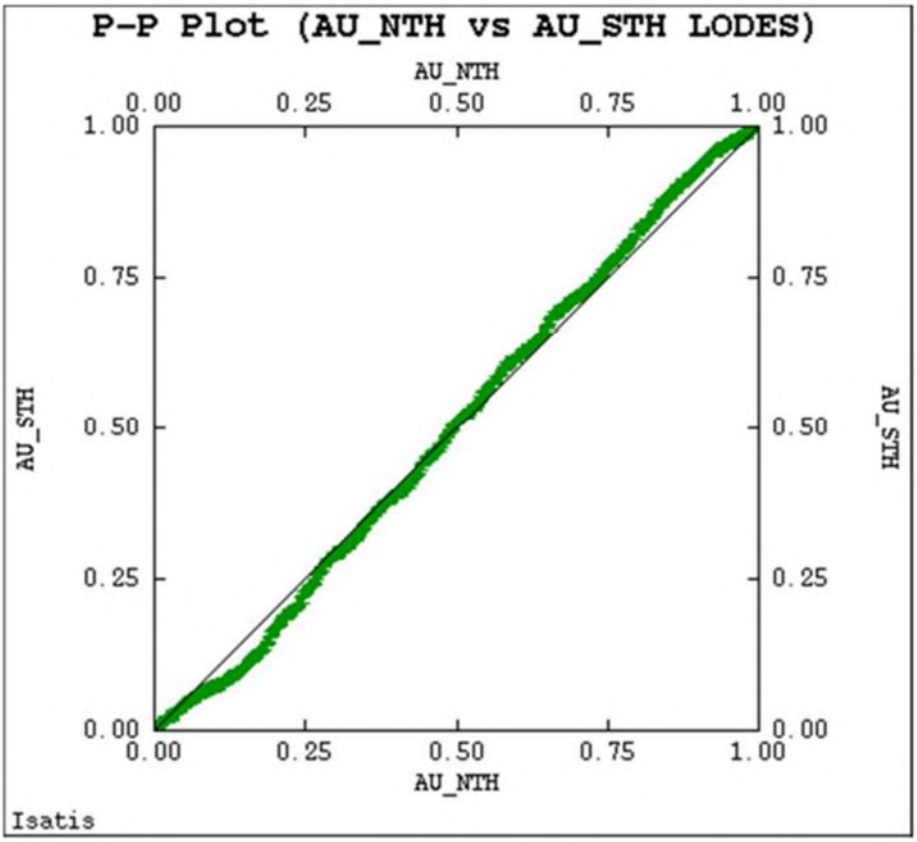

Figure 14-12: PP Plot – Mineralized Lodes North of the OCS versus Mineralized Lodes South of the OCS

Statistical analysis of mineralized lodes to north and south of the OCS indicate good correlation of grade distribution.

14.9 Top Cutting

Top cutting, or capping of outlier grades, was determined for each estimation domain.

A number of steps have been undertaken to determine the requirement for top cutting and to ascertain the reliability and spatial clustering of the high-grade composites. The top cutting assessment considered the following:

- Review of the composite data to identify data that deviates from the general data distribution. This was completed by examining the cumulative distribution function
- Comparison of the percentage of metal and data of the Coefficient of Variation (CV) affected by top cutting
- Visual 3D review to assess the clustering of the high-grade composite data.

Examples of top cut analysis have been provided in Figure 14-13 and Figure 14-14.

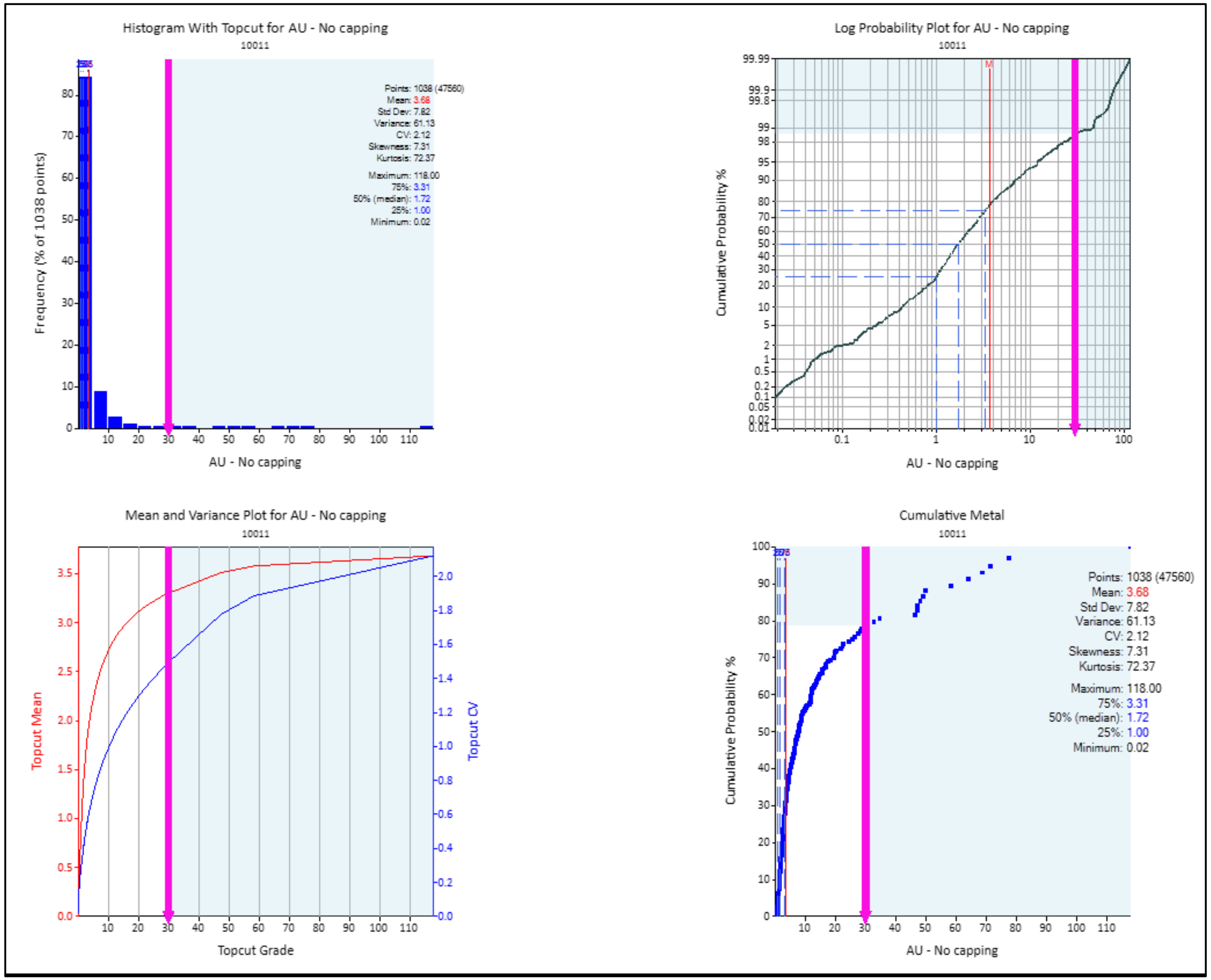

Figure 14-13: Example of the top cut analysis – Mineralized domain 10011 (MZONE=1 – Low grade domain)

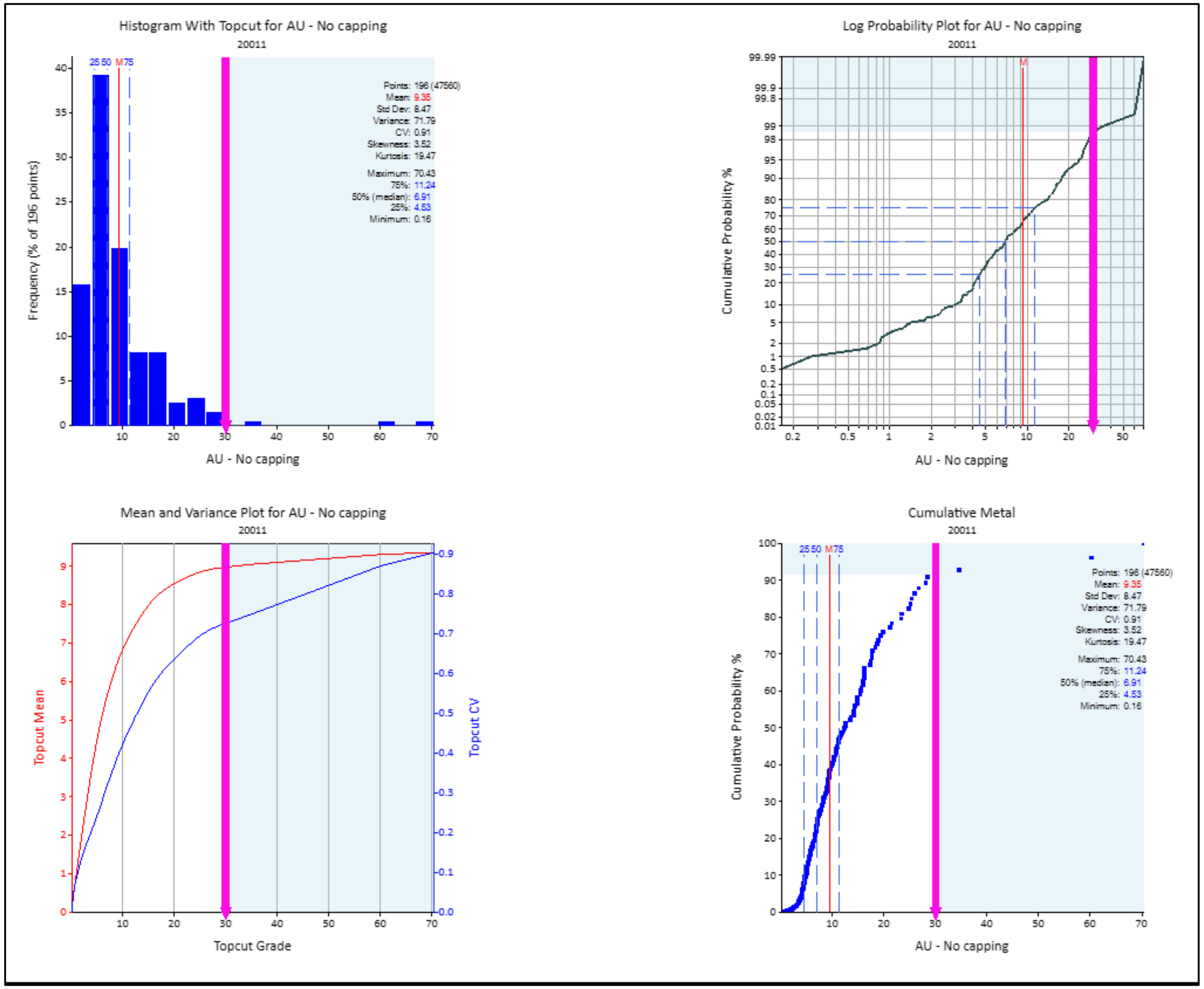

Figure 14-14: Example of the top cut analysis – Mineralized domain 20011 (MZONE=1 – High grade domain)

Based on the assessment, appropriate top cuts were determined for each estimation domain. The application of top cuts resulted in minor reductions in mean Au grade except for estimation domain 10052, where the cutting of one outlier value resulted in a 59% reduction in mean grade.

A top cut of 0.9 g/t Au was applied to estimation domain 10990, due to the presence of highly variable, higher grades within the dominantly lower-grade zone. Top cutting was required to reduce the amount of metal which would be artificially added during the estimation process in these zones due to outlier values having undue influence on the estimated mean grades.

Table 14-7 summarizes uncut and cut Au statistics for each estimation domain.

Table 14-7: Top cut statistics by estimation domain – Au g/t composite data

Domain	Number of Samples		Mean Grade			Top-Cut Value	Standard Deviation		Coeff of Variation		Max Un-Cut Grade	Top-Cut %ile
	Un-Cut	Top-Cut	Un-Cut	Top-Cut	% Diff		Un-Cut	Top-Cut	Un-Cut	Top-Cut		
10011	1038	15	3.68	3.29	-11%	30 (A)	7.82	4.88	2.12	1.48	118	1%
10012	90	4	5.78	5.08	-12%	20	7.62	4.61	1.32	0.91	51.09	4%
10021	139		2.43	2.43	0%	-	2.2	2.2	0.91	0.91	15.83	0%
10022	440	6	3.67	3.46	-6%	30 (D)	6.01	4.33	1.64	1.25	78.94	1%
10031	519	6	4.21	3.77	-10%	35 (B)	9.18	5.09	2.18	1.35	121.45	1%
10032	240	2	3.35	3.25	-3%	25 (C)	4.49	3.63	1.34	1.12	49.01	1%
10041	110	4	3.9	3.25	-17%	25	8.32	4.49	2.13	1.38	71.98	4%
10042	210	4	2.89	2.69	-7%	15	3.94	2.83	1.36	1.05	32.76	2%
10051	10		2.59	2.59	0%	-	1.58	1.58	0.61	0.61	5.22	0%
10052	12	1	13.51	5.59	-59%	20	32.49	7.39	2.4	1.32	115	8%
10061	3		13.14	13.14	0%	-	9.81	9.81	0.75	0.75	23.87	0%
10070	67	2	2.97	2.15	-28%	15 (E)	7.81	2.62	2.63	1.22	62.75	3%
10080	307		2.8	2.8	0%	-	3.42	3.42	1.22	1.22	22.19	0%
10090	75	3	2.96	2.21	-25%	15	7	3.03	2.36	1.37	43.26	4%
10990	21505	571	0.2	0.15	-25%	0.9	0.94	0.2	4.61	1.35	68.89	3%
20011	196	3	9.35	8.97	-4%	30	8.47	6.52	0.91	0.73	70.43	2%

(A) Lode 120 - 25 g/t / Lode 123 -15 g/t (B) Lode 308 - 15 g/t (C) Lode 411 - 18 g/t / Lode 412 - 15 g/t (D) Lode 257- 8 g/t (E) Lode 102- 6 g/t

14.10 Bulk Density Determination

The Ollachea database contains a total of 777 dry in-situ bulk density values. Bulk densities were estimated using the “Archimedean” water immersion method on approximately 10 cm billets of DDH core at recorded down-hole intervals.

The bulk-density analysis used in the 2012 MRE (Table 14-8) has been applied to this MRE.

Table 14-8: In-situ Dry bulk density (BD) samples – grouped by mineralization and dilution

Mineralized Samples		Host rock samples	
Count	Median BD (t/m^3)	Count	Median BD (t/m^3)
103	2.83	674	2.80

14.11 Variography

Variograms have been used to assess the spatial variability of gold.

Spatial variability is traditionally measured by means of a variogram, which is generated by determining the averaged squared difference of data points at a nominated distance (h), or lag. The averaged squared difference (variogram or γ(h)) for each lag distance is plotted on a bivariate plot where the X-axis is the lag distance, and the Y-axis represents the average squared differences (γ(h)) for the nominated lag distance.

In this document, the term "variogram" is used as a generic word to designate the function characterizing the variability of variables versus the distance between two samples. In this case, normal score variograms have been used; in general, the experimental traditional semi-variogram did not exhibit robust structures.

Fitted to the determined experimental variography are a series of mathematical models which, when used in the kriging algorithm, will recreate the spatial continuity observed in the variography.

Snowden's Supervisor software was employed to generate normal score variograms with a 2 structured spherical model and nugget effect; to model the spatial continuity.

Initially, a down-hole experimental variogram was calculated to establish the nugget effect for modelling the directional variograms. The geology and geometry of mineralization controls were also considered in selecting the orientations.

The variogram modelled for the northern mineralized lodes (MZONE 1 North lodes) had significantly greater sample support than the domains south of the OCS and in the Minapampa Far East zones, and displayed robust structure that could be confidently modelled. Therefore, the variogram model generated for the Northern Zone was also used for the estimation in the Southern Zone and Minapampa Far East. A separate variogram model was generated for the host rock zone (MZONE=99).

The nugget effect or short-scale variability in the mineralized lodes was 71% of the back transformed variogram, displaying a high degree of short-spaced variability. For MZONE 99, the nugget effect was approximately 29%.

Results of the variography analysis are given in Table 14-9 and graphically presented in Figure 14-15 and Figure 14-16.

Table 14-9: Normal Scores Variogram models used – Summary

MZONE	Variogram Orientations			Variographic parameters - back transformed						
	Dir 1	Dir 2	Dir 3	C0	C1		A1	C2		A2
MZONE 1	036->243	029->308	040->190	0.71	Dir 1	0.08	50	Dir 1	0.20	64
					Dir 2		46	Dir 2		52
					Dir 3		25	Dir 3		26
MZONE 99	036->243	029->308	040->190	0.29	Dir 1	0.37	20	Dir 1	0.34	113
					Dir 2		17	Dir 2		56
					Dir 3		3	Dir 3		17

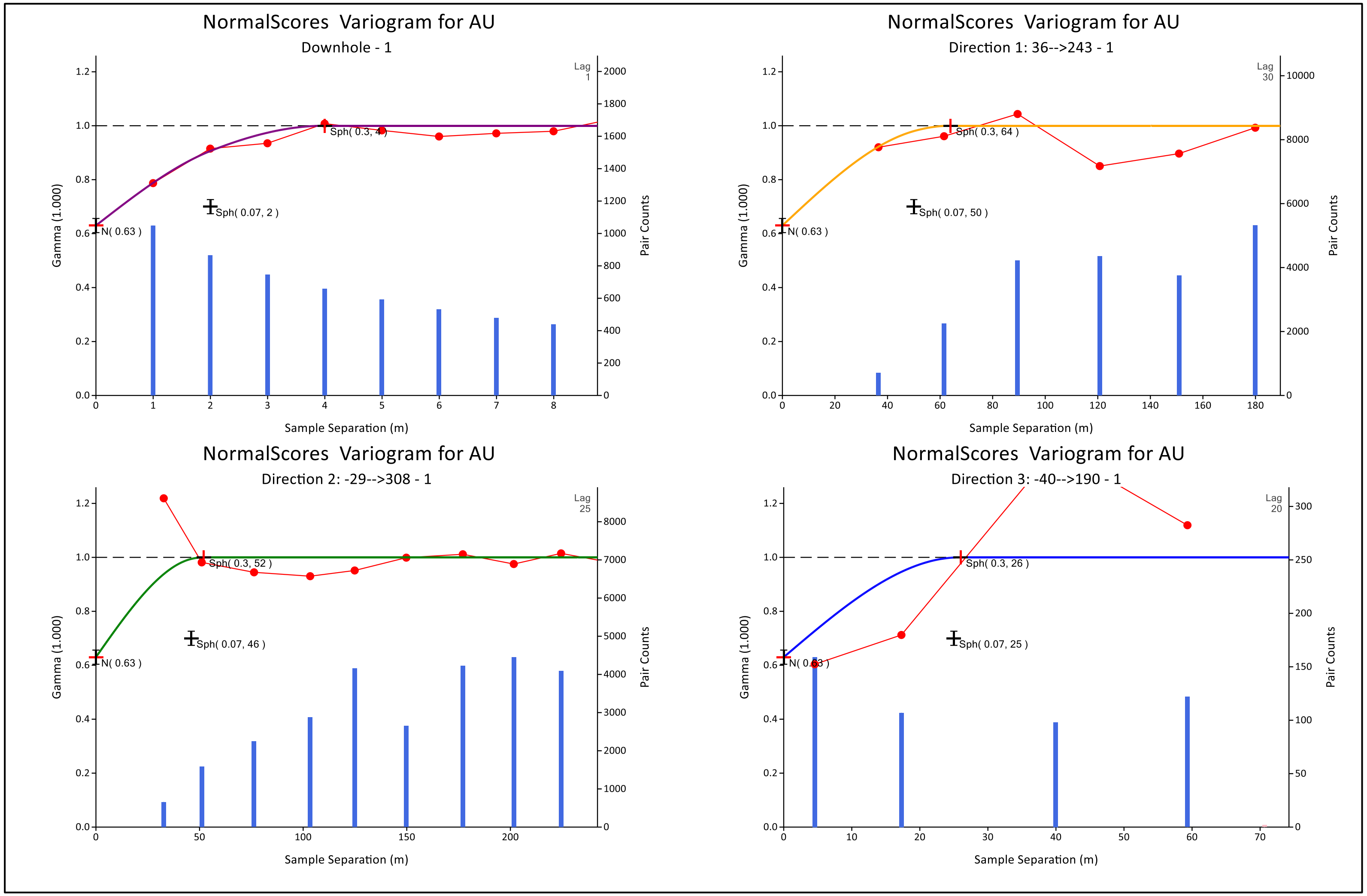

Figure 14-15: Northern mineralized lodes of MZONE=1 – Normal Scores Variogram Model

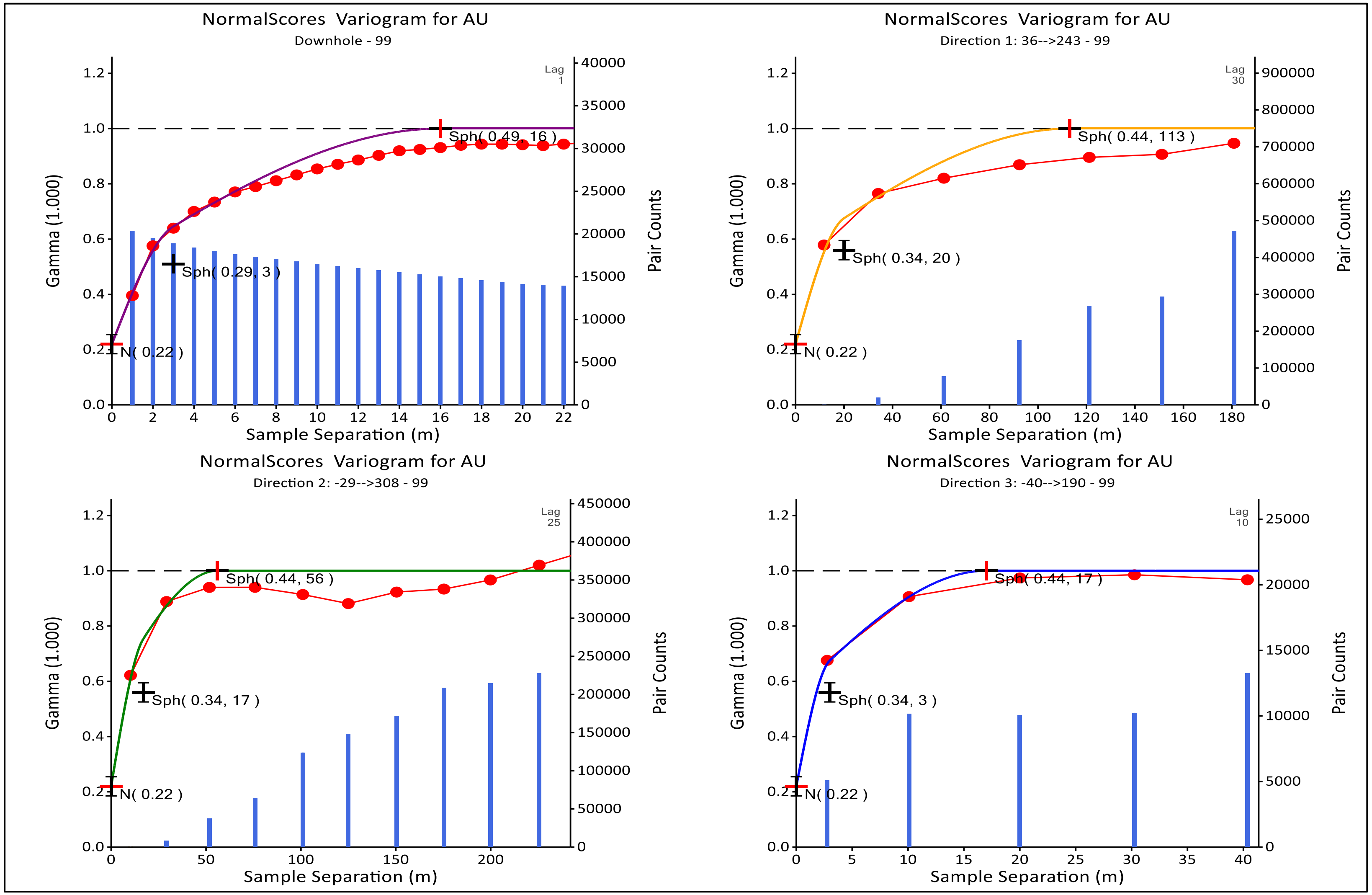

Figure 14-16: Host rock domain of MZONE=99 – Normal Scores Variogram Model

14.12 Block Model

14.12.1 Introduction

A 3D block model has been constructed using Datamine software for the Minapampa and Minapampa Far East zones. The 3D block model has used all the interpreted models (Lodes, high-grade domain) and surrounding host rock to input into mine design work.

14.12.2 Model Construction and Parameters

Block size has been selected based on the geometry of interpreted domains, data configuration and the expected mining method. A parent block size of 10 mE x 5 mN x 4 mRL was selected with sub-blocking to a 2 mE x 1 mN x 2 mRL cell size to improve volumetric representation of the interpreted wireframe models. Sufficient variables were included in the block model construction to enable grade estimation and reporting. Block rotation was not applied.

The surveyed topographic surface has been used to constrain the upper extent of the block model. The block model construction parameters are displayed in Table 14-10.

Table 14-10: Block model parameters

	East	North	Elevation
Origin	338,710	8,473,910	2,400
Extent (m)	1,600	1,000	800
Parent Block Size (m)	10	5	4
Sub-Block Size (m)	2	1	2
Number of Blocks	160	200	200

14.13 Grade Estimation

Grade estimation was performed using the Ordinary Kriging (OK) function provided with Datamine software.

The block model was coded with the number of composites used during the estimation process, the average distance to composites, Kriging Variance and Estimation Pass, which were later used in the determination of the resource classification.

14.13.1 Estimation Methods

The sample search strategy was based upon analysis of the variogram model anisotropy, mineralization geometry and data distribution. Hard boundaries were used in the estimation of individual lodes (LODE 101 to 601), and for MZONE 99.

Generally, the mineralized lodes are orientated with 90-degree strike, and a dip of 48 degrees to the north. However local strain partitioning around the OCS has steepened the dip.

To improve the search ellipsoids and maintain the changes in the orientation of the lodes, search ellipsoids with dynamic anisotropy were applied both in the ellipsoids and in the variograms.

The search strategy used in the block model is described in the following bullet points:

- For the estimated variable (Au), a two-pass estimation strategy has been applied, with progressively expanded sample searches applied to successive estimation passes only considering blocks not previously estimated.
- The sample search criteria by search pass in the mineralized lodes (LODE=101 to 601) in Minapampa and Minapampa Far East were:
  - First pass searches used an anisotropic range of 100 x 55 x 20 meters with the major axis oriented horizontally along-strike
  - If a block was not estimated in the first pass, a second pass search utilised a maximum range of 300 x 165 x 60 meters
  - The maximum number of composites used for any estimate was restricted to 12 composites for both passes
  - The minimum number of composites used for any first pass estimate was 6 composites
  - The minimum number of composites used for any second pass estimate was two composites
  - A maximum of four samples per drill hole were used.
- The sample search criteria by search pass in the host rock zone (MZONE = 99) were:
  - First pass searches used an anisotropic range of 30 x 20 x 10 meters with the major axis oriented horizontally along-strike for restrict the high grades in the host rock zone
  - Second pass searches used an anisotropic range of 115 x 90 x 10 meters with the major axis oriented horizontally along-strike. Grades were capped at 0.3 g/t Au to avoid smearing of higher grades.
  - If a block was not estimated in the first pass and a second pass search utilised a maximum range of 345 x 270 x 30 meters
  - The maximum number of composites used for any estimate was restricted to 12 composites variables for both passes
  - The minimum number of composites used for any first pass estimate was 6 composites
  - The minimum number of composites used for any second pass estimate was two composites
  - A maximum of four samples per drill hole were used.

- For all estimated zones, no octant search was applied; the only search restriction applied was the local change in variogram and search anisotropy for the mineralized lodes (LODE=101 to 601) and the host rock (MZONE=99), mentioned above
- All mineralized lodes divisions (LODE= 101 to 610) and MZONE=99 were treated as hard boundaries during the estimation process. Grade estimates were interpolated into parent cells and all sub-cells were assigned the parent cell grades
- A parent cell discretisation of 5 (X) x 5 (Y) x 4 (Z) was used
- For those blocks that had not been estimated after three search passes, the 25th percentile of each estimated lode was assigned.

14.13.2 Depletion of Underground Workings

Artisanal underground mining is ongoing in the area and as soon as practicable, a survey should be performed to understand as to precisely what depth the current artisanal workings have reached, and the resource updated accordingly.

14.13.3 Mining Lease Boundary

There is a small portion of the currently modelled mineralization that is not owned by MKK, and this area has been coded into the final block model. All blocks within this area have been flagged (MLEASE=0) and the MRE has only been reported for blocks within the mining lease (MLEASE=1), see Figure 14-17.

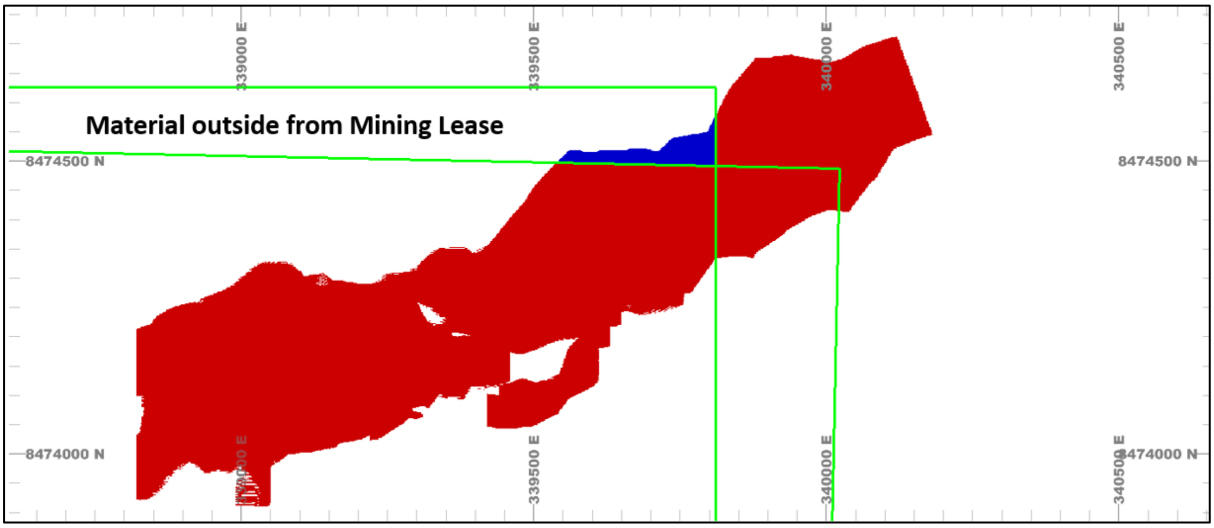

Figure 14-17: Plan view of blocks within mining lease for MZONE 1 to 9 (Red in mining lease / Blue outside mining lease)

14.13.4 Model Validation

Global Bias

Mining Plus has performed simultaneous estimates applying the inverse distance square (ID) and the nearest neighbour (NN) methods to determine the global bias for each mineralization domain. Mining Plus considers that the NN estimate provides a declustered mean and is suitable for global comparison and determination of global estimation bias.

Table 14-11 shows the comparison between the estimated OK and NN grades, where >10% difference is considered to be over- or under-estimated. In general, it is observed that the resource estimate presents an acceptable bias; however the following has been observed:

- Lodes 116, 202, 207, 357, 411, 413 and 463 of Minapampa are overestimated and represent 3% of the total mineralized domain (MZONE 1-9) volume. This overestimation reflects lodes with a low number of samples
- Lodes 123, 153, 154, 156, 253, 255 and 257, 358, 407 453, 205, 601 of Minapampa are underestimated and represent 4% of total mineralized domain volume. Lodes 102, 203 and 301 of Minapampa Far East are underestimated and represent 10% of total mineralized domain volume. This underestimation also reflects lodes with a low number of samples
- Due to the spacing of the drill holes in the Minapampa Far East Zone, the estimate in this area displays poorer validation results than the Minapampa Zone, and has been considered during the resource classification

Table 14-11: Global Bias comparison between model OK and NN model

Zone	LODE	Volume	AUOK	AUID	AUNN	% Volume MZONE 1-9	% Dif. OK vs NN	No. of Composites
Minapampa	99	56466814	0.13	0.13	0.13		-3%	21505
	101	1122575	2.83	2.79	2.90	14%	-3%	833
	112	25936	7.52	6.55	7.50	0%	0%	16
	113	10164	3.78	3.86	3.58	0%	5%	11
	116	6148	8.79	8.79	7.28	0%	21%	15
	117	15288	4.23	3.90	4.31	0%	-2%	22
	118	5928	3.57	3.63	3.90	0%	-8%	9
	120	79944	3.40	3.40	3.60	1%	-5%	54
	121	59736	5.34	5.25	5.50	1%	-3%	32
	123	4084	5.32	5.64	7.56	0%	-30%	5
	124	3372	3.88	3.82	3.99	0%	-3%	4
	125	37520	4.72	4.78	5.18	0%	-9%	28
	126	4104	5.42	5.99	5.72	0%	-5%	9
	151	32220	3.49	3.45	3.48	0%	0%	15
	152	2648	7.89	7.89	7.55	0%	5%	2
	153	32364	4.00	4.00	4.47	0%	-10%	19
	154	4364	5.27	5.07	6.37	0%	-17%	15
	155	31180	4.79	4.78	5.08	0%	-6%	28
	156	12132	8.82	8.25	10.66	0%	-17%	11
	201	6604	2.56	2.64	2.75	0%	-7%	7
	202	12036	3.23	2.73	2.90	0%	12%	12
	203	23296	2.82	2.82	2.65	0%	7%	23
	206	61856	1.87	1.85	1.86	1%	0%	54
	207	47472	2.85	2.80	2.52	1%	13%	32
	208	11188	2.27	1.95	2.37	0%	-4%	11
	251	352508	3.42	3.44	3.43	5%	0%	211
	252	93260	2.61	2.63	2.51	1%	4%	85
	253	45332	3.25	3.26	3.72	1%	-13%	52
	254	9504	2.49	2.43	2.36	0%	5%	21
	255	28336	5.75	5.94	6.74	0%	-15%	27
	256	43972	2.72	2.75	2.81	1%	-3%	30
	257	72080	4.74	5.73	6.27	1%	-24%	14
	303	267292	3.92	3.89	3.90	3%	0%	134
	305	936701	3.50	3.45	3.58	12%	-2%	371

Zone	LODE	Volume	AUOK	AUID	AUNN	% Volume MZONE 1-9	% Dif. OK vs NN	No. of Composites
	308	20464	4.12	3.76	4.23	0%	-3%	11
	345	15416	3.51	3.45	3.74	0%	-6%	3
	351	292132	3.20	3.19	3.34	4%	-4%	146
	354	36816	4.49	4.38	4.93	0%	-9%	30
	357	26820	3.53	3.28	2.84	0%	24%	34
	358	38620	2.17	2.15	2.56	0%	-15%	20
	361	12460	2.79	2.85	2.64	0%	6%	10
	407	8828	2.69	2.89	3.45	0%	-22%	7
	408	48664	1.72	1.75	1.81	1%	-5%	19
	410	19784	3.61	3.61	3.58	0%	1%	16
	411	91108	2.59	2.68	2.22	1%	16%	41
	412	13756	2.96	3.19	2.71	0%	9%	12
	413	22444	5.91	5.63	4.08	0%	45%	15
	451	185208	2.42	2.47	2.37	2%	2%	68
	453	49588	2.63	2.81	3.41	1%	-23%	18
	454	88596	2.50	2.44	2.66	1%	-6%	43
	456	77272	2.18	2.14	2.26	1%	-3%	34
	457	17452	1.67	1.65	1.71	0%	-3%	15
	461	2400	5.22	4.99	5.13	0%	2%	12
	462	24564	2.60	2.55	2.79	0%	-7%	11
	463	18740	6.37	6.43	5.37	0%	19%	9
	502	17088	2.55	2.51	3.07	0%	-17%	10
	555	9252	16.32	16.20	16.17	0%	1%	2
	556	49024	3.68	3.44	3.61	1%	2%	10
	601	1172	13.24	13.43	19.99	0%	-34%	3
Minapampa Far East	101	212936	8.93	8.86	9.29	3%	-4%	196
	101	422944	2.18	2.15	2.02	5%	8%	35
	102	268344	2.18	2.19	2.82	3%	-23%	27
	201	480704	2.48	2.54	2.36	6%	5%	71
	202	1231276	2.57	2.55	2.71	16%	-5%	236
	203	446896	2.05	1.97	2.45	6%	-16%	75
	301	35252	4.08	4.26	7.68	0%	-47%	5

Block Model Comparison against Drill Data

A detailed validation of the OK estimate was completed for the model and included both an interactive 3D and statistical review. The validation included a visual comparison of the input data against the block models grades in plan and cross section. The distribution of estimation outputs including search pass, average sample distance, number of contributing samples and drill holes were also reviewed.

Validation trend plots, or swath plots, are presented to graphically display comparison of the mean grade of 1 m composites against the estimated grades in the block model. The models were divided into slices by directions (Easting, Northing and RL) and average grades calculated for the various domains. Comparisons were made of the combined mineralized domains.

Dr. Fowler notes that 1 m composite Au grade in the block model honours general trends observed in the input data. Examples of three validation trend plots from MZONE 1 are provided in Figure 14-18 to Figure 14-20.

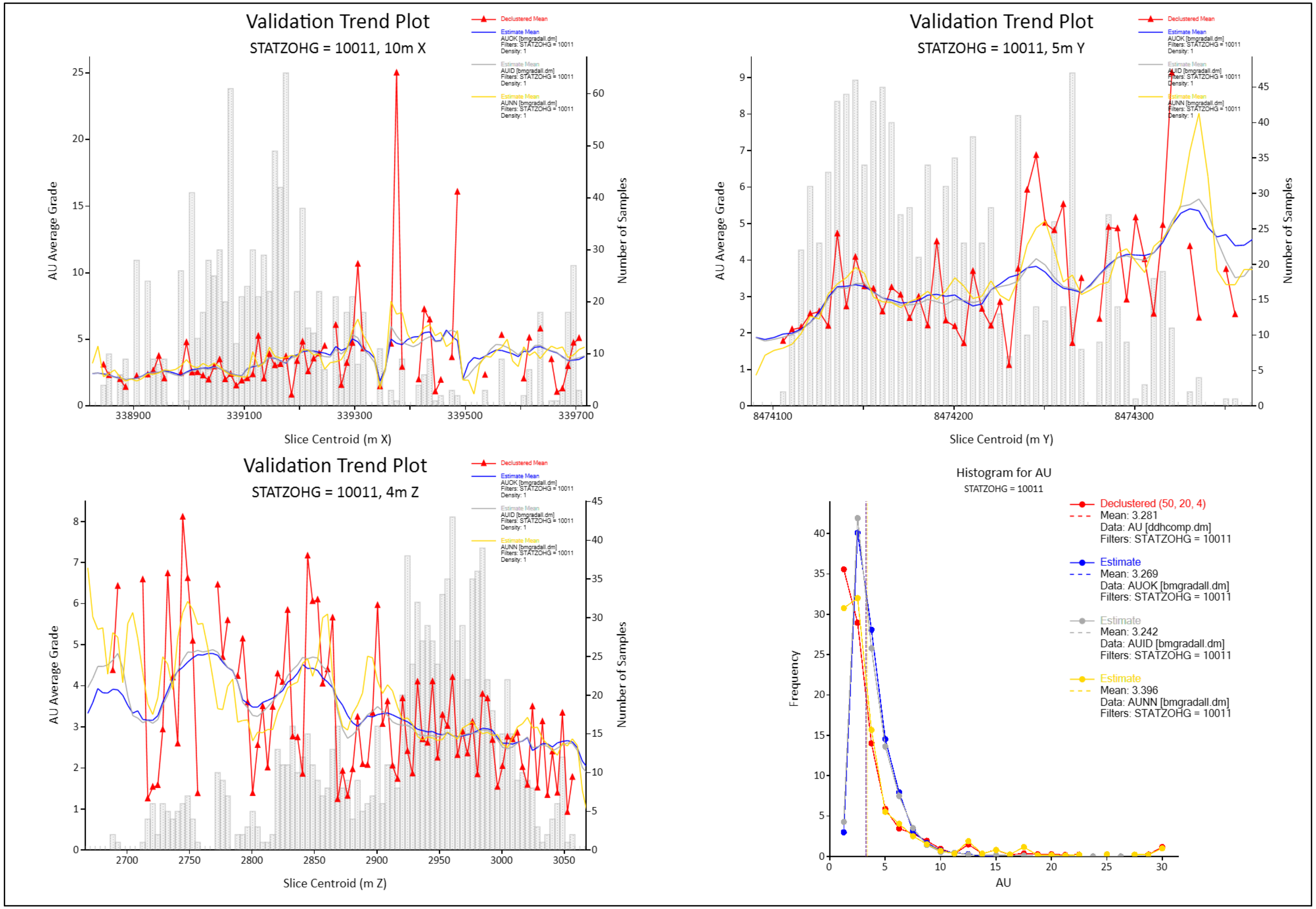

Figure 14-18: Comparison of model OK grade against composite sample declustered and model NN, on sections in the three axis: North, East and RL; (MZONE=1-Low grade)

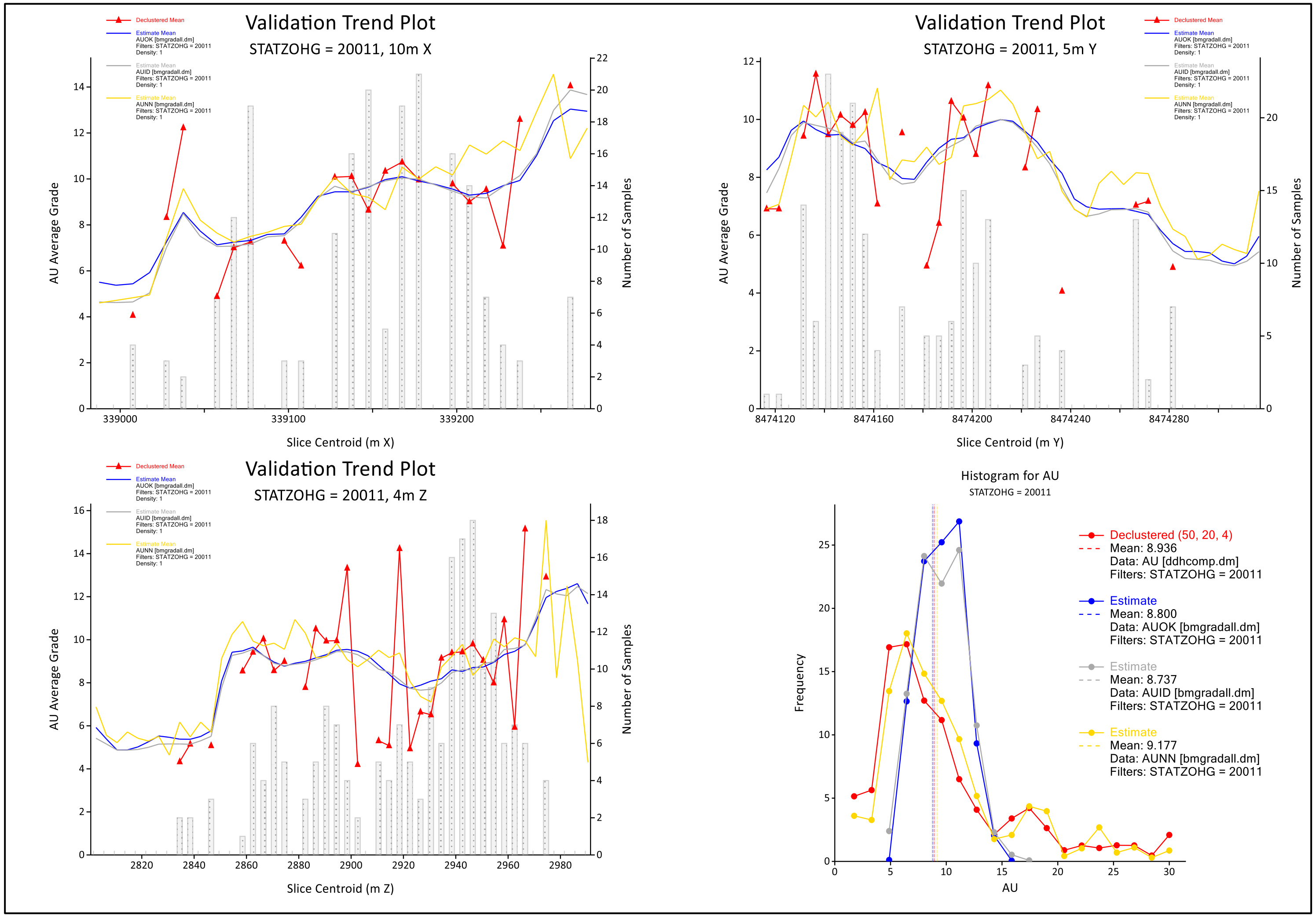

Figure 14-19: Comparison of model OK grade against composite sample declustered and model NN, on sections in the three axis: North, East and RL; (MZONE=1-High grade)

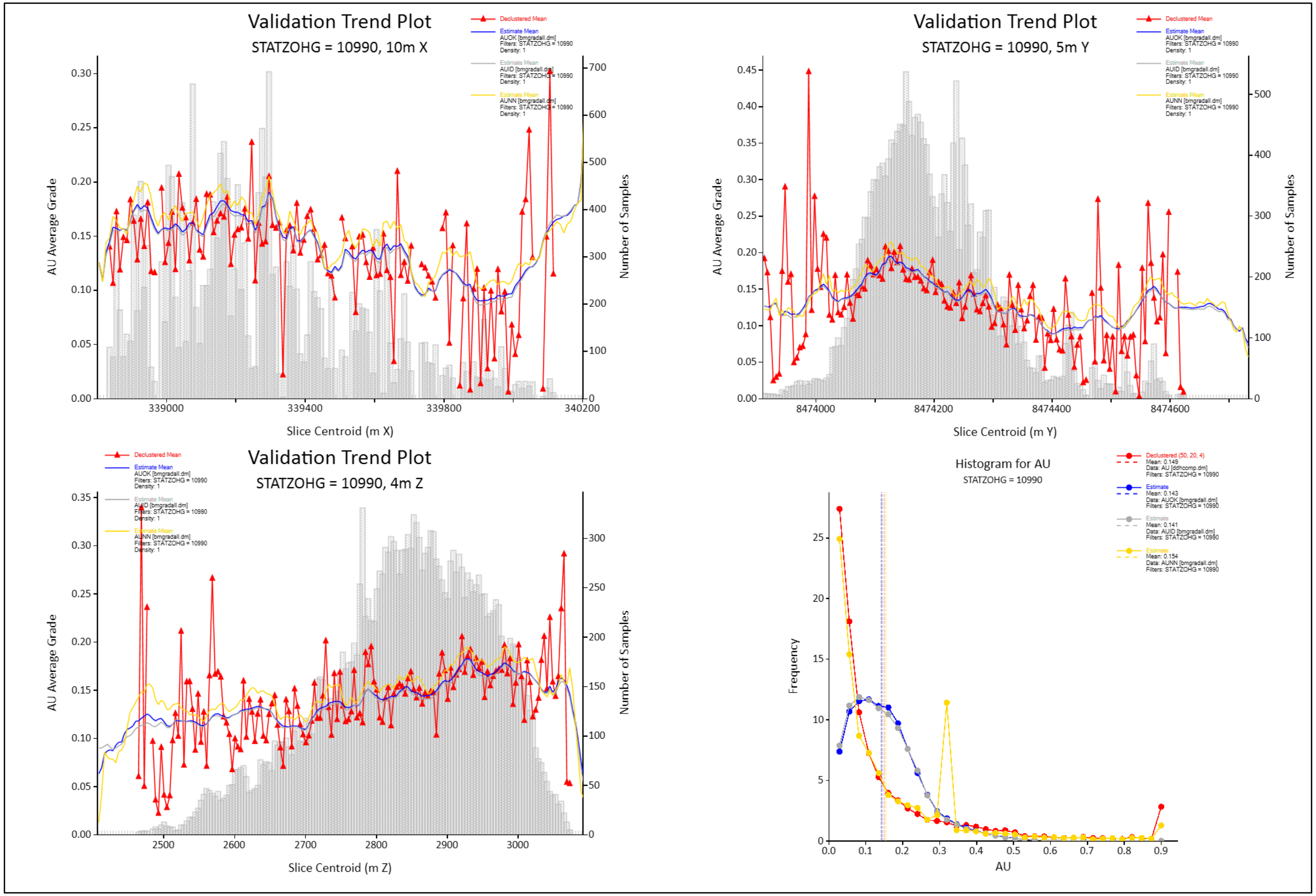

Figure 14-20: Comparison of model OK grade against composite sample declustered and model NN, on sections in the three axis: North, East and RL; (MZONE=99)

14.14 Mineral Resource Classification and Criteria

Inferred and Indicated Mineral Resource confidence categories have been assigned to blocks in the block model using criteria generated during validation of the grade estimates, with detailed consideration of the CIM Definition Standards (CIM, 2014) and the CIM best practice guidelines (CIM, 2019).

The resource classification coding was applied to the Ollachea block model based on a two-stage process. Blocks were first coded with confidence levels according to the data generated during estimation and other parameters, such as:

- Distance to the nearest sample used in the estimate
- Pass in which the estimate was generated
- Number of samples involved in the estimate
- Confidence in interpretations for individual lodes
- Assumptions about expected mining scenarios, mineralized continuity, and appropriate limits to lower cut-off grades applied to the model for reporting purposes.

A detailed review was then completed of the confidence levels in 3D prior to the construction of final wireframes outlining the Mineral Resource categories. The wireframes were used to select and flag the blocks with the final Mineral Resource classification.

An Inferred Mineral Resource confidence category was assigned for blocks:

- Having an estimated Au grade
- Within the mineralized lodes or host rock domains (MZONE 1 to 9 or MZONE 99)
- If less than four composited samples are used to represent a lode, it was assigned as Inferred (this occurred for LODE = 152, 345, 555 and 601).
- Located in a portion of the deposit with a density of drilling > 40 m x 40 m, up a maximum of 160 m.

The Indicated Mineral Resource confidence category was assigned to blocks where:

- They were located in a portion of the deposit with a density of drilling of approximately < 40 m x 40 m, and
- The slope of regression was >0.2 in the mineralized lodes or> 0.4 in the host rock zones

Mineral Resource categories are specified by the following RESCODE values in the block model:

- RESCODE=2 codes material classified as an Indicated Mineral Resource
- RESCODE=3 codes material classified as an Inferred Mineral Resource
- RESCODE=4 codes material that remains unclassified

A diagram showing the Mineral Resource classification of the mineralized zones (MZONE=1 to 9) is provided below in Figure 14-21 to Figure 14-24.

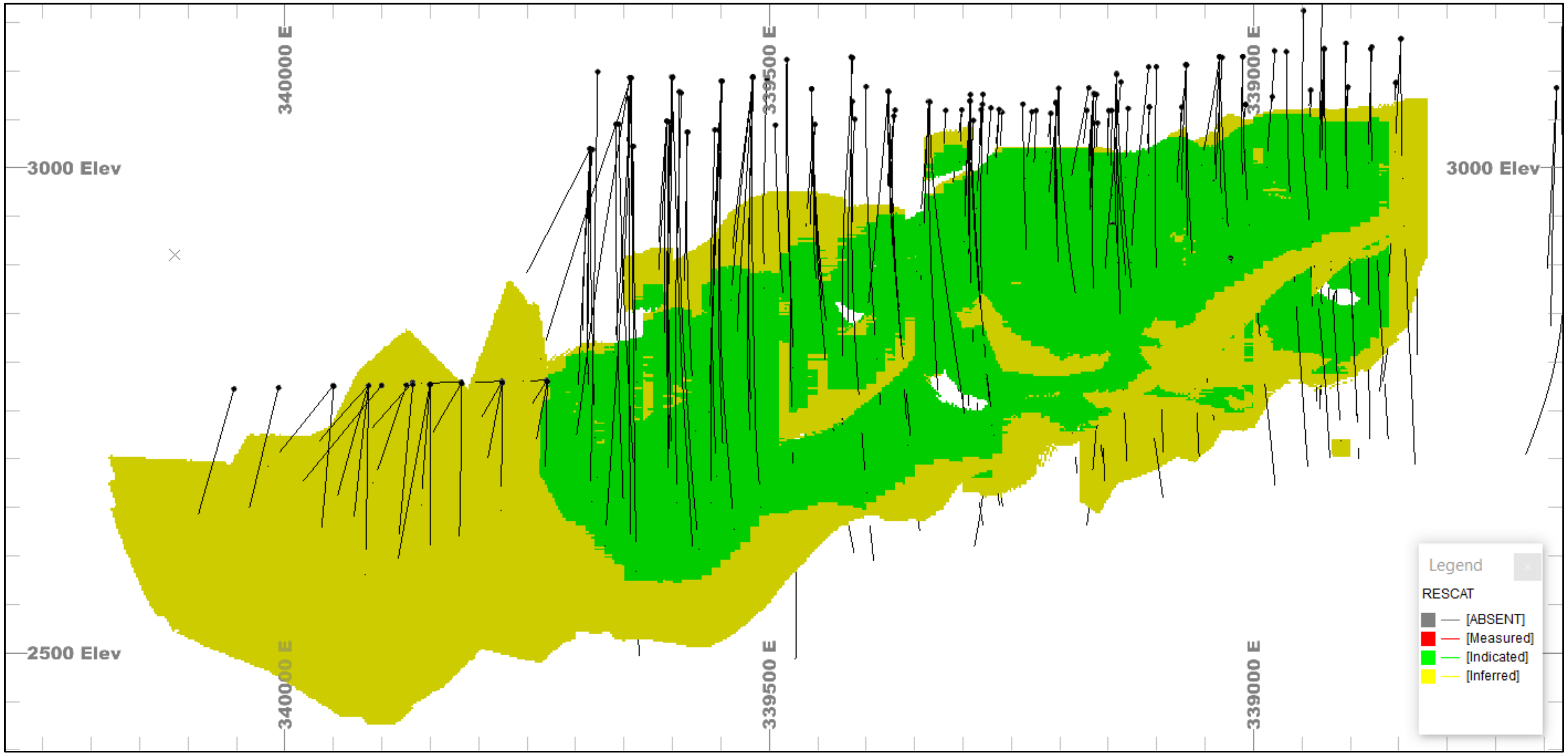

Figure 14-21: Perspective view, looking south of Block Model (MZONE 1 to 9) showing resource classification

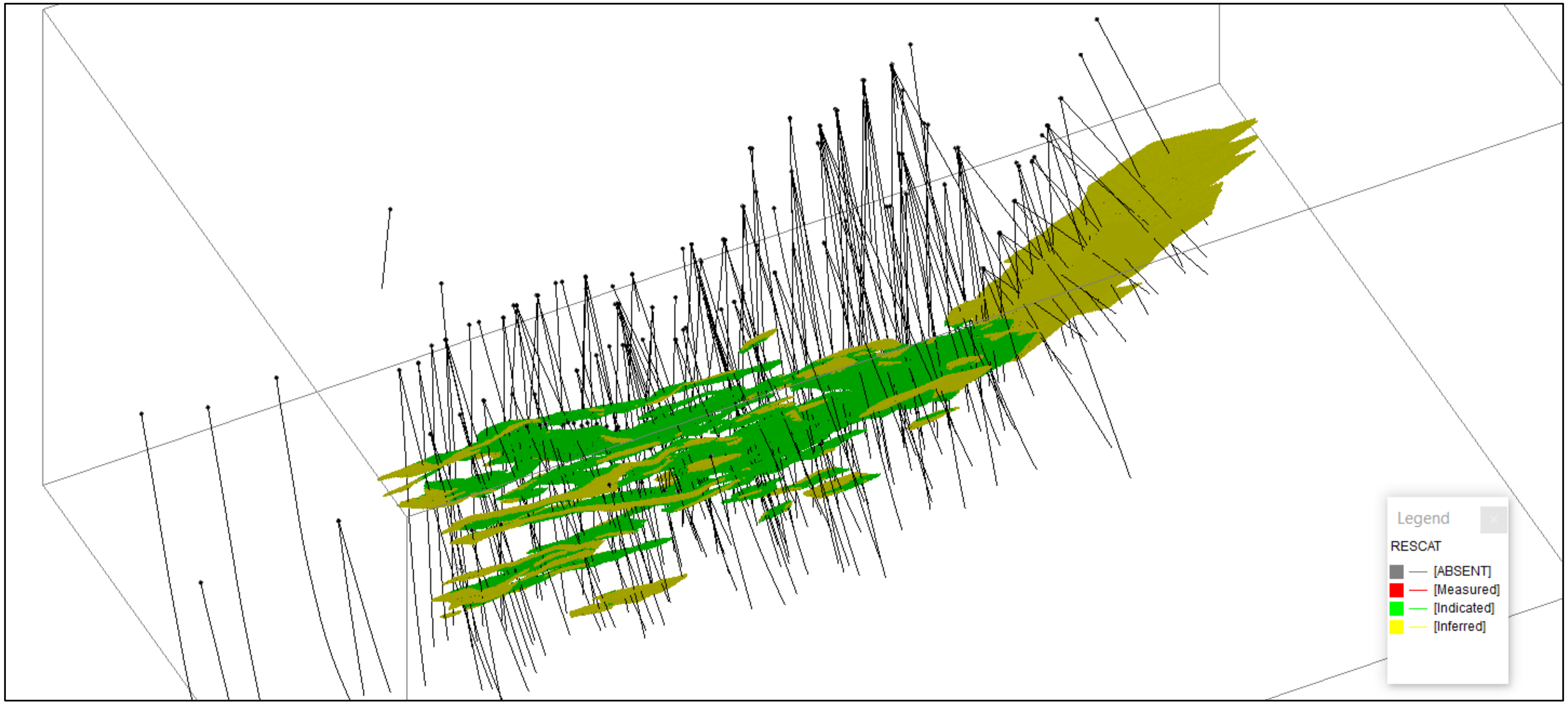

Figure 14-22: Perspective view, looking Northeast of Block Model (MZONE 1 to 9) showing resource classification

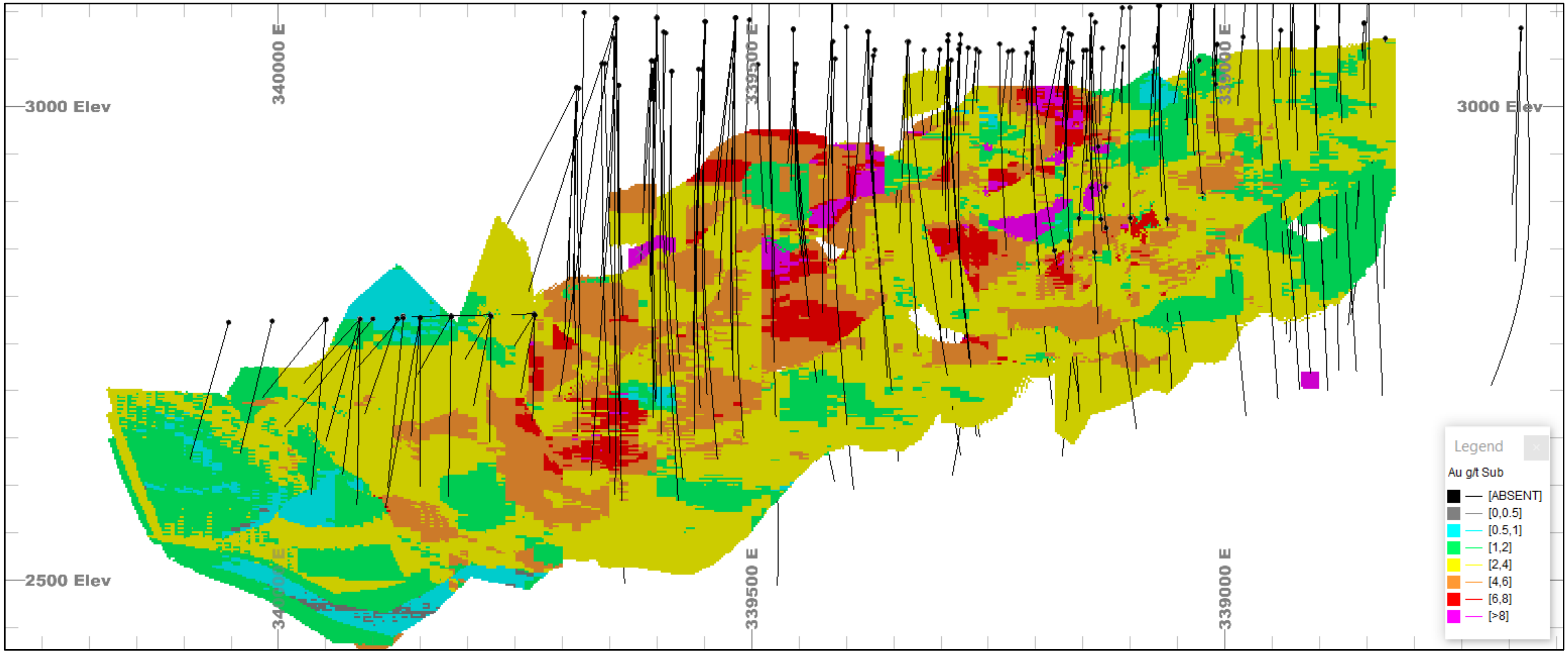

Figure 14-23: Perspective view, looking South of Block Model showing estimated grade (Au g/t) distribution in the mineralized lodes

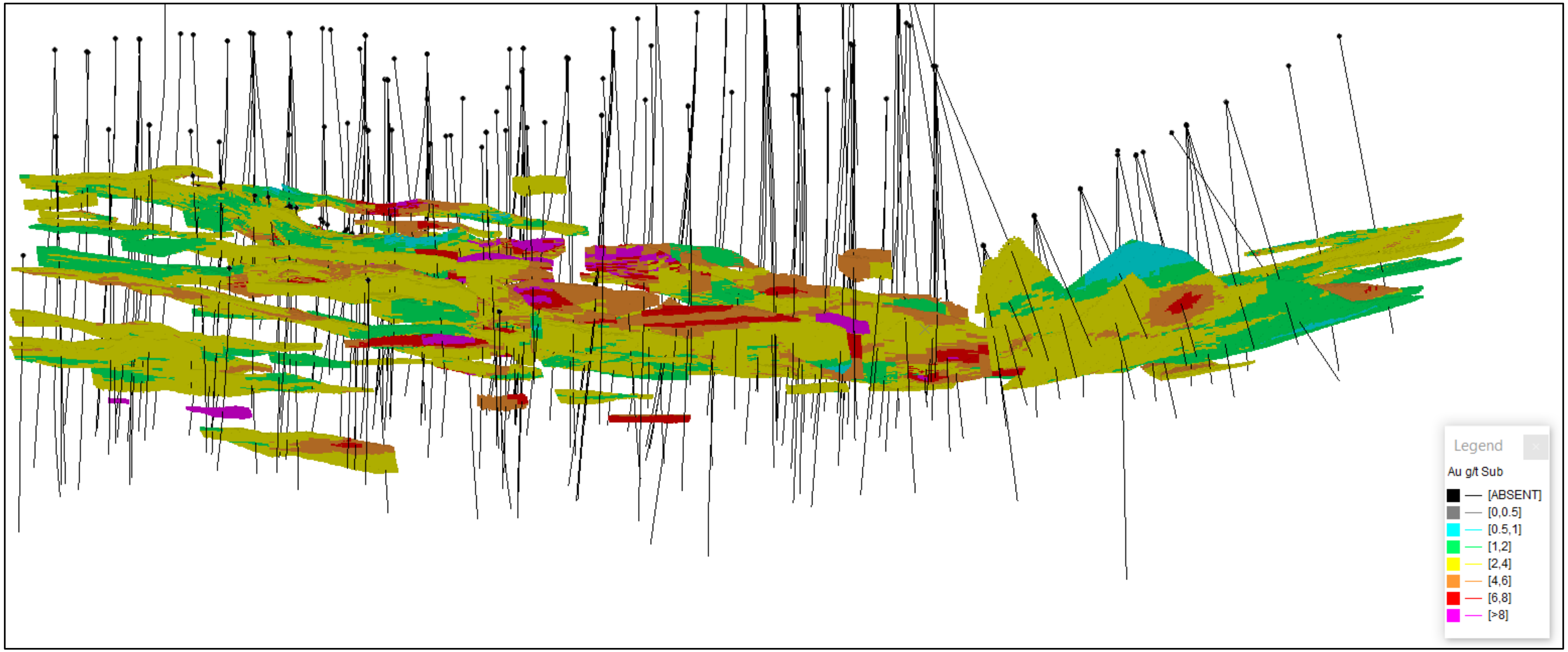

Figure 14-24: Perspective view, looking South of Block Model showing estimated grade (Au g/t) distribution in the mineralized lodes

14.14.1 Mineral Resource Statement

The Mineral Resource Estimate ("MRE") for the Ollachea Project, with an effective date June 30, 2021, has been estimated and classified based on the CIM's Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines (CIM, 2019) and is reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 ("NI 43-101").

Mineral Resources at Ollachea are considered potentially mineable by the long hole open stoping ("LHOS") method and are estimated based on drilling up to the fourth quarter of 2016. The MRE includes an update to the Minapampa Zone that was reported in the Feasibility Study in 2012, and a new zone at Minapampa Far East. The MRE is reported inside optimized underground stope shapes, which were built using a cut-off grade of 1.4 g/t gold and gold price of USD$1,700/troy ounce (see section 14.14.2).

The Qualified Person (QP) for the MRE according to the definition of NI 43-101 is Dr. Andrew Fowler, MAusIMM CP(Geo), Mining Plus Principal Geologist.

The following is a brief summary of the estimation process:

- Grades for diamond drill holes (192 drill hole) were composited to 1 m
- The MZONE 1-9 including a high-grade domain (named as mineralized domains) and host rock domain MZONE 99 were used as primary subdivisions for statistics and geostatistics
- The top-cutting was analyzed and applied by domain where applicable. In some cases, the individual lodes had top cuts applied
- Two variographic models were used in the estimate:
    - A variogram model from MZONE 1, containing the greatest proportion of mineralization and number of samples, was applied to the other estimation domains with insufficient sample pairs for meaningful variography
    - The host rock domain MZONE 99 was modelled and estimated separately.
- Dynamic anisotropy was applied in the search ellipsoids and in the variograms
- The estimation was completed using subcell models in Datamine mining software
    - The grade was estimated into parent cells with dimensions of 10 mE × 5mN × 4 mRL

- Subcells were used to conform to the geometry of the lodes, and the minimum dimensions of the subcells were 2 mE × 1 mN × 2 mRL

- The specific density applied to the block model are average densities based on 777 drill core samples, 2.83 t/m³ for mineralization and 2.80 t/m³ for waste.

The block model "ol17combfull.dm" was used to report with constraints: "AREA = 1 and 2, MSO = 1, RESCODE = 2 and 3 (Table 14-15).

The MRE comprises an Indicated and Inferred Mineral Resource as summarised in Table 14-12.

Table 14-12: Mineral Resource Estimate for the Ollachea Project by classification and Zone

Mineral Resource Estimate for the Ollachea Project – June 30, 2021						
Zone	Indicated			Inferred		
	Tonnes (Mt)	Au g/t	Au Ounces (Moz)	Tonnes (Mt)	Au g/t	Au Ounces (Moz)
Minapampa	10.7	3.28	1.13	1.8	3.0	0.2
Minapampa Far East	-	-	-	5.5	2.6	0.5
Total	10.7	3.28	1.13	7.3	2.7	0.6

1. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.
2. All figures are rounded to reflect the relative accuracy of the estimates.
3. The Mineral Resource was estimated by Ms. Muñoz and supervised by Dr. A. Fowler, MAusIMM, CP(Geo), Independent Qualified Person under NI 43-101., of Mining Plus Consultants who takes responsibility for it.
4. Composite gold grades were capped where appropriate.
5. Mineral Resources are diluted and are reported within optimized underground stope shapes.
6. The stope shapes were optimized at a gold cut-off value of 1.4 grams per tonne, considering metal prices of US$1700 per ounce of gold, and assuming metal recovery of 87% for gold, and total operating costs of $61.18/t.
7. Tonnages reported are metric tonnes and ounces of contained gold are troy ounces.
8. Mining Plus is not aware of any environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues that could materially affect the potential development of the Mineral Resource Estimate.

14.14.2 Reasonable Prospects for Eventual Economic Extraction Requirement

Mining Plus has reported the Mineral Resource inside optimized stope shapes in order to satisfy the "reasonable prospects for eventual economic extraction" requirement in accordance with NI 43-101 and the Mineral Resource and Mineral Reserves Best Practices Guidelines (CIM, 2019). These optimized shapes may include material <1.4 g/t Au; however, the average grade of the complete stope is >= 1.4 g/t Au cut-off grade. Similarly, outside of the optimized shapes, there are blocks with grades >=1.4 g/t; however, by applying reasonable mining parameters, they have become diluted to the point that they do meet the reasonable prospects for eventual economic extraction criterion.

The optimization of the underground designs was carried out using the Datamine Mineable Shape Optimizer (MSO) for an underground mining method by LHOS with paste fill. The optimization was applied to the subcell block model on Indicated and Inferred blocks and restricted to the Mining Lease Property (MLEASE=1).

The "ol17combclean.dm" block model was used for the optimization, which is a planning version of the subcell model of the Ollachea Estimate, where the unnecessary fields for the underground optimization process were eliminated.

Table 14-13 summarises the extraction costs and mineral processing costs, metallurgical recoveries and additional economic parameters applied to the calculation of the cut-off grade, Table 14-14 summarises the MSO Parameters.

Table 14-13: Cut-off grade calculation

COG Analysis	Value
Price (US$/oz)	1700.00
Exchange Rate ($/US$)	0.00
Mining Cost (US$/t)	38.43
Processing Cost (US$/t)	18.95
G&A (US$/t)	3.79
Total Cost US$/ton	61.18
Cut-off g/t	**1.4**

Table 14-14: MSO Parameters

MSO Parameters	Unit	Value
Cut-off grade	g	1.4
Min mining width	m	2
Max mining width	m	100
Level spacing	m	15
Section spacing	m	20
Max waste fraction	m	NA
Min waste pillar width	m	7.5
Near wall dilution	m	NA
Far wall dilution	m	NA
Min dip angle	deg	40
Max dip angle	deg	140
Max strike angle	deg	80
Max strike angle change	deg	15
Max side length ratio		2.25
Default dip	deg	50
Default strike	deg	0

14.14.3 Summary of Block Model fields

The list fields present in the final model (ol17combfull.dm, containing 2,032,370 records) are presented in Table 14-15.

Table 14-15: Block model parameters and fields

Variable	Default	Description
Zone	1	1=Minapampa, 2=Minapampa Far East
AU	-999	Final Au
AUID	-999	Au estimated with ID2 with gold topcutting
AUOK	-999	Au estimated with OK with gold topcutting
AUNN	-999	Au estimated with NN with gold topcutting
AUOKNCUT	-999	Au estimated with OK with gold no topcutting
DENSITY	2.8	Bulk density - assigned value of 2.80m³/t where MZONE=99 and 2.83m³/t where MZONE=1-9
ESTDOM	-999	Estimated domain (LODE+HG+AREA*100000)
HG	10000	High grade zone: 10000=HG, 20000=LG
INSITU	1	Numeric depletion flag. INSITU 0=material has been mined / removed. 1=material is insitu.
KV	-999	kriging variance
LAGR	-999	Lagrange value
LODE	-999	Individual Lodes used to produce each MZONE eg. Lode 101 - 156 (MZONE 1). 200 series for MZONE 2 etc. The lodes are also numbered depending on what side of the Oscco Cachi Fault they are located. For example MZONE 1 (100 - 150 - north side, 151 - 199 - south side), MZONE 2 (100 - 250 - north side, 251-299m- south side) etc.
MINDIS	-999	Average Sample distance (transformed)
MZONE	-999	1 = mined, 0 = un-mined (in-situ)
MLEASE	0	1 = inside mining lease, 0 = outside mining lease
MSO	0	1 = inside MSO stope, 0 = outside MSO stope
NSAMP	-999	Number of composites
OXSTATE	0	Default oxidation state - 0=Fresh
PASS	-999	Estimation pass in which the block estimate was generated for Au.
RESCODE	-999	1 = Measured, 2 = Indicated, 3 = Inferred, 4 = Unclassified
STATZOHG	-999	Estimation domains codes
TRDIP	-999	Dip estimated
TRDIPDIR	-999	Dip direction estimated
XMORIG	338710	X coordinate of model origin.
YMORIG	8473910	Y coordinate of model origin.
ZMORIG	2400	Z coordinate of model origin.
XINC	10	Cell X dimension
YINC	5	Cell Y dimension
ZINC	4	Cell Z dimension
NX	160	Number of parent cells in the X direction.
NY	200	Number of parent cells in the Y direction.
NZ	200	Number of parent cells in the Z direction.

14.14.4 Grade Tonnage Curve

Table 14-16 shows the grade tonnage values for different gold cut-off grades. The same data is shown graphically for Indicated and Inferred categories in Figure 14-25 and Figure 14-26.

Table 14-16: Tonnage and Grade by cut-off and Resource Category

Cut off	Indicated		Inferred	
Au g/t	Tonnes (t)	Au g/t	Tonnes (t)	Au g/t
0.00	65,854,720	0.77	109,658,288	0.34
0.20	27,824,479	1.67	20,778,739	1.42
0.40	13,498,263	3.15	10,585,302	2.55
0.60	11,818,604	3.54	10,035,399	2.66
0.80	11,726,736	3.56	9,770,784	2.72
1.00	11,665,090	3.57	9,360,299	2.80
1.20	11,558,184	3.59	8,990,723	2.87
1.40	11,301,526	3.65	8,634,619	2.93
1.60	10,780,490	3.75	7,917,123	3.06
1.80	10,227,633	3.86	7,250,998	3.19
2.00	9,568,380	4.00	6,401,432	3.36
2.20	8,810,450	4.16	5,635,544	3.53
2.40	8,022,648	4.34	4,909,151	3.71
2.60	7,250,965	4.54	4,138,893	3.94
2.80	6,481,104	4.76	3,524,456	4.16
3.00	5,752,177	4.99	3,123,286	4.32
3.20	5,061,670	5.25	2,661,283	4.53
3.40	4,515,832	5.48	2,247,187	4.76
3.60	4,014,040	5.73	1,900,433	4.99
3.80	3,592,530	5.97	1,613,833	5.22
4.00	3,240,060	6.20	1,419,684	5.40

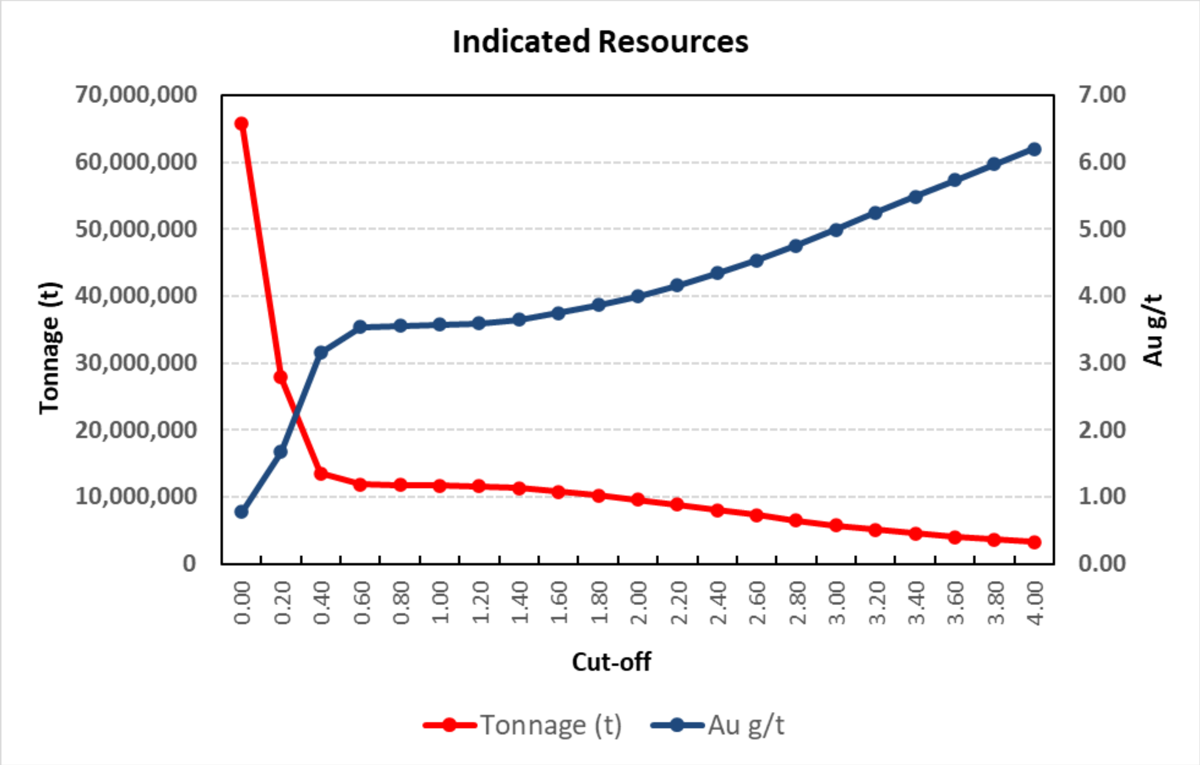

Figure 14-25: Indicated Resources - Grade and Tonnage by Au cut-off grade

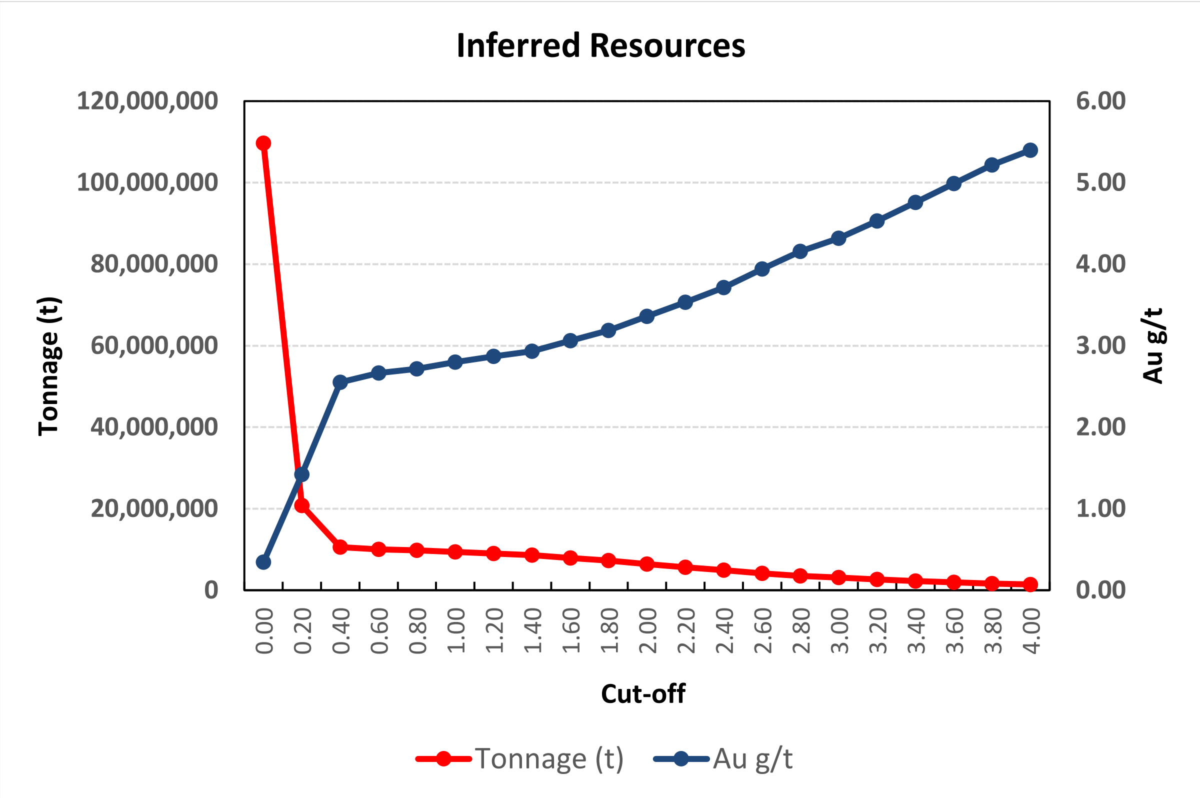

Figure 14-26: Inferred Resources - Grade and Tonnage by Au cut-off grade

15 MINERAL RESERVE ESTIMATES

There are no Mineral Reserves declared in this Technical Report.

16 MINING METHODS

Edgard Vilela (QP) is responsible for the content of Section 16 of the Technical Report.

16.1 Background

In 2012, the Ollachea Feasibility Study (2012 FS) was completed and announced to the market. AMEC assumed responsibility for all engineering aspects, whilst Coffey Mining completed the Resource and Mine Design components. This Study defined Bottom-up Long Hole Open Stoping (LHOS) with paste-fill as the preferred mining method with a production rate of 3000 tonnes per day (“tpd”).

In 2013, Minera Kuri Kullu (“MKK”) completed an updated geological interpretation for the Ollachea mineralization. Based on the updated geological interpretation, a revised resource block model was developed. Confidence in the geological interpretation improved.

In 2014, Mining Plus was commissioned by Minera IRL to produce an Optimized Mine Plan (MP 2014) utilising the updated resource model, aiming to maximise project NPV. Also, through undertaking this work, improvements were made to the precision of the mining and geotechnical aspects of the 2012 FS, with a focus on dilution, MSO optimisation, development and production schedule and ground support for the planned mining method. The fundamentals of the 2012 FS (production rate, mining method, portal locations, etc) were not modified in the 2014 study.

In 2016, Mining Plus was engaged to investigate options to ramp up production rather than starting at 3000 tpd as defined in the 2012 FS. Mining Plus was able to demonstrate at a conceptual level that it was possible to ramp up production whilst maintaining an acceptable gold production by targeting the higher-grade material early in the mine life.

In 2017, in an internal report titled “Ollachea Mining Optimization Study 1500 to 3000 Tonnes Per Day” (MP 2017), the ramp up approach was further investigated using a modified block model (an internal model with the higher-grade domain re-blocked) to investigate the feasibility of mining at a lower production rate and a higher cut-off grade.

The present technical report builds upon the work commenced in 2017.

16.2 Mining Method

The 2012 FS considered bottom-up Long Hole Open Stoping (LHOS) with paste-fill as the optimal mining method for the Property.

Edgard Vilela (QP) considers that LHOS with paste fill is the optimal mining method for the mineralization reported at the Property. Edgard Vilela (QP) notes that mineralization reported at the Property has good continuity along strike, and that he has seen LHOS successfully applied to numerous mines with mineralization with a similar geometry. Mr. Vilela (QP) notes that compared to the 2012 FS, mineralization in the 2014 Model has an increased average dip of 50˚ which is favourable for the LHOS method.

LHOS is defined as a moderate production, non-entry, bulk mining method most applicable to large, regular mineralized bodies. Level intervals are nominally 15 – 25 m floor to floor. Production holes are drilled by longhole methods, from either the upper or lower development horizons. A slot raise is excavated first then the remaining block of mineralized material in the stope is blasted. Loading is done from the drive by a load-haul-dump machine (LHD). To ensure the safety of the LHD operator, once the brow of the stope has opened this operation is done remotely.

The mining sequence retreats from stopes at the extremities of a mineralized body back towards the access. This means each stope has an individual slot, and the stope is retreated to maximum safe distance, determined based on the rock quality index, where it is completed, and is generally paste filled from the level above. Once the fill has cured, the next stope in the sequence on that level can begin. To ensure continuity within the production profile, it is imperative that multiple levels are available for extraction. Paste filling removes the need to leave pillars and effectively allows for total extraction of the mineralized material.

With respect to the Ollachea Property, in areas where stope widths range from 2 to 18 meters, a single drive on the top and bottom of the stope would provide sufficient extraction capability. Stopes with widths from 18 to 34 meters would require two drives at the top and bottom. Stopes with widths greater than 34 meters require three drives at the top and bottom.

LHOS offers several approaches with regards to the location of capital infrastructure and the mining sequence. Ramp infrastructure can be located such that single access to the mineralized material is provided only at the end of the strike or it may be located such that access to the mineralized material is positioned at a midpoint along strike.

A schematic of LHOS has been provided in Figure 16-1.

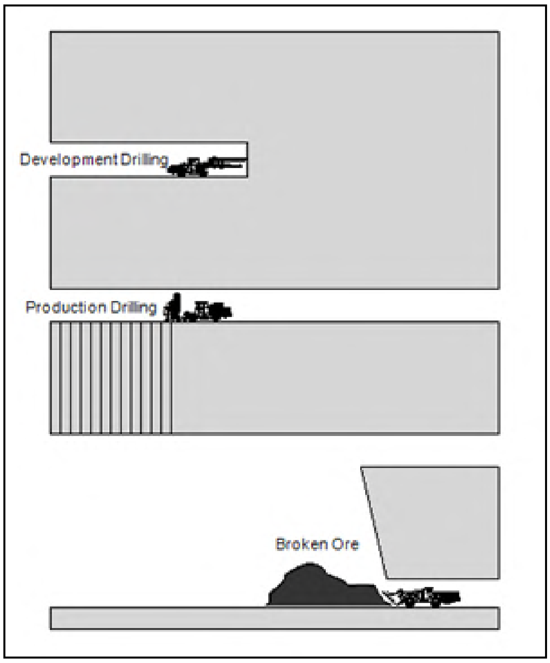

Figure 16-1: Schematic - Long Hole Open Stoping Mining Method

16.3 Hydrological Parameters

Eight hydrological drill holes were completed in 2011, ranging between 20 and 150 m deep. Hydrological drill holes were located in the Oscco Cachi valley and above Minapampa.

Hydrological parameters for the 2012 FS were based on the 2011 hydrological drill program. Hydrological investigation has not advanced since 2012. Mr. Vilela (QP) notes that inflows of water reported from the 1,234 m Exploration Tunnel (completed in 2013) were approximately 10 l/s (litres/second), significantly less than anticipated in the 2012 FS.

Based on reported inflows of water from the Exploration Tunnel, Mr. Vilela (QP) considers that inflows of water in the Ollachea Mine will be manageable with standard pumping infrastructure. Notwithstanding this, Mr. Vilela (QP) recommends that hydrological investigations are updated based on the exploration tunnel results.

16.4 Geotechnical Parameters

Based on an updated geological model and structural interpretation developed by MKK (2014 Model), Mining Plus was commissioned to re-evaluate the geotechnical parameters, specifically stable stoping spans and ground support requirements.

Incorporating new data from the Exploration Tunnel, stable stoping spans and ground support requirements were re-evaluated by Mining Plus in June 2014 in the report titled “Ollachea underground Mining Study” (MP 2014). Findings have been summarized below.

16.4.1 Rock Mass Characterization

Mr. Vilela (QP) considers that the geotechnical parameters applied at this stage of the study are representative of the average rock mass conditions and appropriate for the level of study.

The Q-system (rock mass quality index) according to Barton and Grimstad, 1993 was utilised to assess the rock mass quality of mineralization and host rock and to give an estimate of stope/development stability and support requirements. The Q-system quantitatively describes three aspects of rock mass based on six geotechnical parameters, to arrive at a Q value:

Rock block size defined based on the following geotechnical parameters:

RQD - the Rock Quality Designation as described by Deere, 1967

Jn - the joint set number count

Joint shear strength defined based on the following geotechnical parameters:

Jr - the joint roughness factor
Ja - the joint alteration factor

Confining stress defined based on the following geotechnical parameters:

Jw - the joint water reduction factor
SRF - the stress reduction factor

According to Barton et al, (1974) (Equation 1) Q values can range from 0.001 (very poor rock conditions), to 1000 (excellent rock conditions). MP 2014 report provided a detailed assessment of Q values based on available records.

$$Q = \left(\frac{RQD}{Jn}\right) * \left(\frac{Jr}{Ja}\right) * \left(\frac{Jw}{SRF}\right)$$

Equation 1: Q-system Rock Mass Quality Index

Mining Plus (MP 2014) used Q values to determine the ground support requirements for the Ollachea Mine Plan (Table 16-1).

Mining Plus (MP 2014) used Q’ values to determine stope design parameters where Jw and SRF are set to 1.0 (Table 16-1).

Table 16-1: Average Representative Q & Q' values for Rock Mass

Average Q'	Average Q
5.1	1.69

Considering calculated Q values (MP 2014), the rock mass conditions in the mineralized zones and the immediate hanging wall are amenable to the LHOS with paste fill.

Mr. Vilela (QP) notes that there are likely to be zones of increased fracturing and weathering around faults/shear zones etc., that could result in a local reduction in rock mass where increased ground support may be required. Similarly, there will be zones with better-than-expected geotechnical conditions where ground support requirements could be reduced. Both these cases need to be managed on a case-by-case basis as more data becomes available. This data will become available when the operation starts, at which point the stope and development support design can be optimized.

16.4.2 Ground Support

Ground support requirements have been assessed based on the Norwegian Geotechnical Institute rock quality index, "Q". This is an internationally accepted empirical method used to assess support requirements. The Q-System rock reinforcement design chart (Grimstad et al, 1993) relates the rock quality, extraction span and life of support requirements, and the method has been used for access development.

According to the rock mass quality data, the 4 m x 4 m drives could effectively be unsupported. In order to increase the safety in these drives, and to manage unexpected zones of potentially poorer rock mass conditions (i.e., reduced RQD, unfavourable jointing), it is recommended that all drives are supported with systematic bolting and mesh as a minimum requirement.

The meshing and bolting recommendation acknowledges that at this stage of the study we do not have enough detailed information to understand all the areas that may contain below-average rock mass conditions.

Similarly, the design recommendation of Mesh (Floor to Floor) with 3 m long rock bolts on 1.5 m x 1.5 m spacing for the ramp and stockpiles is also designed based on the expected ground conditions, but must be revised and refined on a case-by-case basis as the development advances.

For the Crib Room, Magazine and Workshop the recommendation is to utilise shotcrete with fibre and bolts as a minimum, as the empirical design recommends.

16.4.3 Stope Sizes

Based on the review of available geotechnical information, the rock mass conditions appear to be generally favourable, with relatively high RQD numbers, low inflows, and largely unaltered rock. An assessment on the stopes has been undertaken using the Modified Stability Number (N') proposed by Potvin (1988) and based initially on Q', where:

N' = Q' x A x B x C

Where;

- Q' is the modified Q Tunnelling Quality Index (after Barton et al 1974)
- A is the rock stress factor
- B is the joint orientation factor
- C is the gravity adjustment factor

To infer ground support requirements for the stopes proposed in the Ollachea Mine Plan, the modified stability number (N') and Tunnelling Quality Index (Q') were plotted on the stability graph according to Potvin, 1988. The graph can be seen in Figure 16-2.

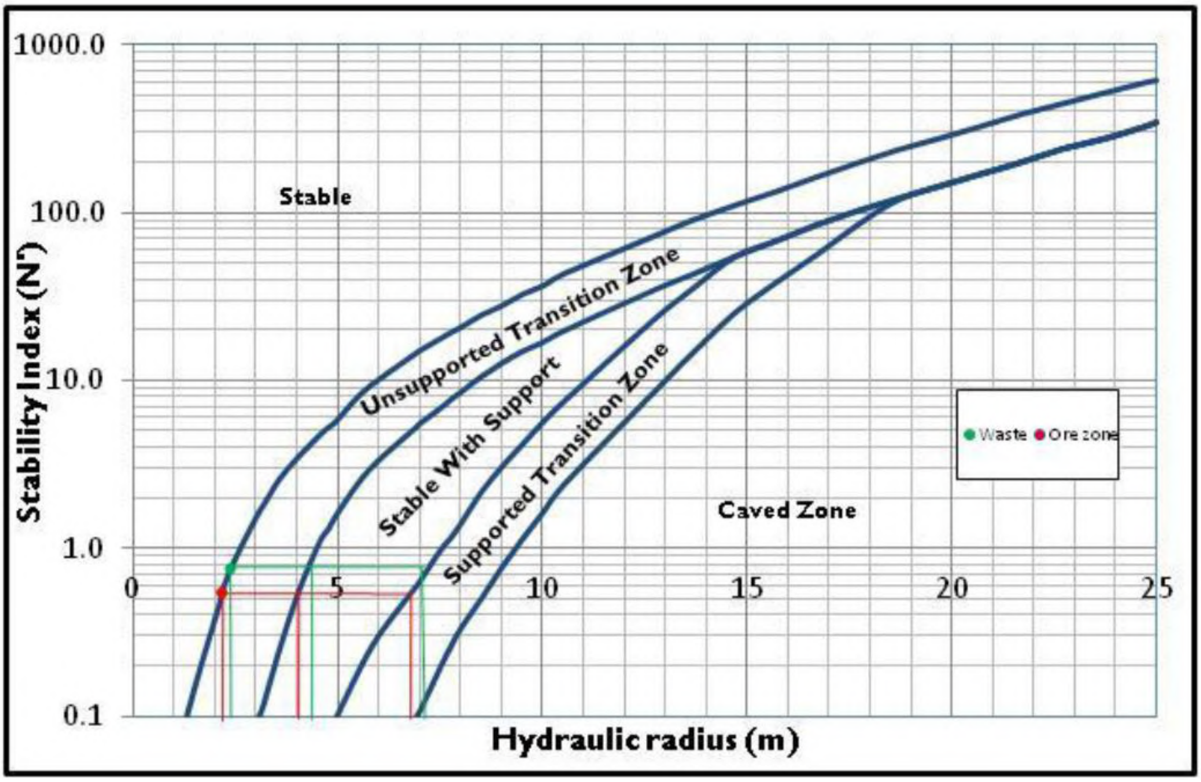

Figure 16-2: Stability Graph Method

The Stability Graph Method is widely used in underground hard rock mines as a basis for open stope support design and is frequently used in the mine planning phase as a tool to assess the viability of stope geometries and to determine maximum permissible spans. The stability graph method uses a concept called the "hydraulic radius" (HR), which is determined by dividing the exposed wall area by the wall perimeter. The stability graph method defines a maximum HR depending on the rock mass conditions and the support applied.

Based in the analysis of N' and Q' values, the stopes proposed in the Ollachea Mine Plan are expected to be stable. The following recommendations have been made:

- 15 m sublevels
- For a stope width of 18 m, use a maximum strike length of 25 m with cable bolt support
- For a width of 18 m – 34 m, use a maximum strike length of 20 m with cable bolt support
- For a width of >34 m, use a maximum strike length of 15 m with cable bolt support.

The Ollachea Mine Plan requires cable bolts in order to support the hanging wall and backs, based on the hydraulic radius and stope stability number N' in order to achieve the desired stope dimensions to support the desired productivity.

The recommended bolt spacing in stopes is 2.08 x 2.08 m for walls and 2.00 m x 2.00 m for the backs (MP 2014).

16.5 Cut-off grade

In 2017, Mining Plus was commissioned to perform an Optimization Study titled "Ollachea Mining Optimization Study 1500 to 3000 Tonnes Per Day May 2017" (MP 2017).

The cut-off grade philosophy used for the Ollachea Mine Plan has been based on the MP 2017 study which is discussed below.

Initial high-grade optimization runs were completed on the 2014 Mineral Resource Estimate ("MRE") block model to determine how the orebody reacts to higher cut-off grades. An initial MSO run was completed at a cut-off grade of 3.0 g/t Au (Figure 16-3).

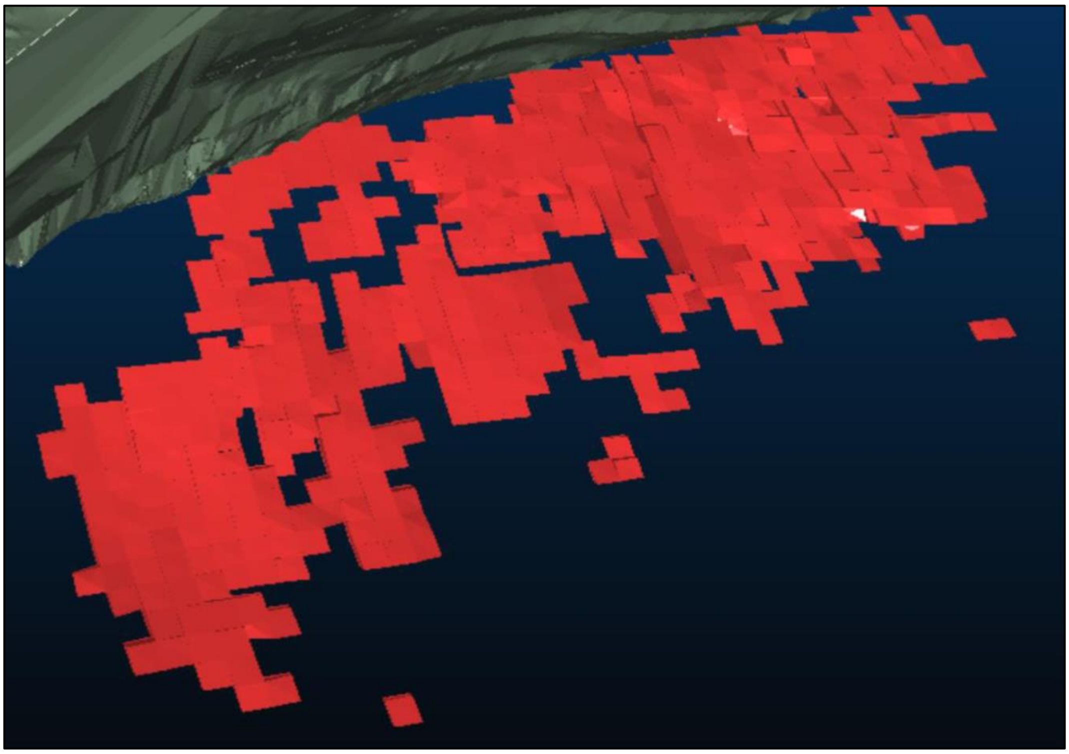

Figure 16-3: Perspective view southwest of 3.0 g/t Au MSO stope shapes

This same exercise was repeated for cut-off grades of 3.5 and 4.0 g/t Au as seen in Figure 16-4 and Figure 16-5.

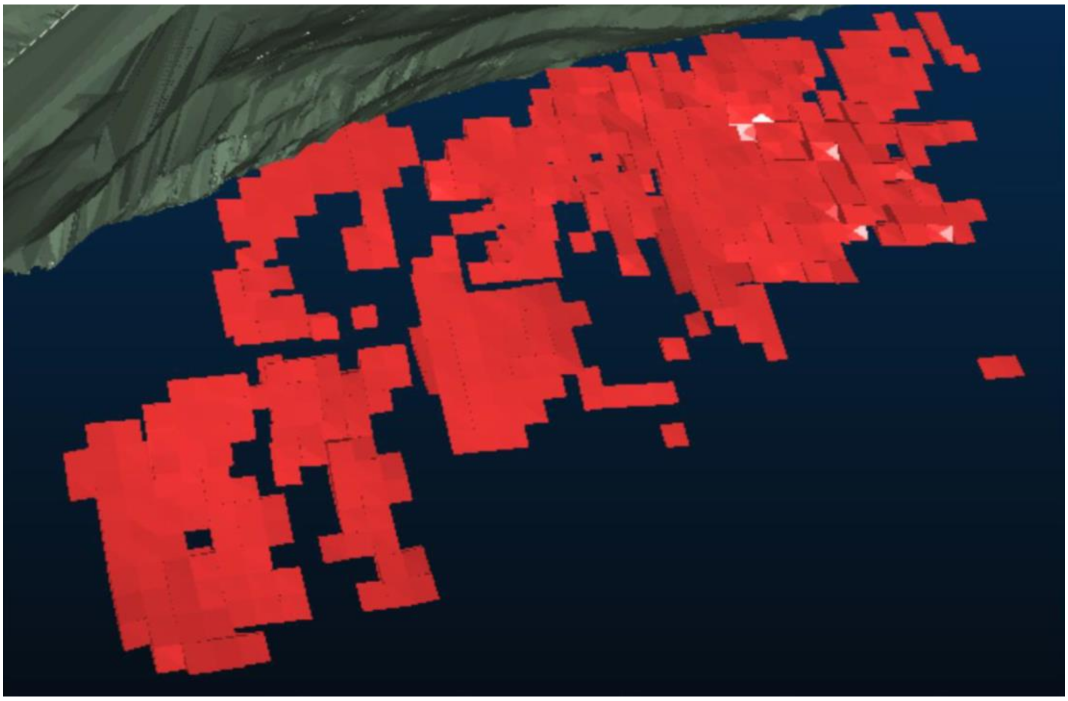

Figure 16-4: Perspective view southwest of 3.5 g/t Au MSO stope shapes

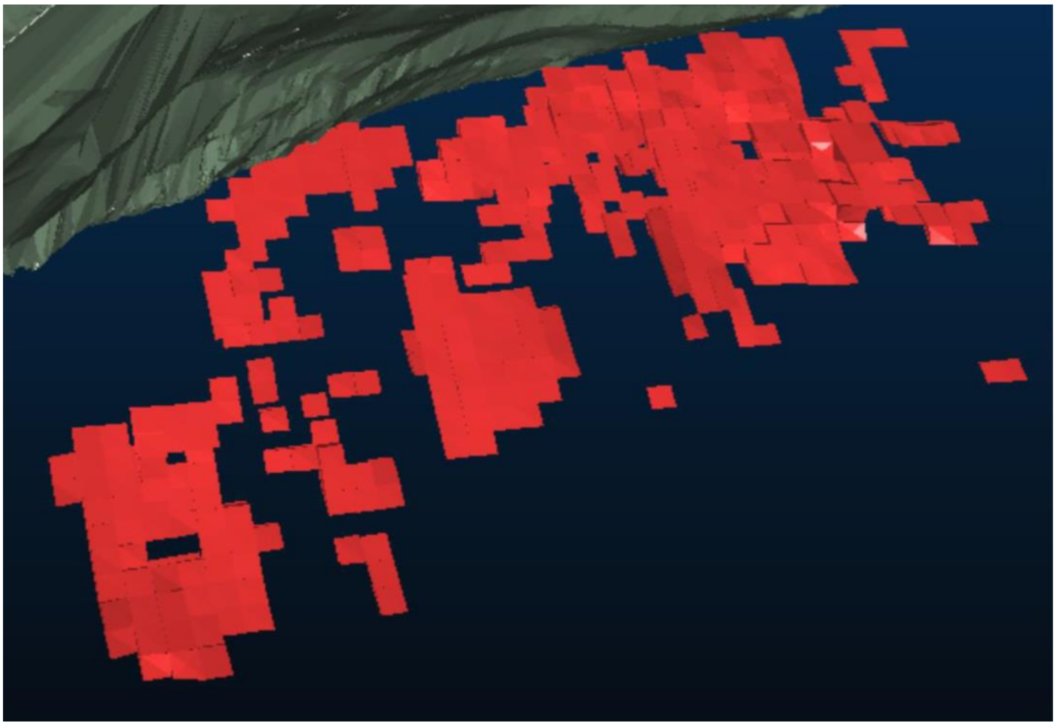

Figure 16-5: Perspective view southwest of 4.0 g/t Au MSO stope shapes

Often, when higher grade cut-offs are applied to mineralized material, it can break up with many isolated stopes some distance apart from each other. When this is the case, the high-grade option requires a similar amount of development as required for a low-grade cut-off option to access fewer tonnes, which largely negates any benefits of the high-grade option for the deposit. At the same time, a portion of the lower-grade material is sterilised after mining the higher-grade material (the low-grade material is no longer accessible).

The initial high-grade MSO runs demonstrated the effect of the use of a higher cut-off grade early in the mine life. The benefits of the higher cut-off grade/targeting only higher grade early include:

- More revenue per tonne mined
- The opportunity for reduced development cost early in the mine life (mine less tonnes for the same revenue)

The downside of increasing the cut-off grade is a reduction in the overall ounces in the reserve (some lower-grade material will not be mined).

Based on the cut-off grade work, Mining Plus proposed the following:

- The high-grade areas should be targeted early in the mine life to increase the head grade and allow for a more gradual ramp up to steady state production, with the lower head grade stopes mined later in the mine life
- The high-grade areas should be used to produce an acceptable metal profile for the schedule whilst mining less tonnes overall during the mine start-up phase.

In the case of the Ollachea Deposit, the higher-grade material can be efficiently mined in the highest grade and thickest part of the mineralized material and the lower-grade material need not be sterilized to a large degree. The MSO stope shapes do begin to break up significantly and would not be efficient to access at a cut-off grade of 4 g/t. In order to balance the compromise between cut-off grade, mining efficiency, and ounces produced, a 3.0 g/t cut-off grade was selected in the initial years of the mine life, then reverting to 2.1 g/t for the remainder of the mine life. Additional stopes at an incremental cut-off grade of 1.4 g/t were also added where no additional development was required to mine them.

This cut-off grade philosophy allows for an increase to the mined grade early in the mine life, and a lowering of the cut-off grade later in the mine life as those areas of the mineralized material are better suited to a lower cut-off grade.

Figure 16-6 shows the location of the > 3.0 g/t Au cut-off grade stopes in relation to the rest of the mine design and the exploration tunnel.

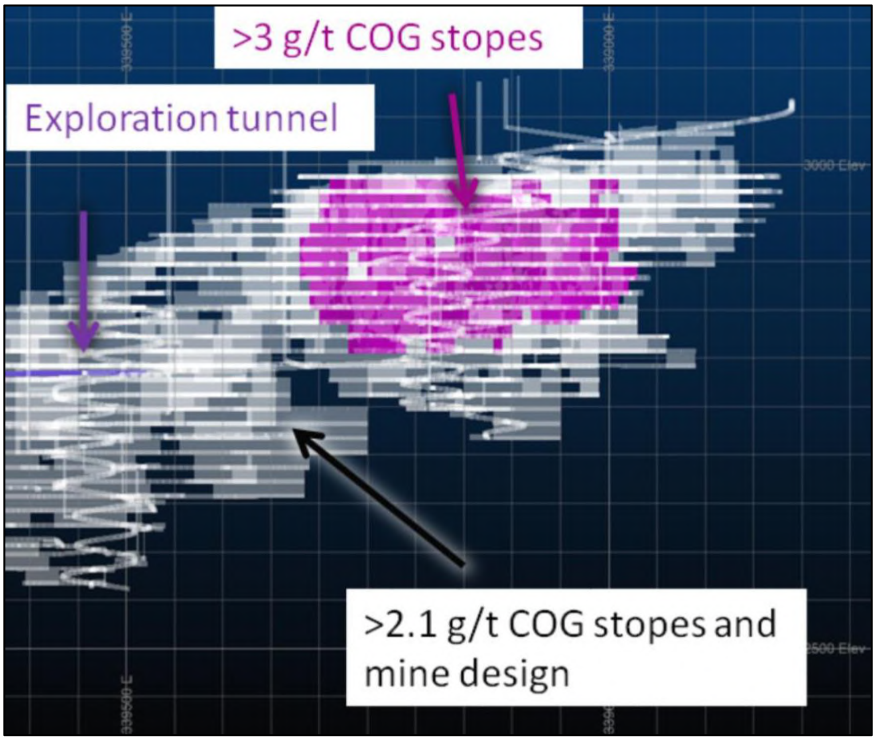

Figure 16-6: >3 g/t COG stopes and > 2.1 g/t COG stopes

16.6 Dilution Estimates

Mining dilution estimates for the stopes were assessed in the MP 2014 study.

Various factors were taken into consideration when defining the stope dilution estimates for the Ollachea Mine Plan, such as:

- External dilution into the footwall, hanging wall, stope back and adjacent filled stopes (inclusive of dilution due to fill in the floor)
- Quantified in terms of overbreak distance for each of the stope width categories considered
- Differential overbreak distances assumed for footwall and hanging wall accounting for likely performance of these zones
- Undercutting of higher-strength paste-fill sill pillars in the mine plan
- Consideration for production drillhole deviation and blast damage, given strong foliation present within the Ollachea host rock
- Time-dependent footwall and hanging wall failure as a function of stope volume (related to the time the stope remains open during production cycle)
- Long anchor ground support (cable bolts) planned to be installed into hanging wall, thereby minimising hanging wall dilution
- Secondary dilution from the paste fill
- Experience and case studies from other mines with similar geometries.

Overbreak parameters in the footwall, hanging wall and from adjacent filled stope (adjacent stope) have been estimated based on experience with similar geometry.

For all continuing work, Mining Plus concluded that a dilution factor of 17.5% be applied when determining actual stope tonnages (MP 2014).

Mr. Vilela (QP) has reviewed the MP 2014 report and is satisfied that estimated dilution estimates are appropriate for the Ollachea Mine Plan.

It should be noted that the mineralized material is effectively non-visual, so sampling procedure and grade control will be very important.

16.7 Mining Recovery Estimates

Mining recovery estimates for the stopes were assessed in the MP 2014 study.

Mining recovery estimates for the Ollachea Mine Plan are based on:

- LHOS mining method.
- Mineral losses resulting from underbreak.
- Blasted mineralized material which cannot be sufficiently mucked from a stope.
- Mineralized material with fill in the floor.

Unrecovered crowns have been considered on a stope-by-stope basis when calculating overall stope recovery. Un-mucked mineralized mineral and the footwall and hanging wall underbreak from the in-situ design stope tonnage, have been calculated based on stope width (inclusive of dilution). The overall stope mining recovery estimate is effectively the weighted average across all stope widths.

It was concluded that a mining recovery factor of 96.2% should be applied (MP 2014) when determining actual stope tonnages. Mining recovery for lateral development was estimated at 100%. Mr. Vilela (QP) has reviewed the MP 2014 report and is satisfied that estimated mining recoveries in stopes of 96.2%, and in lateral development of 100% is appropriate for the Ollachea Mine Plan.

16.8 Underground Design Criteria

The Ollachea Mine Plan largely maintains the design parameters used in the 2012 FS. A comparison of cross-sectional shapes for development considered in the 2012 FS and Ollachea Mine Plan are shown in Table 16-2.

Table 16-2 - Sectional shapes

CODE	Description	Sectional Shape – Updated Design (m)
DEC	Ramp	5.5 x 5.5
ACC	Access	5.0 x 5.0
XC	Crosscut	5.0 x 5.0
OD	Ore Drive	5.0 x 5.0
SP	Stockpile	5.0 x 6.0
SMP	Sump	5.0 x 5.0
RAD	Return Air Drive	5.0 x 5.0
FAD	Fresh Air Drive	5.0 x 5.0
EWD	Escapeway Drive	5.0 x 5.0
RAR	Return Air Raise	Ø 4.5
FAR	Fresh Air Raise	Ø 4.5
EWR	Escapeway Raise	Ø 1.5
CUD	Cuddy	4.0 x 4.0
EXP	Explor Drive	5.0 x 5.0
WKS	Workshop	8.0 x 5.3
MAG	Magazine	8.0 x 5.3
CRB	Crib Room	5.0 x 5.3
FAR	Fresh Air Raise	Ø 3.0
RAR	Return Air Raise	Ø 3.0

Mr. Vilela (QP) notes that further work can be completed to optimize the development of cross-sectional shapes to better suit equipment requirements.

Access to the mine will be via two portals (Figure 16-7). Development has already commenced from the lower portal, with the exploration ramp. The updated design continues from the point at which the exploration ramp stops. The upper portal will be in broadly the same location as described in the 2012 FS and Mr. Vilela (QP) considers the design associated with the upper portal construction to be relevant to the updated design, but needs to be re-assessed based on the current status of the artisanal workings.

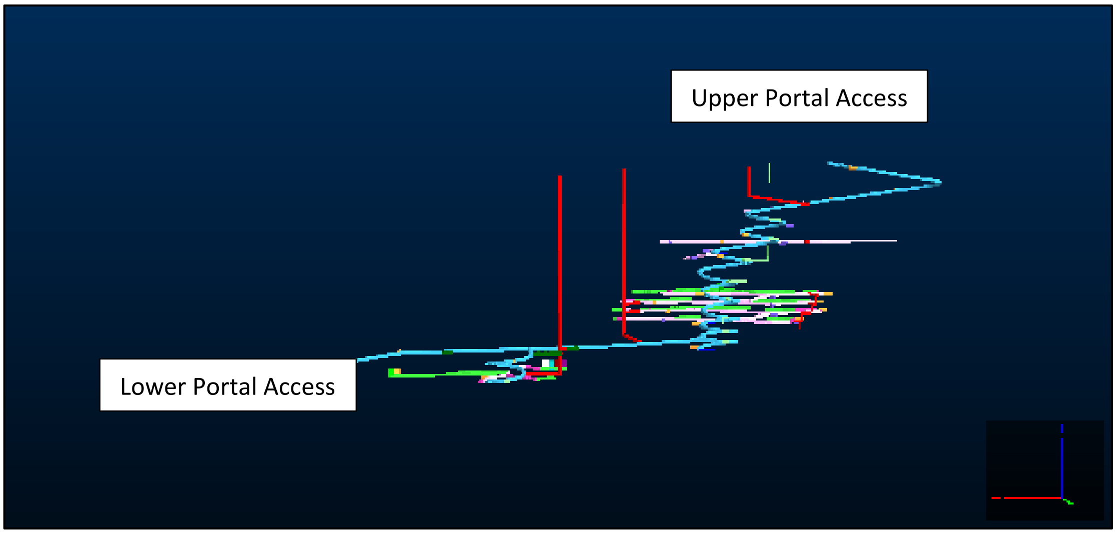

Figure 16-7: Mine Access Portals

Mr. Vilela (QP) notes that there is significant artisanal mining activity around the upper portal, and he recommends that location should be reassessed and modified based on the location of the artisanal mines.

Other design criteria are:

- Maximum grade for ramp is 1 in 7
- Minimum turning radius is 30 m in main ramp
- Minimum turning radius is 25 m in auxiliary ramp.

16.9 Mine Design and Sequencing

Stopes will be accessed longitudinally (along strike) on each level by, one, two or three strike drives, dependent on lode thickness.

The direction of mining for the deposit will be from the bottom up. In general, as each mining level is completed, the next level will start using the backfilled stope voids as the mining platform.

To achieve a reasonable mining rate, the mine will be split into multiple mining panels that can be mined simultaneously (as is common practice). The lowest level of each of these mining panels requires an artificial sill pillar to be created using high strength paste fill. This high-strength sill pillar allows mineralization located directly beneath it to be completely extracted. This artificial pillar is to contain a higher cement content than ordinary paste fill to be applied to the remaining stopes in the mine plan, in accordance with the geotechnical and backfill assessments considered in the 2012 FS. The 2012 FS determined that the maximum safe stope span to be opened beneath the high-strength paste-fill sill pillar is 10 meters. This dimension has been honoured in the Ollachea Mine Plan.

Mine development has been sequenced to minimise the upfront development costs, whilst maximising the ounces produced and targeting (as best as practicable) full utilisation of the plant design capacity.

Examples of the mine development can be seen in the following figures (Figure 16-8 to Figure 16-10).

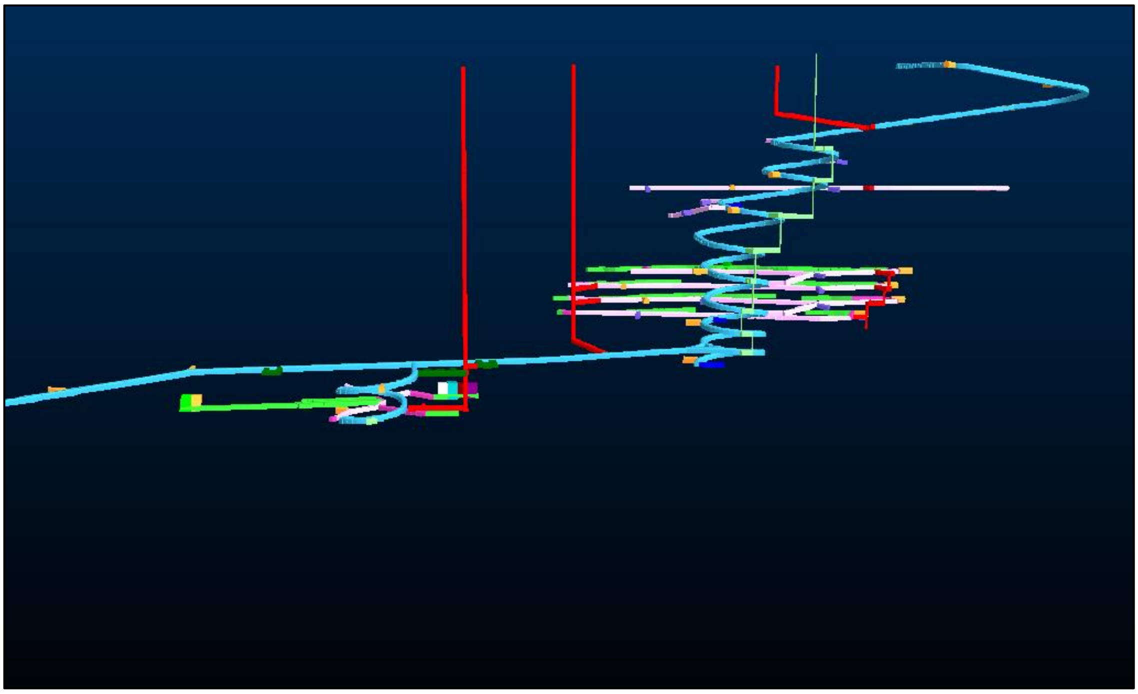

Figure 16-8: Mine development end of year 1 (Looking South from the hanging wall)

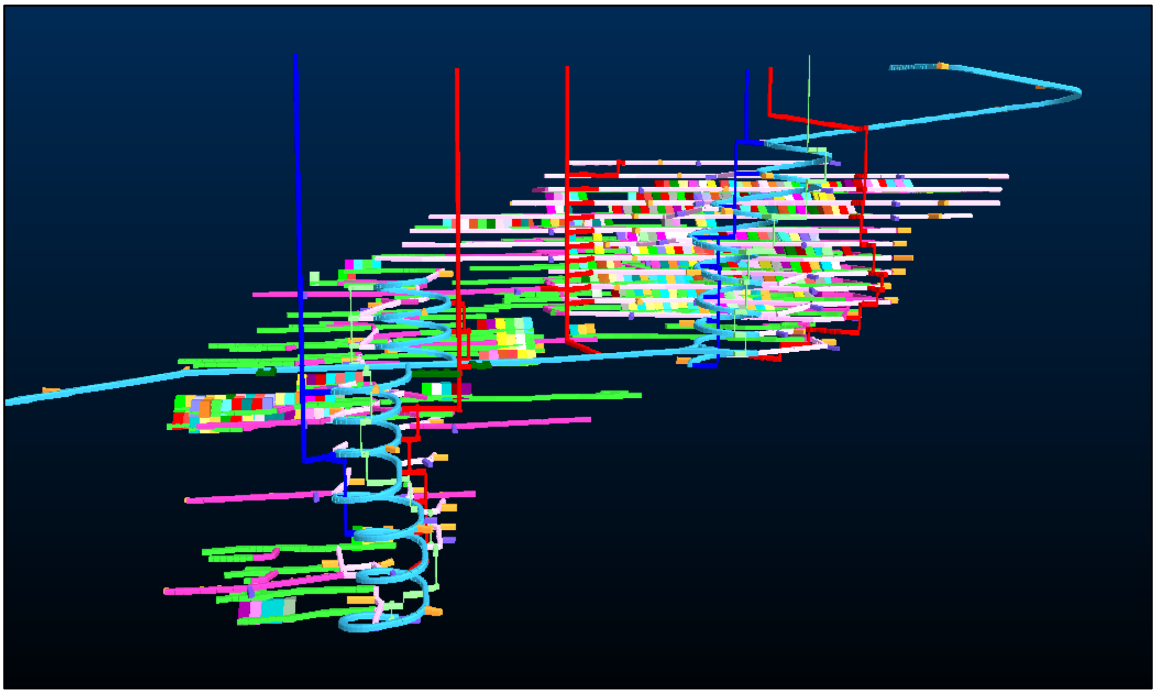

Figure 16-9: Mine development end of year 6 (Looking South from the hanging wall)

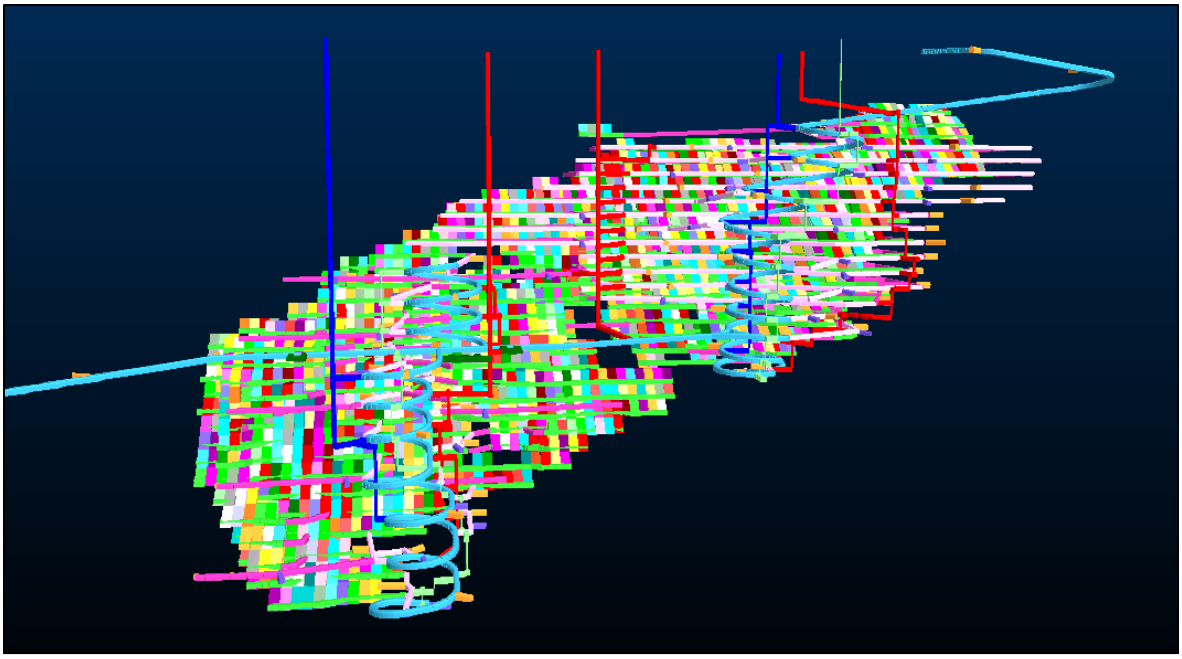

Figure 16-10: Mine development end of year 11 (Looking South from the hanging wall)

Note: Development on the upper right-hand side on the image is designed for grade control drilling for improved definition of the stopes prior to mining.

Figure 16-11 shows the stacked nature of the mineralized material looking south along strike of the mine design.

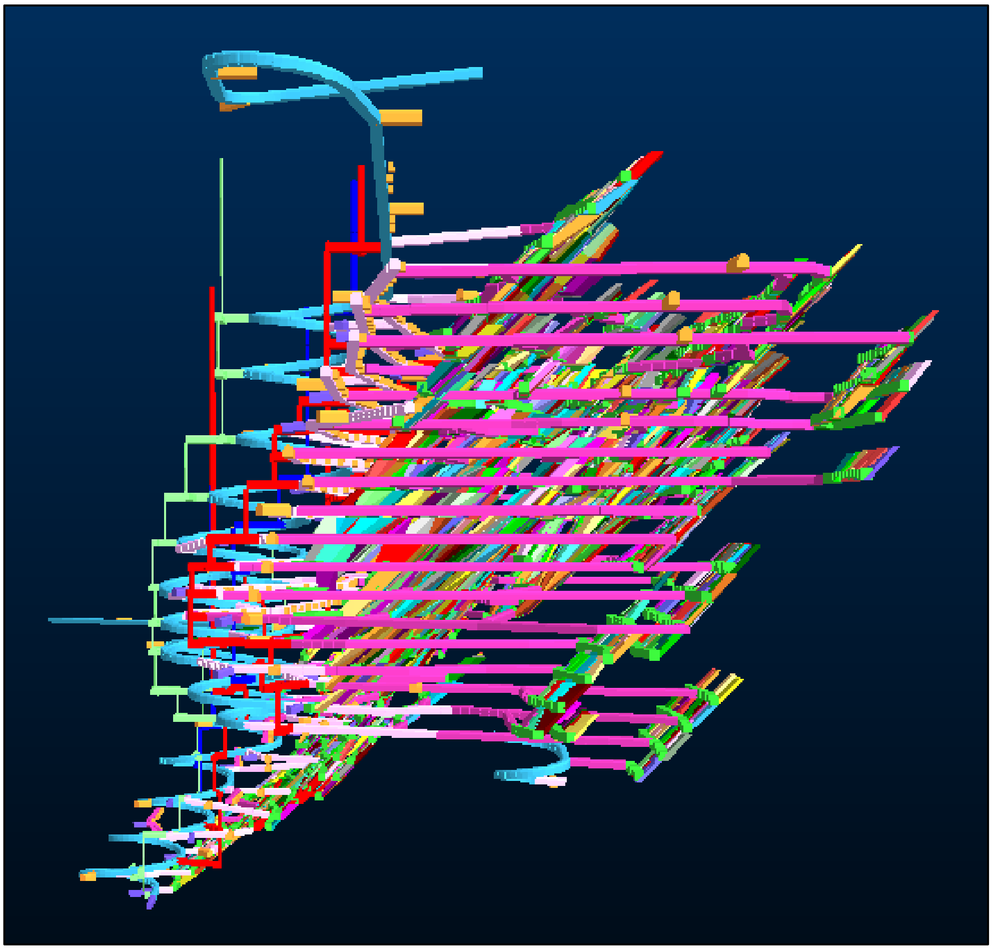

Figure 16-11: Looking south along strike of the mine design

16.10 Ventilation

16.10.1 Primary Ventilation

The primary ventilation design consists of:

- One surface intake shaft
- Three surface return shafts with primary fan duties of 100 m^3/s, 250 m^3/s and 350 m^3/s
- Two surface intake ramps
- Internal return and intake airway shafts.

The 2012 FS stated that a peak flow of 700 m^3/s at an air density of 0.8 kg/m^3 is required; this airflow is the equivalent of 470 m^3/s at 1.2 kg/m^3.

Mr. Vilela (QP) notes that mine development plans presented in the 2012 FS mine plan and the Ollachea Mine Plan have not significantly changed. Mr. Vilela (QP) considers that the 2012 FS primary ventilation parameters are appropriate for the Ollachea Mine Plan. Figure 16-12 shows the primary ventilation design. For the first stage of the mine, the fresh air will come from the upper portal ramp (the ramp on the right of Figure 16-12). Figure 16-12 also shows the quantities of airflow through the different return air raises ("RAR"), and also shows the Fresh Air Raise ("FAR").

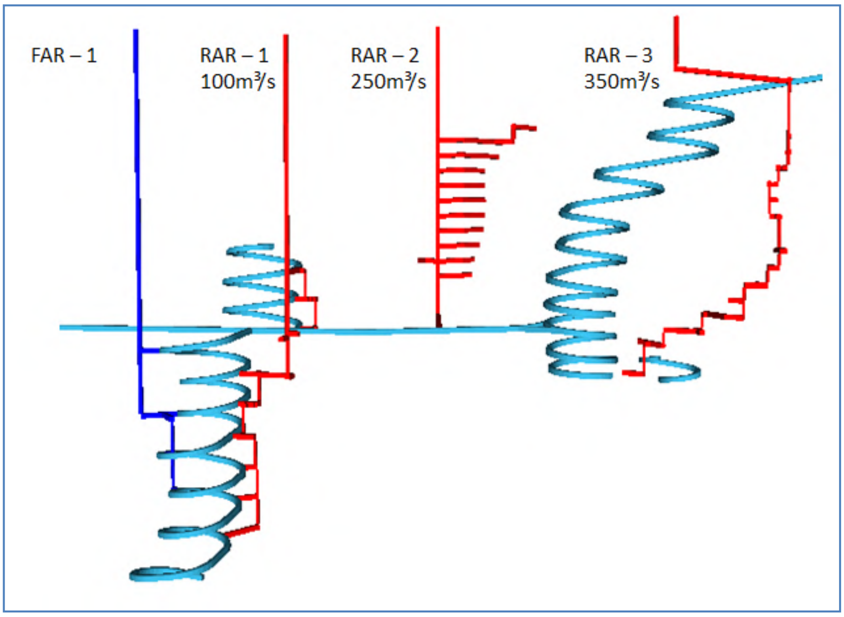

Figure 16-12: Primary ventilation design

Ventilation raises considered in the Ollachea Mine Plan have the following diameters:

- 4.5 m – Surface FAR and RAR
- 3 m – Internal FAR and RAR.

During the first two years of production, RAR 2 will be required to be installed as the primary production ventilation raise. All production in the central part of the Ollachea Mine Plan is contingent to this raise being completed prior to production commencing in these areas. To provide primary ventilation return for the lower ramp, magazine, and workshop, RAR 1 will be installed early in the mine life.

16.10.2 Secondary Ventilation

The Ollachea Mine Plan relies on secondary fans located along the access drives prior to the installation of RAR's.

16.11 Paste Backfilling

The 2012 FS states that the LHOS mining method and extraction sequence adopted is reliant on the use of paste fill. Paste back fill has been considered in the Ollachea Mine Plan.

Process plant total tailings will be used to produce the paste fill. Approximately 43% of the process plant tailings will be used as paste fill. Waste rock will be used as a floor cap to paste-filled stopes, for loading and tramming requirements.

The lowest level of each mining panel requires an artificial sill pillar to be created using high strength paste backfill to allow the mineralization located directly below to be completely extracted.

The paste is located in the vicinity of the upper portal and the paste will be delivered by way of a purpose-drilled borehole, then distributed throughout the mine. The paste plant is further discussed in Section 17 of this report.

16.12 Stope Control Drilling

Stope control drilling is incorporated and costed in the mine plan to drill out the mineralized areas prior to their development. The purpose of this stope control drilling is to better define the mineralization from underground and adjust the stop design based on the increased drilling density.

16.13 Production Schedule and LOM Development

The Ollachea Mine Plan and production schedule is based on subset of the mineral resources and considers an 11-year life of mine ("LOM"). Production during years 1 to 3 will be at 1500 tpd before expanding to 3000 tpd from year 4 to 11.

The production schedule consists of 95.9% indicated material and 4.1% inferred material.

The production schedule contains material from Minapampa only. The material in the Minapampa Far East has not been considered in the current mine plan. Minapampa Far East offers an opportunity to extend the mine life beyond the eleven years currently presented and should be investigated in further studies.

The cut-off grade applied whilst operating at 1500 tpd is 3 g/t. The cut of grade then reduces to 2.1 g/t for the remainder of the current mine plan.

The mineralized material will be mined from two sources: the stopes and the mineralized material development drives. As the name suggests, the mineralized material development drives will predominantly be constructed in mineralized material. That material will be sent to the plant for processing. In the early years of the mine life, mineralized material development makes up a significant percentage of the total processed material as the mine is being "opened up" ready for stope development. All non-essential development has been pushed as far back as possible in the schedule to reduce the cash expenditure in the early years of the LOM.

The production schedule developed based on physicals of mine development anticipated for the Ollachea Mine Plan is presented in Table 16-3. The production profile of tonnes versus mined grade is shown in Figure 16-13.

Table 16-3: Proposed Mine Schedule

	Units	Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Total
Mine Development														
Vertical Development	Meters	668	1,208	107	296	-	56	-	-	-	-	-	-	**2,335**
Horizontal Development	Meters	5,298	18,846	3,264	7,613	13,666	5,467	1,723	11,588	996	-	-	-	**68,461**
Mine Production														
Mineralized Material From Development	Tonnes	11,989	476,219	37,794	32,401	676,232	192,661	87,660	378,419		-	-	-	**1,893,375**
Mineralized Material From Stopes	Tonnes	-	4,318	504,800	507,230	145,022	895,569	997,256	703,661	1,045,761	1,101,030	1,084,974	764,170	**7,753,791**
Mineralized Material To/From Stockpile	Tonnes	-11,989	- 30,617										42,606	
Total Mineralized Material Processed	**Tonnes**		**449,920**	**542,594**	**539,631**	**821,254**	**1,088,230**	**1,084,916**	**1,082,080**	**1,045,761**	**1,101,030**	**1,084,974**	**806,776**	**9,647,166**
Gold Grade	g/t		4.18	4.32	4.68	2.99	3.67	2.99	2.80	2.83	2.71	3.01	3.09	**3.23x**
Contained Metal	Ounces		60,483	75,334	81,311	78,909	128,411	104,167	97,421	98,687	94,518	105,027	79,689	**1,003,957**
Mining Rate tpd	tpd		1,250	1,507	1,499	2,281	3,023	3,014	3,006	2,905	3,058	3,014	2,241	
Waste Tonnes	Tonnes	410,028	850,408	191,088	494,782	282,219	194,727	35,760	423,781	32,280	18,211	0	5,013	**2,938,297**
Paste Fill Volume Required	m^3	-	1,574	217,169	214,159	63,188	386,192	433,215	298,859	445,765	467,815	480,183	332,009	**3,340,128**

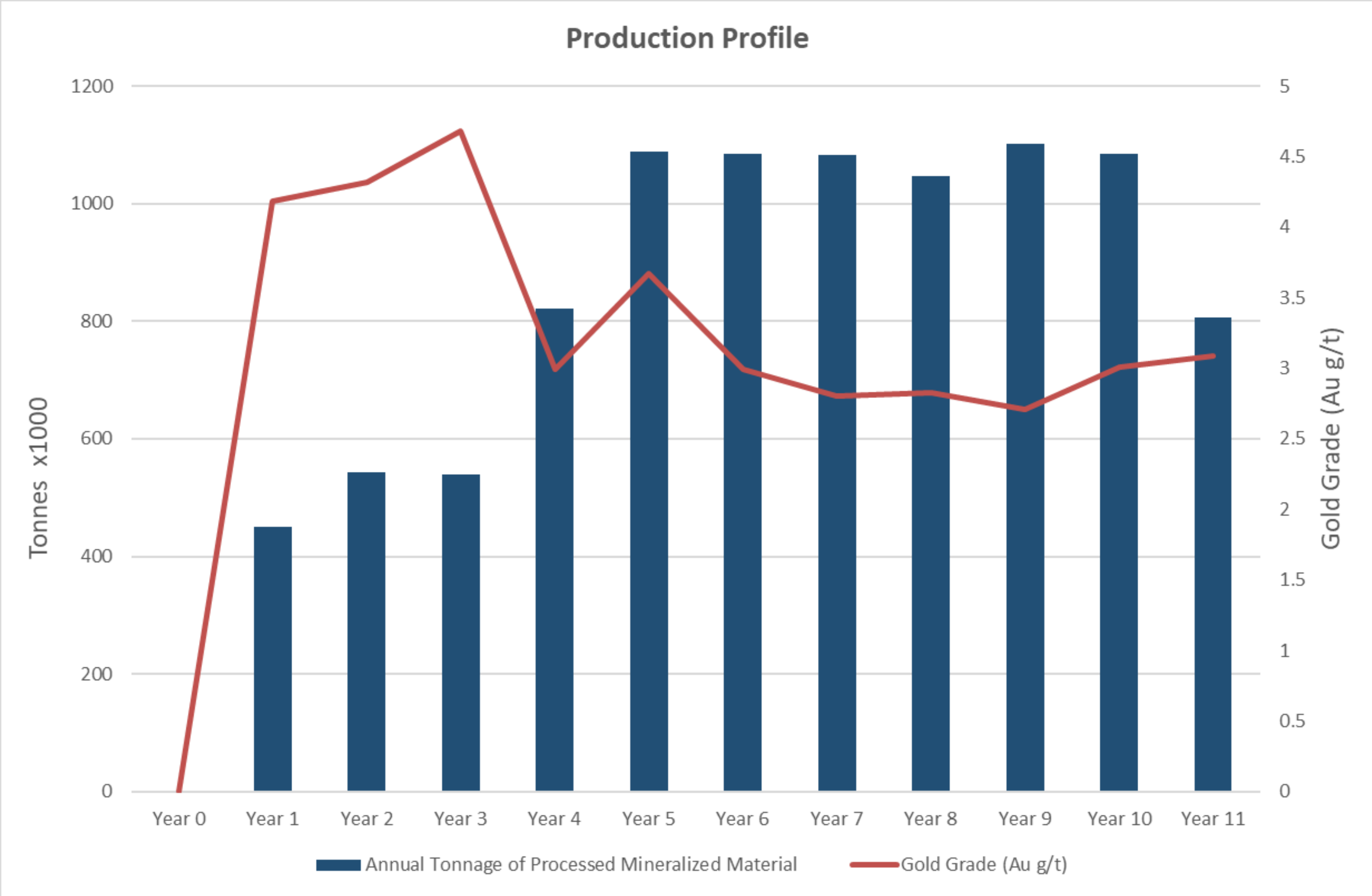

Figure 16-13: The production profile of tonnes versus mined grade

16.14 Workforce

It is envisaged that the mining operations will be carried out by a contractor. The contractor will be responsible for the development of the access drives, as well as the preparation and exploitation of the stopes. The owner's team will be responsible for managing the mining contractor and will take on the technical roles.

Under this scenario, the owner's team proposed workforce can be seen in Table 16-4, and the contractor's proposed workforce can be seen in Table 16-4.

Table 16-4: Owner's team proposed workforce

Department	Position	Count
Mine Management	Mine Manager	1
	Mine Superintendent	1
Geology	Geology Manager	1
	Chief Geologist	1
	Ore Control Geologist	3
	Assistant Geologist	2
	Ore Control	2
	Field Samplers (Drilling)	10
	Geotechnician	2
	Surveyor	1
	Assistant Surveyor	2
Projects	Head of Projects	1
	Assistant to the Head of Projects	1
	Field Supervisor	1
	Field Foreman	3
	Mason	2
	Laborer	6
Maintenance / HV	Maintenance Chief	1
	Maintenance Supervisor	2
	Welder	4
	Electrical HV Supervisor	2
	Electrician	3
	Instrument Technician	1
Total		**53**

Table 16-5: Contractors proposed workforce

Contractor’s workforce	Workers
Manager	2
Operations supervisors	9
Safety supervisors	5
Planning Supervisors	5
Maintenance Supervisors	26
Administration	7
Operators	81
Total	**135**

16.15 Equipment

The equipment fleet estimated for the mine was provided by a contractor and can be seen in Table 16-6 and Table 16-7.

Table 16-6: Contractors proposed workforce (Production fleet)

Production fleet	Equipment Number
Jumbo 12’	4
Scooptram 6 yd3	5
Scaler	2
Mining Trucks	8
Total production	**19**

Table 16-7: Contractors proposed workforce (Equipment Numbers)

Auxiliary equipment	Equipment Number
Pumps	4
Compressor	1
Electrical generator	1
Ventilator	3
Jackleg	3
Pick up	10
Total	**22**

The costs for the owners’ team have been built up and form part of the G&A. The costs of the contractor’s personnel and equipment has been costed and forms part of the Mining Operational Cost.

Mr. Vilela (QP) has revised the equipment and personnel numbers, and deems them appropriate for the mine operation proposed in this Technical Report.

16.16 Comments on Section 16

The conceptual mine plan considered in this PEA (the Ollachea Mine Plan) includes inferred mineral resources (4.1% of the material considered in the mine plan) that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability and there is no certainty that the PEA results will be realized.

The LOM schedules and forecasted revenues and costs are based upon forward-looking information. This forward-looking information includes forecasts with material uncertainty which could cause actual results to differ materially from those presented herein.

Instances of the word 'economic' in this section are intended to be conceptual only, and prospects for economic extraction have not been demonstrated.

In the vicinity of the proposed upper portal of the mine, there is significant artisanal mining activity. Prior to the start of operations, it is understood that the artisanal miners will leave the area to make it safe for formal mining to commence. At this point, all the artisanal workings should be surveyed, and the resource and mine plan adjusted based on the findings of that survey. This is further discussed in Section 20 of this report.

17 RECOVERY METHODS

John Thomas (QP) is responsible for the content of Section 17 of the Technical Report.

17.1 Staged Construction

The treatment plant will be developed in two stages.

Stage one will consist of a 1,500 tonnes per day ("tpd") treatment plant which will use gravity concentration to produce two concentrates, a high and lower grade gravity concentrate. The high-grade concentrate will be further upgraded to a smeltable grade using a shaking table, and the table reject and the other lower-grade concentrate will be leached in a carbon in leach (CIL) circuit. The estimated mass of these two products is 0.5%, and 15% of the feed mass respectively. Tailings will be filtered and used to make paste fill for the underground operation, with the excess being deposited in a dry stacked tailings management unit.

Stage 2 which is planned to come into operation during year 4 of operation, will double the treatment rate to 3,000 tpd. Some parts of the plant will have already been built to accommodate the full 3,000 tpd, other parts will need to be modified or duplicated.

17.2 Plant Layout

The plant will be located on three platforms as was planned in a previous study (2012 FS) and which has been permitted. The overall plant layout is shown in Figure 17-1. The mineralized material stockpile and crushing plant will be located on the upper platform, the mill and gravity concentration circuits will be located on the middle platform and the tailings filtration plant will be located on an extension to this platform. The leach and elution circuits will be located on the lower of the three platforms.

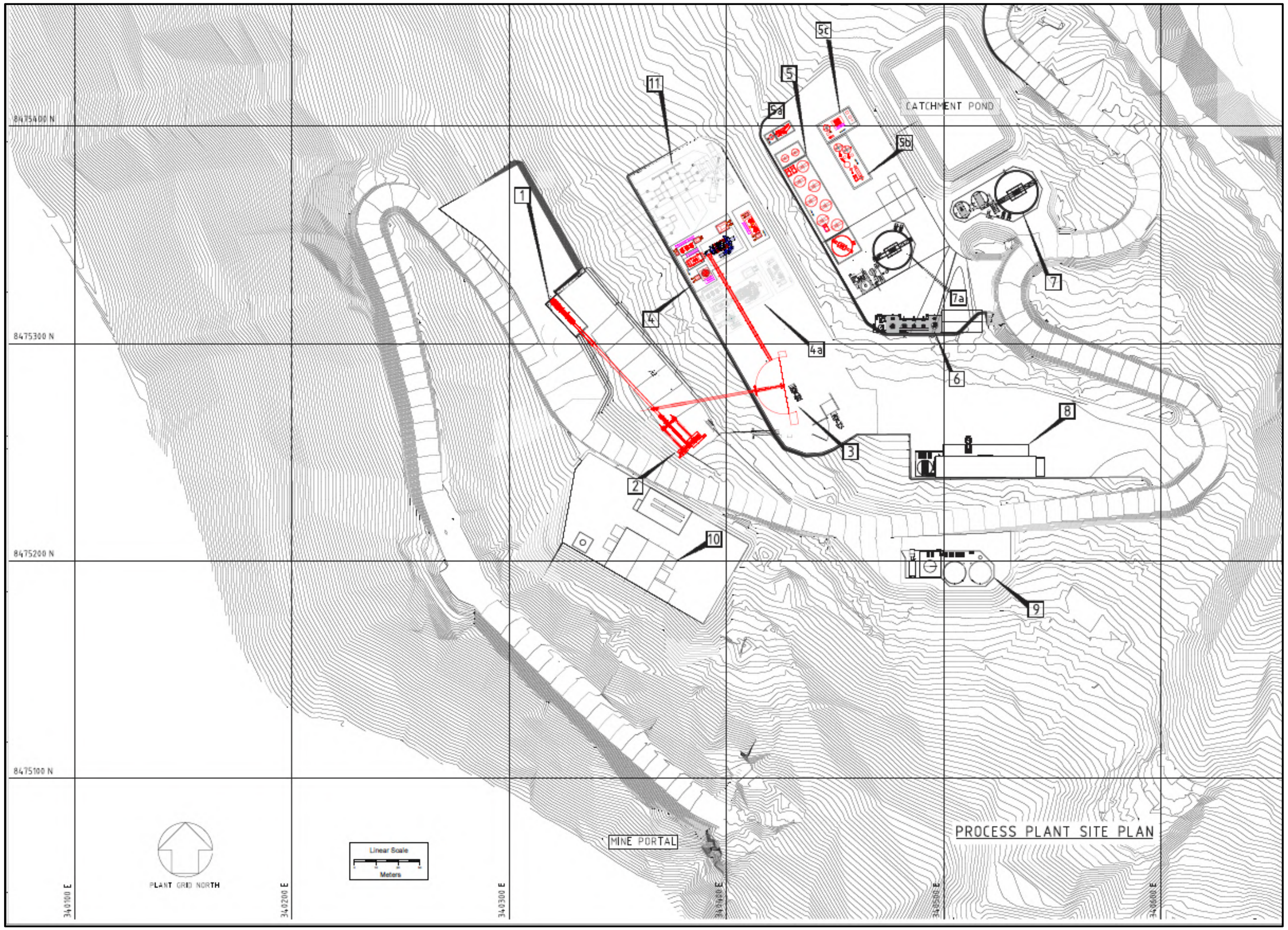

Figure 17-1: Process Plant Site Plan

In addition to the above, a water treatment plant will be installed alongside a lined catchment pond. All contact water, including water used in the process will be treated prior to discharge.

The flowsheets are based on test work carried out by Met-Solve / Sepro in 2017 and confirmed in 2021. The mineralized material is crushed in three stages to a P_{80} of 8 mm and then ground to a P_{80} of -75 microns using a ball mill operating in closed circuit. Semi-batch centrifugal gravity concentrators operating on cyclone underflow recover gold to a high-grade gold concentrate which will be further concentrated to a smeltable grade using a shaking table. This will be smelted directly to Doré bullion. The cyclone overflow is treated in continuous gravity concentrators and the tailings are filtered and either used for paste fill or stored in the tailings management facility. A concentrate of approximately 15% of the feed mass is produced and this concentrate will be added to the shaking table tailings and leached in a CIL circuit. The loaded carbon will be treated in a conventional Zadra elution circuit and the precipitate produced in the electrowinning cells will be smelted to doré metal.

17.3 Crushing

The crushing unit will be located on the upper platform together with the run-of-mine stockpile. For a processing rate of 1,500 tpd, the unit has been sized to process 150 tonnes per hour ("tph"), which will require 10 operating hours per day, leaving more than adequate time for maintenance etc. Mineralized material will be fed into the feed bin (10 m^3) using a wheel loader and a vibrating feeder fitted with a 75 mm grizzly will feed the oversize into a 610 x 914 mm Sepro jaw crusher. A conveyor (900 mm wide) will feed crushed mineralized material onto a 1.5 x 4.8 m vibrating screen fitted with 22 and 12.7 mm screen decks. The -12.7 mm material is the crusher product, the +12.7 – 22 mm material will be conveyed to a Sepro Blackhawk cone crusher (100 kW) and the +22 mm material to a FL Smidth XL350 Raptor cone crusher with a 220 kW drive. The product from both cone crushers is conveyed back to the screen. A stacker conveyor conveys the -12.7 mm material to the crushed mineralized material stockpile on the middle platform. The layout of the unit showing the location of the conveyors is shown in Figure 17-1.

The crushing unit will be supplied complete with conveyors, steel supports, electrical switch gear, electrical cable, and controls. The crusher and screening modules will be supplied skid mounted to simplify installation.

For the second stage increase to 3,000 tpd, only a larger screen and second Blackhawk cone crusher will be needed, with the nominal crushing rate increased to 250 tph, requiring 12 operating hours per day, again leaving adequate time for maintenance.

17.4 Grinding

Crushed mineralized material will be delivered to a silo which feeds into the mill or to a stockpile from which crushed mineralized material can be fed into the silo using a wheel loader. The mill feed belt will be fitted with a belt scale and the belt feeders will be controlled via a variable frequency drive ("VFD") to deliver the required feed into the mill. Water will be added in the mill feed chute to give a 65 – 68% solids concentration in the mill.

The 14' by 24' (4.26 m x 7.3 m) ball mill, fitted with a 2700 HP (2000 kW) motor is designed to process 1,500 tpd with a design availability of 92% with the product having a P_{80} of 75 microns. The mill will be equipped with rubber liners and will be supplied as a complete package with a lubrication system, instrumentation, cradles, jacks, and the motor starter.

The mill product will fall into a pump box and will be pumped to the cyclone cluster. The cyclone underflow will flow by gravity to a vibrating screen (2.1 m x 4.9 m) and gold will be recovered using centrifugal gold concentrators (see next section – Gravity Concentration). The concentrator tailings will be recycled to the mill discharge pump box and the screen oversize will be returned to the mill by gravity.

17.5 Gravity Concentration

Three semi batch centrifugal concentrators (Falcon SB1350) installed in parallel will treat the -2 mm cyclone underflow. These machines automatically discharge on a fixed cycle (30 minutes is typical) and while discharging, the feed will be diverted to the other two machines by an automatic valve. The concentrate, approximately 35 kg per discharge, will be pumped to the feed holding tank for the shaking table. The concentrators and the shaking table will be in a walled enclosure with limited access and camera surveillance.

The cyclone overflow will be fed to two continuous gravity concentrators installed in parallel (Falcon C2000). The denser material is discharged as a high percent solids slurry (approximately 70% solids) and the mass of concentrate may be varied using an array of air operated pinch valves in the machine. The mass of concentrate will be adjusted to be 12 – 15% of the feed. The tailings from these concentrators will be pumped to the tailings filtration unit.

The concentrate will be diluted to the desired solids concentration (45% solids) and fed to the leach plant.

17.6 Leach Plant

A leach plant will be installed at Ollachea with a capacity to treat 500 tonnes of material per day. The high-grade concentrate will first be passed over a shaking table and a high-grade concentrate with >20% gold will be separated and smelted directly to Doré. The table tailings will be pumped the leach circuit together with the concentrate.

A carbon in leach unit consisting of a pre-aeration tank to which lime (as a slurry) and air will be added, followed by 6 agitated tanks in series, with a volume of 12.5 m^3 (2.5 m diameter, 3 m high with 0.5 m freeboard) per tank, giving a total residence time of 24 hours. Each tank will be equipped with an interstage screen with 0.5 m^2 screen area and a mesh of 0.63 mm. The screens will be of the pump type, allowing all tanks to be on the same level, the screen lifts the slurry to flow into the next tank. A hose pump will be fitted to each tank to allow the slurry to be periodically pumped counter to the main slurry flow, moving the carbon in counter current to the slurry. A vibrating screen will be mounted on the first tank to allow slurry to be pumped to the screen by a hose pump to recover the loaded carbon. Loaded carbon will be collected in a 2 m^3 tank (1 tonne of carbon) and when full, will be pumped to the existing elution circuit using a hose pump. Air will be supplied to the slurry using three spargers set into the walls of the tanks. These spargers will produce a fine stream of air bubbles to promote oxygen mass transfer to the solution.

Sodium cyanide will be added to the first CIL tank and further lime as required to maintain the pH >10.5. The tailings from the CIL unit will flow to the cyanide destruction unit.

A crane fitted with a chain block will be installed on each tank to allow removal of the interstage screens for cleaning.

A diesel generator will be included to supply power to the agitators in the case of grid power failure.

17.7 Carbon Treatment

Carbon loaded with gold and silver (approximately 8000 g/t total gold + silver) will be removed from the adsorption unit, screened, and washed on a vibrating screen fitted with 0.8 mm screen panels. The undersize slurry will return to the adsorption circuit. The carbon will be transferred to an acid leach column for removal of carbonates using dilute hydrochloric acid. Approximately 2 tonnes of carbon will be produced per batch and the acid wash and elution column will be approximately 1 m in diameter and 6 meters high for this amount (approximately 4 m^3).

After washing to remove the residual acid, the carbon will be transferred to the elution column, fabricated in carbon steel, insulated and approximately 1 m in diameter and 6 m high. A hot solution of sodium hydroxide and sodium cyanide (1% and 0.1%, respectively) will be passed through the carbon at a temperature of 140-145 degrees to de-sorb the metal values from the carbon. The operating pressure in the column will be 0.4 MPa. Two heat exchangers and a propane fired solution heater will be used to heat up the solution, control the temperature in the column and the temperature of the solution exiting the column. The solution, containing gold and silver will then flow to an electrowinning cell where gold and silver will deposit on the cathode. The solution will recycle to the elution column and this flow will continue until the gold and silver levels in the carbon have reached approximately 50 g/t each. This generally takes 12 hours and the system will then be cooled, all liquid will be transferred to the barren solution tank and the carbon will then be pumped from the column as a water slurry and transferred to the feed hopper of the carbon regeneration kiln. This will be a rotary furnace, heated using propane, operating at 650 – 700 degrees. The regenerated carbon will be quenched in water, passed over a sizing screen to remove any fines and then be transferred back to the carbon adsorption unit. Makeup carbon will be dumped from its container (a big bag) into a tank fitted with an agitator and after an hour of agitation in water, will be passed over the sizing screen to remove any fines. It will join the main carbon inventory.

17.8 Electrowinning and Refining

Solution from the elution column will flow directly to two electrowinning cells, operating in series where gold and silver will be deposited on stainless steel wire mesh cathodes. The solution exiting the cells will flow to the barren solution storage tank, from which it will be pumped back to the elution column. The barren solution tank will be of sufficient size to accommodate all the solution from the elution column and electrowinning cells.

Every two days, the first cells in line will be drained and the cathodes will be pressure washed to remove the precious metal sludge and it, and sludge accumulated in the bottom of the cells will be washed into a filter press. The filter cake will be dried overnight in a tray drier and then smelted in an induction furnace with the addition of silica and borax fluxes. The furnace will be fitted with an extraction hood and any fume will be collected in a bag filter unit. Excess slag will be poured into slag moulds and the molten metal will be poured into bar moulds. The Doré bars will be cleaned, numbered, and weighed prior to shipping. The slag will be broken up, any large pieces of metal recovered manually, and the rest will be recycled to the ball mill.

The electrowinning cells will be equipped with extraction hoods and fans, and the air will be scrubbed for removal of gases prior to discharging.

17.9 Cyanide Destruction

Cyanide destruction will be carried out in two tanks in series, the piping will allow each tank to be bypassed if required, each tank will provide a 1-hour residence time. The diameter will be 3.4 m diameter and 4.2 m high. A high-power agitator will provide good gas dispersion of the air sparged into the tank by three sparger units set into the tank walls.

Sodium metabisulfite will be used as a source of sulfur dioxide, and copper sulfate solution to maintain a copper concentration of 15 mg/l will be added to catalyse the reaction. The pH in the tanks will be lowered by the formation of sulfuric acid during the cyanide destruction process and the pH will be controlled by lime slurry addition. The pH will be controlled at 8.5 to ensure the destruction of iron cyanide complexes.

17.10 Tailings Filtration

Tailings will be pumped to a 10 m diameter thickener, to produce a feed slurry of approximately 55% solids for the filter presses. Flocculant will be added using the suppliers proprietary control system. The overflow will be pumped to the process water tank, the underflow will be pumped to the two automated filter presses. Each filter press will have 70 plates, 2000 mm x 2000 mm, with a cake volume of 11.5 m^3. Discharge will be automated and controlled with a programable logic controller (“PLC”). Water collected during the filtration cycle will be collected with so called “bomb doors” which will open automatically when cake is ready to discharge. The expected moisture content of the filter cake is 16%.

The thickener and filters will be supplied as a package, with all controls and piping, a control room, flocculant makeup system, filtrate tank and the filter feed pump.

The filters will be installed in a steel framed structure, with a roof and partial sides for weather protection and the product conveyors will stack the filter cake on a concrete pad, from which it will be loaded onto trucks using a wheel loader and transported to the paste plant or to tailings management facility.

17.11 Paste Plant

The estimated flow rate of paste required is 60 m^3/hour of paste at full (3,000 tpd) production. A plant with this capacity will be installed for stage 1 to eliminate the need for expansion.

The plant will receive filtered tailings by truck, and these will be stored on a concrete pad 15 m x 15 m with 1 m high walls on each side to minimize possible contamination with large objects (rocks and the like) which could cause blockages.

The plant package will consist of:

a. Bulk bag unloader with dust collector
b. Cement hopper
c. Feed screw
d. Tailings (filter cake) hopper with live bottom feeder
e. Tailings conveyor
f. Water system
g. Paste mixer
h. Steel frame
i. Paste hopper
j. Full automation, Motor Control Center ("MCC") and controls room.

A paste pump with a maximum discharge pressure of 40 bar and a maximum flow rate of 90 m^3/h will be used to deliver the paste.

The plant will be fed with filter cake using a wheel loader.

17.12 Reagents

The makeup and storage systems for sodium cyanide, copper sulfate, sodium metabisulfite and lime will be housed in a building with sufficient area to also provide storage. Sodium cyanide and sodium metabisulfite ("SMBS") will be delivered in 1 tonne big bags and these will be lifted into the dust tight, vented bag cutting unit. Lowering the big bags will cut them open and the reagent will drop into an agitated tank filled with water (5 m^3). The cyanide dissolution tank will also have 5 kg of sodium hydroxide added prior to adding the sodium cyanide. Dissolution will be rapid and after one hour, the solution will be pumped to a stock tank (10 m^3). Metering pumps will deliver sodium cyanide solution to the CIL circuit and sodium metabisulfite solution to the cyanide destruction circuit. The two units will be very similar, the only difference being that the SMBS unit will have stainless steel tanks. Copper sulfate will be received in 25 kg bags and bags will be manually added to an agitated tank as required. A metering pump will add copper sulfate solution to the cyanide destruction unit. Lime will be stored in a silo next to the reagent building and a mixing and storage tank will be housed inside the same building. All reagents and materials will be delivered by road.

Compressors for instrument air and for general use, and the blowers for the leach circuit, water pumps and fire pumps will also be installed in the reagent building. Water tanks for process water, raw water, fire water and potable water, insulated and fitted with heaters to prevent freezing will be installed adjacent to the reagent and service building.

17.13 Expansion to 3000 tpd

The plant will be expanded to treat 3,000 tpd with the following additions:

- A larger screen and cone crusher will be added to the crushing plant.
- The grinding and gravity concentration section will be duplicated. The CIL circuit will have sufficient capacity to treat the concentrate produced from 3,000 tpd of ore, as will the cyanide destruction and carbon treatment unit.
- The filtration plant will be duplicated.

The water treatment plant, the water supply system and the paste plant will all have sufficient capacity for the 3,000 tpd throughput with no further additions.

18 PROJECT INFRASTRUCTURE

18.1 Site Preparation

The planned site for the processing plant and other infrastructure slopes to the northeast. Horizontal platforms will be made to accommodate this infrastructure. The 2012 Feasibility Study ("2012 FS") planned these earthworks in detail, and this study has been used in the preparation of the permitting application. This Technical Report considers the same values used in the 2012 FS, with the additional increased erosion protection (see Capital Cost Estimate – Section 21) to compensate for the removal of "soil nailing", a technique seldom used in Peru. Key volumes and areas from the 2012 FS have been summarized in (Table 18-1).

Table 18-1: Site Preparation Earthworks - Key volumes and areas

Item	Unit	Count
Balance Cut to Fill	m3	43,661
Excess Cut to Spoil	m3	140,708
Erosion Protection	m2	5,338

Lined channels will be installed to collect non-contact water and direct it to existing drainages. Mine water and run-off from stockpiles and the plant will be collected in a channel system which will discharge into the main catchment pond, prior to treatment.

18.2 Buildings

Accommodation and associated infrastructure

It is envisaged that the mining contractor will construct their own campsite, including all required offices and warehouses for the mine development and exploitation.

Members of the workforce from the local area will return home after each day of work and will not require accommodating.

The Issuer leases several hotels in the town of Ollachea that will be used to house the remaining workforce.

A restaurant with the capacity to feed 120 people a day will be constructed at site.

A first aid room and ambulance garage will also be built.

Laboratory

A grade control and process control laboratory will be built at site for the daily analysis of grade control samples from the mine and process control samples from the plant. Primarily, samples will be analyzed for gold and silver with some capacity to analyze water samples. As is required by law, a third-party laboratory will also be used for the analysis of water samples.

Gold and silver analysis will be by fire assay. The laboratory will be sized to handle up to 100 samples per day and will consist of:

- Sample preparation – jaw crusher and pulverisers
- Fluxing room – fume hoods for adding lead-based fluxes to the samples
- Furnace room – smelting and cupelling furnaces, with area for slag / lead button handling, fume extraction
- Parting room – Fume hood for nitric acid parting
- Weighing room – microbalance for weighing gold prills
- Atomic adsorption – fume hood for dissolving prills, atomic adsorption machine
- Office
- Toilet facilities
- Store – for samples, crucibles and cupels.

This laboratory will be housed in a prefabricated building of 240 m^2 with a solid, concrete floor. It will be built between the river and the Southern Interoceanic Highway.

Truck Shop

A workshop will be required for the maintenance of the mining equipment (supplied by the contractor), mobile equipment used by the plant, and to a lesser extent maintenance of the plant equipment and light vehicles.

A workshop area of 24 x 12 m will be developed, with overhead lifting equipment and workbenches. A compressor and oil and grease units will be located outside the main workshop under a lean-to roof. A concrete wash pad (8 x 10 m), with curb walls, a sump and an oil trap will be provided for washing down the equipment using a hot water pressure washer.

An office (3 x 4 m), a storeroom (6 x 10 m), a lunchroom (3 x 4 m) and toilet facilities will also be included in the truck shop. The total area of the building will be 400 m^2.

Administrative Offices

An office block with 12 individual offices for managerial staff from the mine, plant, and administration and 20 workstations will be built, to include a conference room, toilet facilities and a small kitchen. The total area of the office block will be approximately 300 m^2.

Change rooms and showers (Mine dry)

Changing rooms, showers and toilets will be provided for the mine and plant personnel. The area of this building will be 200 m^2.

18.3 Water System

The overall water system is shown in Figure 18-1. Water from the mine, plant, and water collected from the waste dump will flow into the main catchment pond that will collect all potentially contaminated water. The volume of this pond will be 6000 m^3. The maximum flow rate for treatment, derived from a dynamic simulation (AMEC 2013), is 320 m^3 per hour. This will be used to design the water treatment plant. It is noted that during the development of the Exploration Tunnel, seepage from the tunnel was less than simulations indicated.

The water treatment plant will consist of two agitated tanks in series, of 320 m^3 each, to which air and lime will be added to give a neutral pH (7) and to precipitate iron as ferric hydroxide. Traces of other base metals will also be precipitated. A thickener will settle the remaining solids that will be fed to the tailings thickener and disposed of in the filtered tailings. The treated water will recirculate to the process water tank, and excess treated water will be directed to a lined pond (2000 m^3), which will overflow into the non-contact water drainage system.

Most of the water used in the plant will be recycled from the tailings thickener to the process water tank (800 m^3) and will be reused. Additional water will be needed to compensate for water lost in filtered tailings (15% moisture), water lost to the geotubes (a 50% moisture slurry will be fed to the geotubes and this water will return to the collection pond from the waste dump), evaporation and other losses. Additional water will be sourced from the environment control pond water. The volume of treated water available to the plant via the environmental control pond should exceed requirements. If additional water is required, a supply of raw water will be available via a waterline from the river. Additional water requirements for the plant are estimated to be a maximum of 750 m^3 per day, the waterline and pumping station will be sized for this flow rate. Water from the river will be pumped to a raw water tank of 450 m^3. Raw water supply will be made available for fire-fighting purposes at the lower platform.

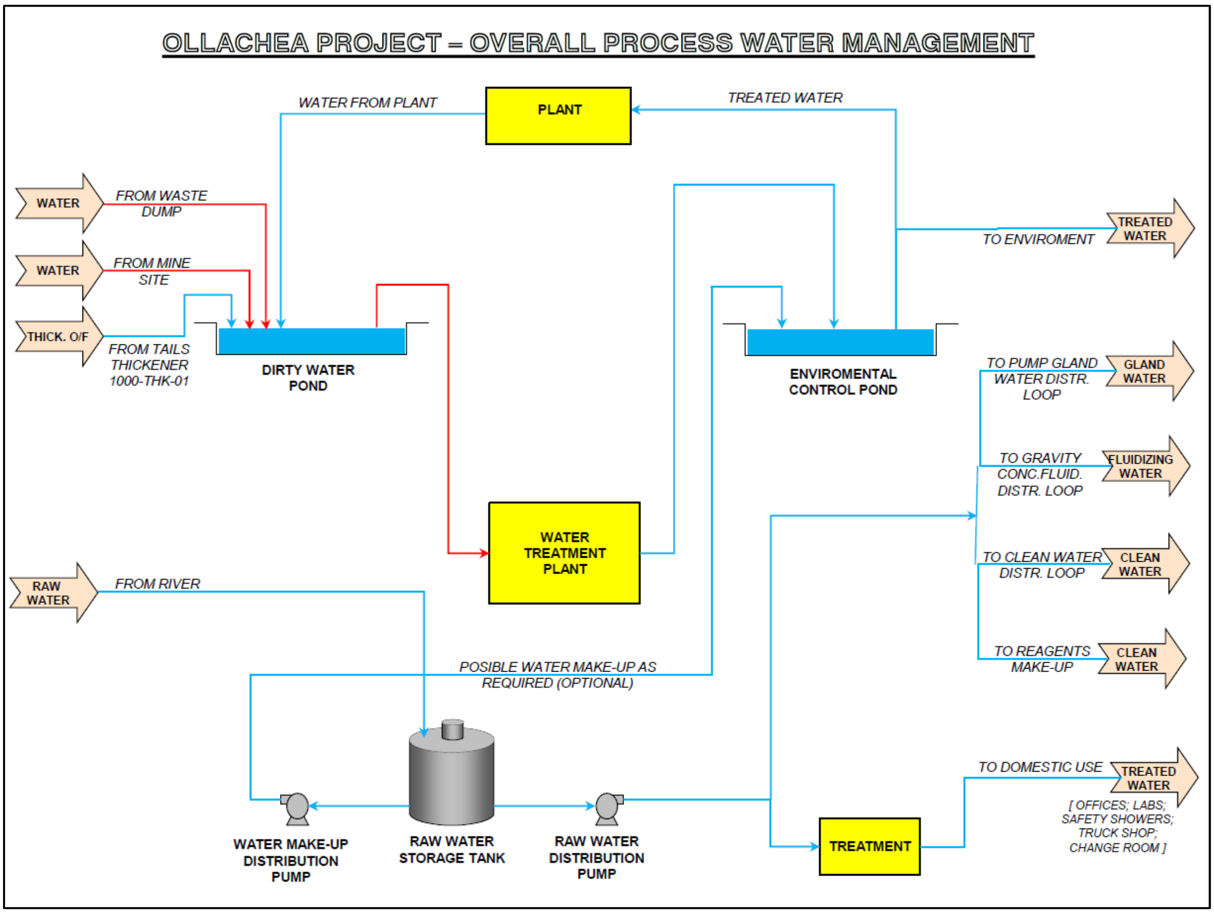

Figure 18-1: Water System

18.4 Electrical Power Supply

Power will be taken from the 138 kV power line that passes approximately 900 m from the sub-station. The substation will transform this voltage to 13.8 kV to supply the mine, the paste plant, pumps at the foot of the waste and tailings management area and for other infrastructure such as the truck shop, offices, and laboratory. The plant will be supplied with 4160 volts for the ball mill motor and 460 volts for the rest of the plant. The substation will have the normal earth grid and lightening protection. A diesel-powered emergency generator will provide power to lighting systems and plant equipment that should not be shut down, such as the leach plant agitators and thickener drives.

18.5 Tailings and Mine Waste Management

The Ollachea mine waste management concept has been developed to minimize the impacts of storing of tailings and waste rock materials. The concept includes the following key aspects:

- 43% of tailings to be returned to the mine as paste backfill.
- Remaining 57% of tailings to be filtered to a low moisture content and stacked in a system of co-disposed mine waste rock and filtered tailings product.
- Co-disposal will occur at two locations: the Lower Portal Co-Disposal Facility (Lower Portal CDF) and the Cuncurchaca Co-Disposal Facility (Cuncurchaca CDF).

Storage of tailings and waste rock at each of the locations is shown in Table 18-2 below.

Table 18-2: Storage of Waste Rock and Tailings by Location

Location	Waste Rock (Mt)	Tailings (Mt)	Total by Location (Mt)
Lower Portal CDF	1.65	0.85	2.50
Cuncurchaca CDF	1.29	4.60	5.89
Underground Backfill	--	4.20	4.20
Total by Waste Type (Mt)	2.94	9.65	12.59

Note: Waste rock and mineralized material tonnages, and mine backfill volumes provided by Mining Plus – Excel file 'Indicative Mine Schedule For Ollachea (002).xlsx' received 12-Jul-2021

The total mine life is approximately 11 years. Filtered tailings will be placed at the Lower Portal CDF during the first 2.5 years, approximately. For the remaining years, the filtered tailings will be transported approximately 4.0 km from the plant site to the Cuncurchaca CDF using 15 m^3 capacity trucks. The trucks will be equipped with covered beds to minimize dusting and spillage during transport. The haul route includes approximately 2.0 km along the Interoceanic Highway and 2.0 km along access roads at the process plant and the Cuncurchaca CDF.

Filtered tailings was selected as the most suitable tailings processing method, primarily to obtain the required storage volume within a relatively limited distance from the process plant. This was not possible with conventional slurry tailings disposal or thickened tailings disposal methods, due to topographic limitations in the project area. Additional benefits offered by filtered tailings, relative to conventional or thickened tailings, include reduced land disturbance, and reduced tailings storage facility seepage/effluent.

Tailings from the CIL circuit will be thickened to 60% solids and pass-through cyanide detoxification prior to being dewatered using pressure filtration. The filtered tailings are anticipated to be dewatered to a moisture content of approximately 16%, as required to achieve sufficient compaction at the co-disposal facilities.

Contingency planning for 'out-of-spec' tailings, that have a higher moisture content due to upset conditions at the filtering station, consists of the use of geotube tailings storage. Geotubes are very large geosynthetic bags, designed to retain the tailings solids, while allowing water to drain out, and thereby allowing consolidation of the tailings to a low moisture content, similar to mechanically-filtered tailings. The geotubes will be located within the body of the Co-disposal Facilities ("CDF"), such that separate contingency areas are not required.

18.5.1 Site Conditions – Lower Portal Co-Disposal Facility

Site conditions at the Lower Portal CDF have been assessed in previous studies. Information in this section is primarily summarized from the document 'Ollachea Gold Project, PERU, NI-43-101 Technical Report on Feasibility Study; Prepared by AMEC; Effective Date 29 November 2012' (2012 FS).

The Lower Portal CDF will be located in a small valley south of the process plant. The valley is constricted at the toe of the waste dump providing lateral confinement of the lower portion of the waste dump. Typical natural ground slopes range from 20 to 27 degrees within the waste dump footprint.

The planned Lower Portal CDF area is characterized by debris-flow deposits overlying slate and meta-sandstone bedrock corresponding to Ananea and Sandia Formations. Slate bedrock outcrops are observed in the upper slopes (southern portion) of the waste dump. A high angle fault has been inferred near the projected toe of the waste dump.

Geotechnical site investigation information from previous versions of the site were reviewed. These investigations included test pits and boreholes. The test pits revealed shallow soil conditions consisting of approximately 0.3 m of topsoil overlying silty sand and gravel with cobbles and boulders (USCS classification GM and GP-GM). The soils were generally dry to moist, non-plastic, and medium dense to dense.

The boreholes in the vicinity of the Lower Portal CDF indicate soils to depths of 21 to 40 m. Soil samples recovered during drilling were generally characterized as dry to moist, medium dense to very dense, silty sand, gravel, and cobbles. Penetration testing indicated the foundation soils are typically medium dense to very dense.

Two standpipe piezometers were installed in boreholes BH7 and BH8 along the valley bottom within the waste dump footprint. These piezometers indicate depth to groundwater at approximately 3 m along the valley bottom. Piezometers installed in boreholes on the west side of the waste dump indicate depths to groundwater ranging from 22 m to greater than 30 m.

Bedrock was encountered at a depth of 30 m near the projected toe of the waste rock dump (BH7) and at a depth of 11 m on the slopes of the central area of the waste dump footprint (BH8). The bedrock is constituted by slate and meta-sandstones with basic RMR values ranging from 32 to 49 for the slate, and from 20 to 47 for the meta-sandstone.

Based on site-specific hazard analyses, the site is considered to be of moderate seismicity.

18.5.2 Site Conditions – Cuncurchaca Co-Disposal Facility

Site conditions at the Cuncurchaca CDF have been assessed in previous studies. Information in this section is primarily summarized from the document 'Ollachea Gold Project, PERU, NI-43-101 Technical Report on Feasibility Study; Prepared by AMEC; Effective Date 29 November 2012', and is based on the following:

- 10 geotechnical boreholes ranging in depth from 40 to 85 m.
- 10 test pits at Cuncurchaca CDF and along proposed Cuncurchaca access road.
- Soil and rock sampling, SPT/LPT, and in situ permeability testing.
- Standpipe piezometers were installed in five boreholes to monitor depth to groundwater.
- Two inclinometers were installed immediately down gradient of the proposed TSF to monitor potential ground movements.

The TSF area consists of Quaternary sandy gravel and cobble deposits resulting from a series of debris-flow events from the Cuncurchaca drainage basin. Soils were characterized as dry to moist, medium dense to very dense, sandy gravel and cobbles with trace to little non-plastic fines. These Quaternary deposits are estimated to have thicknesses ranging from 50 to 150 m and overlie lightly metamorphosed sandstone from the Paleozoic Sandia formation. It is inferred that these deposits once dammed the Ollachea River but have since been eroded to form steep slopes on the west side of the river. In recent years, slope-ravelling and shallow slope failures have been observed, likely a result of construction cuts for the Southern Interoceanic highway. No indications of deep-seated or active large-scale slope movement were observed during the field reconnaissance.

Based on site-specific hazard analyses, the site is considered to be of moderate seismicity.

18.5.3 Lower Portal Co-Disposal Facility – Design Considerations

The mine waste schedule, as provided by Mining Plus (Excel file 'Indicitive Mine Schedule For Ollachea (002).xlsx' received 12-Jul-2021), includes 1.65 Mt of mine waste rock and 0.85 Mt of filtered tailings to be placed at the Lower Portal CDF location (see Table 18-2), during approximately the initial 2.5 years of operations. As such, the Lower Portal CDF will comprise approximately 66% waste rock and 34% tailings by mass.

The Lower Portal PCDF is located within a few hundred meters of both the Process Plant, where the filtered tailings will be produced; and also, the Lower Portal access to the mine, from where the vast majority of the mine waste rock will be extracted. So, the Lower Portal CDF will be the first of the two CDF's to be developed, in order to defer the development of the Cuncurchaca site.

The Lower Portal CDF can be described as a 'sidehill' waste stack, as both the waste rock and tailings will be placed against an existing steep slope. Site preparation, prior to placement of waste materials, will include:

- Stripping of topsoil from the footprint area. Stripped topsoil will be salvaged for use in reclamation activities.
- Construction of perimeter diversion structures to manage surface runoff.
- Excavation of the current slope in a benched manner, to increase available volume, and to improve stability to an acceptable level.
- Placement of impermeable liner materials at the base of the dump: a combination of natural clayey soils and geosynthetic clay layers (GCL – to be placed on cut slopes).
- Construction of CDF underdrain system, to capture seepage through the waste stack.
- Construction of a starter berm at the toe of the stack, to control runoff and increase stability of the stack during the initial stages of placement.
- Construction of ponds to receive both contact and non-contact water, prior to potential treatment and discharge.

The final configuration of the waste stack will have a maximum height of approximately 125 m, from toe to crest. The ultimate configuration includes 30 m lifts at angle of repose (assumed 1.4H:1V) with 15 m benches between lifts to maintain a global waste dump slope of 2.0H:1V. The Lower Portal CDF can be seen in Figure 18-2.

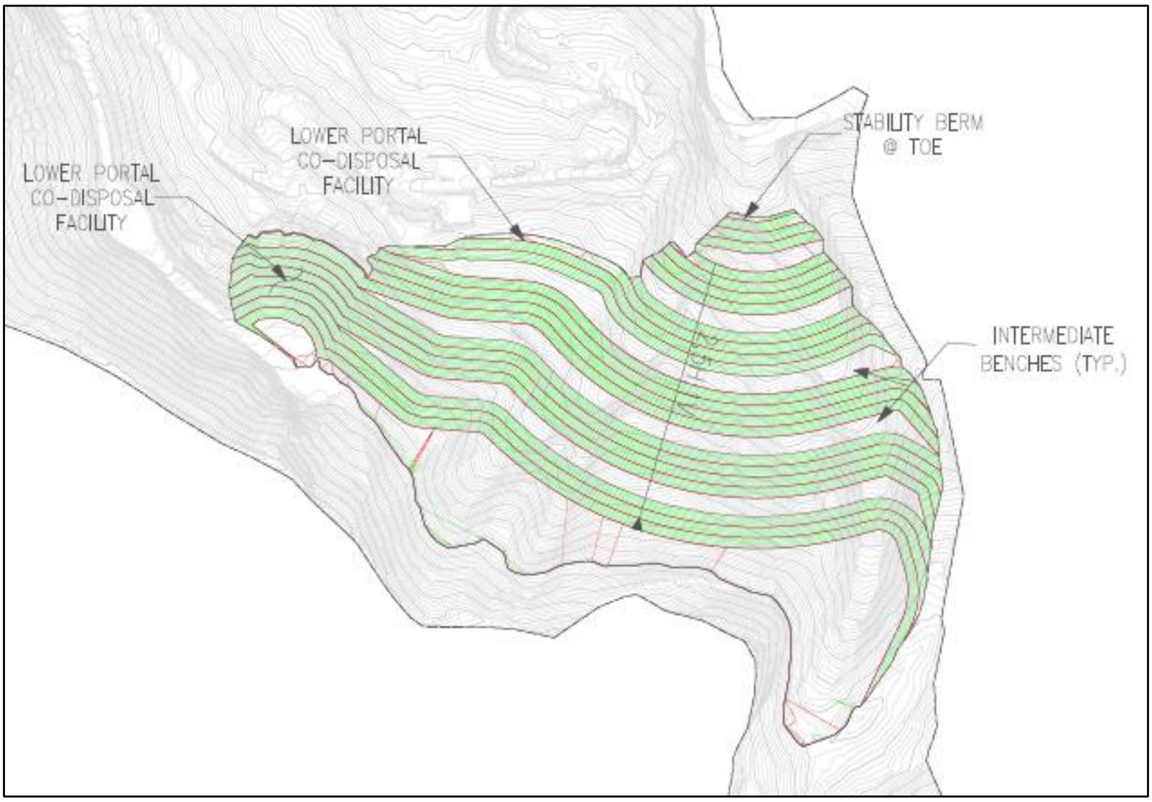

Figure 18-2: Lower Portal Co-Disposal Facility

Both waste rock and filtered tailings will be placed in controlled, horizontal lifts, compacted to a density sufficient to meet shear strength and stability requirements. It is expected that the filtered tailings would need to be produced with a moisture content of around 16%, which is typically near the optimum to ensure proper compaction at the stack.

To further improve stability, the higher-shear-strength waste rock will be placed in controlled lifts toward the outside of the stack, where in-stack slope failures would be more likely to occur. Conversely, the filtered tailings would be placed toward the inside of the stack, against the hillside. The waste rock will also serve as an erosion -protection layer at the outside slope of the CDF.

As indicated previously, as a contingency to manage off-spec tailings, that might have moisture content higher than the optimum of around 16%; the design of the Lower Portal CDF includes for pumping and piping these tailings as a slurry, to the in-progress tailings zone within the stack. At that location, the slurry tailings will be discharged into permeable geosynthetic geotubes. The geotubes will retain the solids within very large geosynthetic bags, while the water from the slurry will decant off, through the non-woven geotextile. As such, a separate area for managing off-spec tailings will not be required.

Lower Portal CDF inter-lift benches will include surface water ditches to capture and shed contact water and direct it to the contact water management pond. A raincoat system will also be used, primarily on the areas of filtered tailings, to convey direct precipitation to the contact water management system, and prevent infiltration and wetting of the tailings.

18.5.4 Cuncurchaca Co-Disposal Facility – Design Considerations

The mine waste schedule, as provided by Mining Plus (Excel file 'Indicitive Mine Schedule For Ollachea (002).xlsx' received 12-Jul-2021), includes 1.29 Mt of mine waste rock and 4.60 Mt of filtered tailings to be placed at the Cuncurchaca CDF location (see Table 18-2); beginning approximately at 2.5 years into operations, and continuing to the end of LOM, at year 11. As such, the Cuncurchaca CDF will comprise approximately 22% waste rock and, 78% tailings, by mass.

The Cuncurchaca CDF is located approximately 4 km from the Lower Portal and the Process Plant and will be developed after the Lower Portal CDF.

The Cuncurchaca CDF can be described as a 'sidehill' waste stack, as both the waste rock and tailings will be placed against an existing steep slope. Site preparation, prior to placement of waste materials, will include:

- Clearing and grubbing of low-lying vegetation; stripping of topsoil from the footprint area. Stripped topsoil will be salvaged for use in reclamation activities
- Construction of perimeter diversion structures to manage surface runoff
- Excavation of the current slope in a benched manner, to increase available volume, and to improve stability to an acceptable level
- Construction of CDF underdrain system, to capture seepage through the waste stack
- Construction of a starter berm at the toe of the stack, to control runoff and increase stability of the stack during the initial stages of placement
- Construction of ponds to receive both contact and non-contact water, prior to potential treatment and discharge
- For seepage control of the Cuncurchaca CDF stack, it is noted that this stack will have a much higher proportion of low-permeability, filtered tailings, than the Lower Portal CDF

(78% vs 34%). As such, the tailings themselves will serve as the low-permeability, seepage reduction element at the Cuncurchaca CDF site; separate materials such as borrowed clay, GCL, or plastic liners would not be required.

The final configuration of the Cuncurchaca CDF waste stack will have a maximum height of approximately 150 m, from toe to crest, at an overall, global slope of 2.5H:1V. The Cuncurchaca CDF can be seen in Figure 18-3.

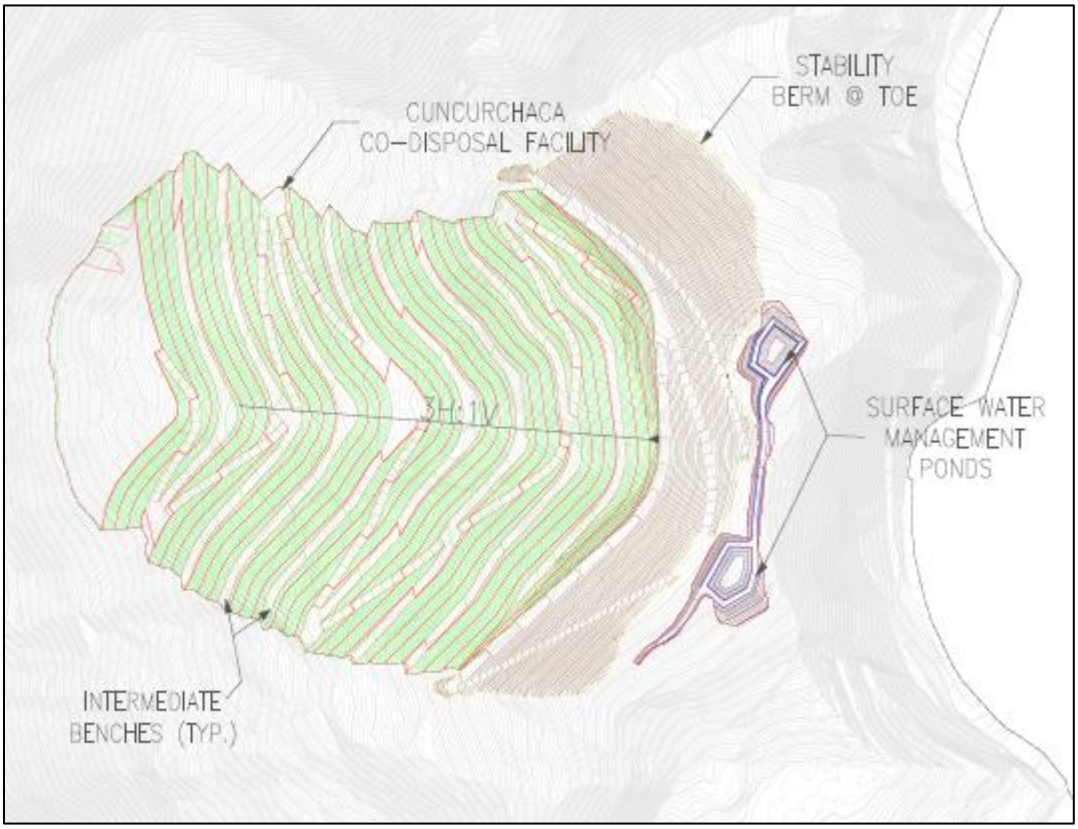

Figure 18-3: Cuncurchaca Co-Disposal Facility

Both waste rock and filtered tailings will be placed in controlled, horizontal lifts, compacted to a density sufficient to meet shear strength and stability requirements. It is expected that the filtered tailings would need to be produced with a moisture content of around 16%, which is typically near the optimum to ensure proper compaction at the stack.

To further improve stability, the higher-shear-strength waste rock will be placed in controlled lifts toward the outside of the stack, where in-stack slope failures would be more likely to occur. Conversely, the filtered tailings would be placed toward the inside of the stack, against the hillside. The waste rock will also serve as an erosion-protection layer at the outside slope of the CDF.

As indicated previously, as a contingency to manage off-spec tailings, that might have moisture content higher than the desired optimum of around 16%; the design of the Lower Portal CDF includes for pumping and piping these tailings as a slurry, to the in-progress tailings zone within the stack. At that location, the slurry tailings will be discharged into permeable geosynthetic geotubes. The geotubes will retain the solids within very large geosynthetic bags, while the water from the slurry will decant off, through the non-woven geotextile. As such, a separate area for managing off-spec tailings will not be required.

Cuncurchaca CDF inter-lift benches will include surface water ditches to capture and shed contact water and direct it to the contact water management pond. The Cuncurchaca CDF has been designed for a capacity of 5.85 Mt of filtered tailings with overall ultimate slopes of 2.5H:1V, and an approximate maximum ultimate height of 150 m as measured from the toe of the starter buttress to the crest. A contingency area for temporary tailings management has been designated near the TSF access road for drying and temporary storage of “off-spec” tailings resulting from upset conditions at the plant or wet weather. The temporary tailings management area will include a geomembrane-lined area and contact water pond.

18.5.5 Co-Disposal Facilities - Closure Considerations

Closure of the CDFs will include construction of a vegetative soil cover system; implementation of water management controls; and re-vegetation of disturbed areas. Progressive reclamation of slopes will be required during operations to control erosion and fugitive dust. For final closure, the un-reclaimed portion of the stack surfaces will be graded to promote drainage to areas designated by the closure surface water management plan. A final cover system will be constructed over the CFDs. The CDFs and other disturbed areas will be revegetated.

Water drainage courses will be formed for closure conditions including upgrades to the CDFs’ perimeter surface water channels. Although seepage from the CDFs is expected to be negligible, seepage will be monitored and treated if necessary to meet Peruvian water quality standards.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Metallurgical test work indicates that the main product from the Property will be gold doré bars. Gold has a readily available market for sale in the form of gold doré or gold concentrates.

19.2 Commodity Price Projections

Gold Price analysis has been provided by a Peruvian investment banking and asset management firm (Capia Servicios Financieros) in an internal document titled Project Thor – Gold Price Analysis dated June 2021.

For the economic analysis, the gold price is assumed at $1,600/oz. The price guidance follows industry consensus on long-term metal prices and fundamental macroeconomic analysis. The gold price was kept consistent throughout the life of the Project.

19.3 Contracts

No sales contracts or off-take agreements for the sale of concentrate products from the Property are in place at the time of writing the Technical Report. It is expected that sales contracts would be typical of, and consistent with standard industry practices comparable with the Corihuarmi mine which Minera IRL operates.

19.4 Comments on Section 19

Edgard Vilela (QP) considers the gold price assumptions are appropriate and consistent with other current studies and are suitable for use in the mine plans and financial analysis.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Edgard Vilela (QP) is responsible for the contents of Section 20 of the Technical Report.

To complete Section 20, Mr. Vilela has relied on information provided by MIRL, also stated in Section 4.

20.1 Baseline Studies

A physical, biological, and socioeconomic baseline has been established from ongoing social, environmental, and archaeological baseline studies conducted by MKK during the preparation of the EIA, in addition to quarterly environmental monitoring data between 2007 to Q4 2020 (physical baseline data from 2013 to 2020).

20.1.1 Physical Baseline

The study area is sited in the Ollachea river sub-watershed, located in the Inambari river watershed (spans the Puno and Madre de Dios regions), which flows into the Atlantic Ocean basin. According to monitoring activities performed from 2013 to 2020 at five stations situated across the project site, a peak value of 101.3 m^3/s was reported in Q3 2014, with the lowest value of 8.5 m^3/s reported in Q1 2018.

Water Quality

Results of the quarterly water quality monitoring in the study area undertaken between 2014 to 2020 indicate that water quality generally meets national water quality standards. Exceptions include cases of high concentrations of copper and manganese that have been found in the Ollachea river over the past years, related to waste from artisanal mining activities placed at the margin of the Oscco Cachi valley.

Air Quality

Air quality has been measured through 12 monitoring points, where the parameters analyzed meet the Peruvian environmental regulations. Records show that the concentrations of particulate matter (PM10 and PM2.5) between 2012 and 2020 were below the Environmental Quality Standard (ECA, Spanish acronym) for air quality, except in the year 2014, where the AIR-06 station exceeded the standard values. The unexpected readings were most likely caused by vehicle traffic near the monitoring point (Air-06). All analyzes of gases show that the SO2, NO2, H2S and O3 concentrations from 2012 to 2020 comply with the ECA standard for air quality.

Noise Pollution

The baseline noise levels have been monitored from 2013 to 2020 in the industrial zones of the study area and are below the national day and nighttime ambient noise standards. The noise levels recorded in the city of Ollachea were above the day and nighttime standards, due primarily to the traffic on the Interoceanic Highway.

Current land use in the study area consists of natural and cultivated pastures or planted forests, as well as unused or unproductive land. It has been identified that the land could be potentially used for forestry production, grazing, permanent agriculture, and protection land (land with no economic potential intended for natural plantations that prevent any loss of soil by erosion).

20.1.2 Biological Baseline

Vegetation in the study area consists of subtropical montane rainforest, subtropical montane lowland rainforest, and humid subtropical montane lowland rainforest.

Sixty-three (63) plant species were identified in the study area during the 2019 wet season. They have been grouped into 24 botanical families and these in turn are grouped into 16 botanical orders, 73% (46 species) of which belong to Magnoliopsida, a large group characterized worldwide by its diversity, followed by Liliopsida with 25% (16 species), which is considered as the species best adapted and within which the predominant are grasses and the Gnetopsida class with 2% (1 species).

No species listed in Peruvian Supreme Decree 043-2006-AG - Classification of Threatened Species of Wild Flora, were found in the study area. As per the Convention on International Trade in Endangered Species of Wild Fauna and Flora (CITES, 2017), one single species was found in the study area: Myrosmodes cf. gymnandra. This species belongs to the Orchidaceae family and is classified under Appendix II. The classification under Appendix II includes species that are not considered threatened with extinction, but with which trade must be controlled to protect the species ensuring its survival. There are no records of species classified according to the International Union for Conservation of Nature – IUCN. In addition, no endemic species were found in the study area.

20.1.3 Socioeconomic Description

To analyze and understand the socioeconomic situation in the area related to the Project, the approved area of direct social influence (“ADSI”) has been defined and is comprised of the Ollachea Population Center and the small town of Asiento. In addition, the approved area of indirect social influence (“AISI”) is comprised of the Ollachea Farming Community, which includes the urban population center having the same name.

The Ollachea Population Center is an urban group with a population of 6568 inhabitants, which demographically account for 27% of the Ollachea district. The district also covers the population centers of Parusani, Bellavista, Quicho, Munaypata, Palca, Pumachanca and Altiplano Chia, identified in the 2017 National Census (INEI 2017). Ollachea is the capital of the district.

The Ollachea Farming Community was recognized in the Public Records Office in 1974 and encompasses 27,584 hectares with 750 registered community members according to the records provided by its Community Board. However, the records of the 2017 census of families and homes reported only 650 members, which is explained by migration to other cities while retaining communal affiliation.

A small urban population center has been developed within the boundaries of the Farming Community of Ollachea, which has improved its interconnection with other towns through the Southern Interoceanic Highway. The families of the community have established their usual residence and built their homes within the urban population center.

The Ollachea community is the main demographic group of the population center of the same name, although it also includes immigrants made up of public officials, professionals, and merchants.

The company's operations and administrative offices are located in the Ollachea Population Center, which is the area of direct social influence. It is with the community, as a legally recognized social organization, that negotiations and mutually beneficial agreements were entered into prior to and during the exploration phase, which extend through to the construction and production phases.

The local government is elected by the district's citizens. Therefore, all of them share the same territory, the same authority (Mayor and municipal councilmen, Governor, Justice of the Peace) and the same budget. The tax benefits resulting from future mining operations will significantly benefit the entire population of the district. The highest authority is the Board of the Farming Community of Ollachea, which two-year term is renewed by the vote of the registered community members. Board members cannot be re-elected.

20.1.4 Community and Social Projects

The Ollachea Mining Project has strong support from local communities, both in the areas of direct and indirect influence. This project is expected to become a major source of employment in the area. MKK has conducted a continuous program of community awareness and communication workshops and has worked closely with the Ollachea Community since it entered into the agreement to acquire the property from Rio Tinto in 2006.

MKK's cooperation in the formalization of illegal mining activities on the Property and its surface rights agreement with the Ollachea Community are part of a plan to engage the community to the maximum extent possible in the progress and future operation of the Project.

The Peruvian Government has authorized the Processing Plant construction and declared the admissibility of the Mining Start-Up Authorization based on the 30-year agreement entered into with the Ollachea Farming Community.

The Environmental Certification granted by the Government includes a Community Relations Plan, as well as the community relations protocols.

The social aspect is a dynamic one and the mine owners, such as MKK, have social relations programs in place that are under permanent review. The priority programs include the creation of community companies, the acquisition of local goods and services, local employment, communication channels, social relations protocols, among other activities and efforts.

Something that specifically sets this project apart from others in the region is the fact that the surface rights agreement between MKK and the Ollachea Community is combined with a strategic social management plan that involves the different stakeholders around the project.

A total of 26 social programs are underway, benefiting over 65% of the population since 2007. In compliance with the United Nations Sustainable Development Goals ("SDGs"), the company has worked to eradicate malnutrition among children and the elderly, provide technical training for employment, develop community companies, contribute to the education of children and young people, provide employment and health care, among other initiatives.

An important program is the Ollachea Music and Dance Center ("CEMDAO"), with school performances that encourage and promote the natural talent of more than 120 children and youngsters in the community, providing an opportunity for them to become virtuoso dancers and artists of the charango, mandolin, panpipe, violin, and guitar.

CEMDAO has contributed to a reduction of the number of children working in the artisanal mine shafts and played a role in maintaining the cultural heritage of the Puno region through the recovery of ancestral songs and dances. In 2014 this program received an award for Social Innovation.

The Association of Artisan Women of Ollachea ("AMARE") is a group of female community members and is another example of a social program supported by the Company. The sale of hand-woven, fine alpaca wool garments provide financial support for community families.

AMARE products are successfully sold in several artisanal fairs and through its own webpage www.amare.pe

MKK continues with its community programs and maintains an excellent relationship with the Ollachea Farming Community.

20.2 Archaeological Baseline

The Semi Detailed Environmental Impact Assessment on the Ollachea Exploration Project, submitted in 2008 by Compañía Minera Kuri Kullu S.A. ("MKK"), identified several areas with archaeological evidence as part of the first exploration program. The archaeological survey defined the existence of an archaeological site and one pre-Hispanic road in the Oscco Cachi valley area, seven archaeological sites and three pre-Hispanic roads in the Cuncurchaca valley area, and three archaeological sites in the Corani valley area.

The "Amendment to the Semi Detailed Environmental Impact Assessment on the Ollachea Exploration Project" in 2009 included a new archaeological reconnaissance project that was carried out in the expansion area (216 hectares), which confirmed the existence of five areas with archaeological evidence in the Oscco Cachi valley area.

20.2.1 Archaeological Evaluation Projects

The report "Archaeological Evaluation Project with Excavations on the Cuncurchaca Exploration Area of the Ollachea Mining Project - Department of Puno" was submitted in April 2012. The archaeological evidence found in this sector included one circular structure, one semicircular structure, and one rock shelter.

20.2.2 Archaeological Rescue Project

In February 2012, MKK submitted to the Ministry of Culture the "Emergency Archaeological Rescue Project of Unexpected Findings in the Challuno Area of the Ollachea Mining Exploration Project - Minera Kuri Kullu", to begin with the recovery of the archaeological evidence found in this sector.

Minapampa Area Project, Sector 1

The Report 059-2015-APC/DDC PUN/MC dated April 6, 2015, prepared by Alexis Gonzalo Pizarro Cárdenas, identified by National Archaeologist Register (RNA) Number CP0784, an archaeologist of the Decentralized Directorate of Culture - Puno, concluded that no surface archaeological remains were found in this project area. The relevant Certificate of Non-Existence of Archaeological Remains (CIRA, Spanish acronym) No. 054-2016 was issued, which states that there are no surface archaeological remains in this project area.

Challuno Area Project, Sector 1

The Report 060-2015-APC/DDC PUN/MC dated April 6, 2015, prepared by Alexis Gonzalo Pizarro Cárdenas, identified by RNA No. CP0784, an archaeologist of the Decentralized Directorate of Culture - Puno, concluded that no surface archaeological remains were found in this project area. The relevant CIRA No. 055-2016 was issued, which states that there are no surface archaeological remains in this project area.

Cuncurchaca Area Project, Sector 1

The Report 061-2015-APC/DDC PUN/MC dated April 6, 2015, prepared by Alexis Gonzalo Pizarro Cárdenas, identified by RNA No. CP0784, an archaeologist of the Decentralized Directorate of Culture - Puno, concluded that no surface archaeological remains were found in this project area. The relevant CIRA No. 056-2016 was issued, which states that there are no surface archaeological remains in this project area.

Ollachea Project's Supplementary Areas, department of Puno

The Report 000145-2016-APC/DDC PUN/MC dated December 16, 2016, prior to the issue of CIRA No. 336- 2016, concluded that no surface archaeological remains were found in this project area.

Challuno Area and Minapampa Area

Director's Resolution 036-2020-DGPA-MC approved the final report on the Archaeological Project and CIRA No. 101-2021-DDCPUN/MC was issued in respect of the Challuno Area (Sector 1, Sector 2, and Sector 3) and the Minapampa Area.

20.3 Environmental Management

The Environmental Management Instruments ("IGA") include environmental certifications, closure plan, permits, among others, and are mainly granted by the Ministry of Energy and Mines, the National Environmental Certification Service for Sustainable Investments ("SENACE"), and the National Water Authority ("ANA"). The IGAs for the Ollachea Mining Project were approved through Director's Resolution 363-2013-MEM/AAM and are valid throughout the life of the mining project, as ratified by SENACE through Report No. 090-2018-SENACE-PE/DEAR.

Table 20-1 lists the approved IGAs of the Mining Project.

Table 20-1: Ollachea - Approved Environmental Management Instruments

Environmental Management Instruments		
Environmental Certification	**Effective Date**	**Approved Instrument**
Director's Resolution 363-2013-MEM/AAM	September 25, 2013	Environmental Impact Assessment ("EIA") for mining and processing at Ollachea
Director's Resolution 120-2014-MEM-AAM	March 12, 2014	Rectification of Director's Resolution 363-2013-MEM/AAM
Director's Resolution 615-2014-MEM-DGAAM	December 18, 2014	Supporting Technical Report (ITS), Ollachea Mining Project
Director's Resolution 061-2016-SENACE/DCA	August 4, 2016	Extension of the Validity of the Environmental Certification until September 25, 2018
Report 090-2018-SENACE-PE/DEAR	October 5, 2018	Confirmation of Validity of the Environmental Impact Assessment of the Ollachea Mining Project

Environmental Certification has been granted for the Ollachea Mining Project considering an underground operation and 3000 tpd production over 10 years.

There are no known environmental issues that could materially impact the issuer's ability to extract mineral resources or mineral reserves, beyond what is established and approved by the Mining Authority in the Environmental Management Instruments.

Under the current surface rights agreement with the Ollachea Community, MKK is supervising artisanal miners and taking steps to mitigate the additional environmental liability associated with small-scale mining activities. This includes regular monitoring of water quality both upstream and downstream of the mine to identify any possible contamination related to artisanal mining activities.

Section 18 of the Preliminary Economic Assessment (PEA) addresses the requirements and plans for the disposal of mine waste and tailings produced, the supervision of the disposal site, and water management, both during the mine operations phase and post mine closure.

The PEA incorporates the same environmental control and quality and safety principles set out in the approved Environmental Impact Assessment ("EIA"). Mineral processing approved in the EIA applies gravity concentration and leaching. These processes have been optimized and are detailed in Section 13 of the PEA. This optimization involves technological improvement with no significant impact on what was approved under the EIA and will be supported before SENACE through a Supporting Technical Report ("ITS") to obtain the relevant Environmental Certification.

20.4 Permits

Ollachea is one of the most advanced mining projects, yet to enter production, in Peru in terms of exploration carried out and authorizations obtained for prompt construction and start-up. The PEA considers an initial start-up with a capacity of 1500 tpd, followed by an expansion in year 4 to reach production of 3000 tpd, as established in the approved EIA.

The Environmental Management Instruments ("IGA") approved for the Ollachea Mining Project consider and ensure the Environmental Management and Mine Closure Plans, these include:

- Waste disposal plans: The waste dumps have been approved
- Tailings disposal plans: The Tailings Storage Facility Design and Construction Authorization have been approved
- Environmental monitoring: The monitoring plans required for environmental management at the Ollachea Mining Project have been approved. Such plans include the Air quality (noise, dust, gases), water quality, and soil quality monitoring, among others. The approved Environmental Management Plan and its environmental monitoring plan are applicable to the different project stages, namely, construction, mining, and closure.

Certain mining components of the Ollachea project were built during the exploration stage and will be used during the mining stage, including a waste dump, access roads, main portal, 1,234 m Exploration Tunnel, effluent management facilities, among others.

Exploration Stage:

Table 20-2 lists the environmental authorizations that have been obtained for the exploration stage of the Property.

Table 20-2: Ollachea - Exploration Stage Approved Environmental Documents

Approval Document	Date	Description
Director's Resolution 241-2008-MEM-AAM	September 30, 2008	Semi Detailed Environmental Impact Assessment (EIAsd) on the Ollachea Mining Exploration Project
Director's Resolution 068-2010-MEM-AAM	March 1, 2010	First Amendment to the EIAsd on the Ollachea Mining Exploration Project
Director's Resolution 140-2011-MEM/AAM	May 6, 2011	Second Amendment to the EIAsd on the Mining Exploration Project – "Ollachea Exploration Tunnel"
Director's Resolution 177-2014-MEM/AAM	April 15, 2014	Third Amendment to the EIAsd on the Ollachea Exploration Project (in force, with suspended activities).

Mining and Processing Stage

The Ollachea Mining Project has authorization for the construction of a 3000 tpd Processing Plant, including all crushing, milling, gravity concentration, leaching, and desorption processes up to the production of doré bars.

All supporting permits required to apply for authorization to mine have been granted. MKK intends to apply for authorization to mine upon filing of the PEA. The typical period of processing of authorization to mine applications is 12 months.

Often (10) permits are granted for a given period and require regular renewal (e.g. Explosives, Water, Discharges, etc.).

Table 20-3 provides lists of approved permits for Ollachea.

Table 20-3: Ollachea Project - Approved Permits

No.	Summary of Permits and Authorizations		Resolution
1	**Processing Plant Construction Authorization** (including the metallurgical plant, TSFs, and auxiliary services)		Director's Resolution 235-2014-MEM-DGM/V
2	**Mining Start-up Authorization**		In progress; CIRA ratification pending.
3	Authorization for the Discharge of Treated Industrial Wastewater from the Ollachea Project		Director's Resolution 027-2015-ANA-DGCRH
4	**Authorization for the Discharge of Treated Domestic Wastewater from the Ollachea Project**		Director's Resolution 131-2016-ANA-DGCRH
5	**Water Use**	Authorization to Use Pallecapampa and Oscco Cachi Water for the Execution of Works.	Director's Resolution 223-2016-ANA (being renewed)
		Authorization to Use Cuncurchaca Water for the Execution of Works.	Director's Resolution 224-2016-ANA (being renewed)
6	**Right of Way**	To Connect the Interoceanic Highway with the Operations Area.	Director's Resolution 375-2013-MTC/20
		For the Discharge Pipeline that runs to the Ollachea River.	Director's Resolution 004-2013-MTC/20
		To Connect the Interoceanic Highway with the access road to the magazine.	Director's Resolution 007-2013-MTC/20
		To Connect the Interoceanic Highway with the access road to the Cuncurchaca area.	Director's Resolution 054-2014-MTC/20

No.	Summary of Permits and Authorizations		Resolution
		To Connect the Interoceanic Highway with the access road to the Espinaspampa Campsite.	Director's Resolution 128-2014-MTC/20
7	**Certificate of Non-Existence of Archaeological Remains**	Includes certain areas in Minapampa, and the waste dump area.	CIRA 054-2015
		Includes certain areas in Challuno, and the Processing Plant area.	CIRA 055-2015
		Includes certain areas in Cuncurchaca, and the TSF area.	CIRA 056-2015
		Includes supplementary areas.	CIRA 336-2016
		CIRA 101-2021-DDCPUN/MC for Challuno Area (Sector 1, Sector 2 and Sector 3) and Minapampa Area	CIRA 101-2021-DDCPUN/MC
8	**Authorization for the Construction of Liquid Fuel Warehouse.**		Regional Resolution 5015-2016-OS/OR PUNO.
9	**Electrical Substation Construction Authorization**	Compliance with the Supporting Technical Report of the component: Variation of the San Gaban II Transmission Line (138 kV) submitted by Compañia Eléctrica San Gabán S.A.	Director's Resolution 117-2014-MEM-DGAAE
		Compliance with the Pre-Operational Study relating to the connection of the Ollachea Mining Project to the National Electric Power Grid (SEIN, Spanish acronym).	COES/D/DP-1367-2013
10	**Surface Property Usufruct Authorization - Social Agreement**		Ollachea Community Agreement.

The project has a valid Environmental Certification approved by the Ministry of Energy and Mines through Director's Resolution 363-2013-MEM/AAM. Upon evidencing before the National Environmental Certification Service for Sustainable Investments ("SENACE") that the construction of certain components (access roads, main tunnel, among others) had already been completed, SENACE confirmed the indefinite validity of the Environmental Certification by means of Report 090-2018-SENACE-PE/DEAR.

Processing Plant Construction Authorization (3000 tpd):

Authorization for the construction of 3000 tpd Processing Plant has been granted by the Ministry of Energy and Mines through Resolution 0235-2014-MEM-DGM/V dated June 25, 2014.

Mining Start-up Authorization:

A favorable report has been issued on the Mining Start-up Authorization procedure (Detailed Engineering for Hydrology, Hydrogeology, Geotechnics, Ventilation, Waste Dump Design, Mining Plan, etc.) and the administrative process is being restarted upon filing of the PEA.

Social License

The Ollachea Project has a surface property usufruct authorization granted by the members of the Ollachea Farming Community, which enabled a successful exploration program and was subsequently renewed for the mining operations stage. The Social License was formally ratified through an agreement signed on May 30, 2012, which extends the validity period of the surface rights permit for 30 additional years, the longest period ever granted in the country, which symbolizes the strength of the commitment and support of the local community.

The Government of Puno and the National Dialogue and Sustainability Office ("ONDS") describe Ollachea's social relations model as an example of tangible coexistence that prioritizes the development and principles of the Shared Value model, due particularly to the granting of a 5% interest in MKK to the community upon commencement of production.

20.5 Artisanal Miners

Artisanal mining activities are carried out in the surroundings of the Ollachea Mining Project. In order to establish socially and environmentally responsible management, the agreement entered into by and between MKK (owner of the Ollachea Mining Project) and the Ollachea Farming Community (owner of the surface property) establishes that, upon commencement of operations, the Company will give priority to local employment by incorporating the artisanal miners into the investment project.

This strategy is in line with the government's legal mandate regarding the mining formalization process, which establishes that it is not possible to formalize or grant areas to artisanal miners within the Ollachea Mining Project, due to the indivisibility principles and the restricted areas included in the approved EIA.

As per the Framework Agreement entered into by and between MKK and the Ollachea Farming Community, prior to the start of the construction phase, artisanal miners must abandon their workings and be included on a priority basis in the project's employment offer, as well as the creation of services companies. A recent study revealed that only 79 of the 123 artisanal miners authorized by the community are active, with one working each.

20.6 Mine Closure Plan

A formal Mine Closure Plan ("PCM") has been developed as part of the Project's feasibility work plan.

Director's Resolution 222-2015-MEM-DGAAM approved the Mine Closure Plan of the Ollachea Mining Unit. Furthermore, in 2020, Director's Resolution 026-2020-MINEM-DGAAM approved the Mine Closure Plan Update.

The core purpose of the Mine Closure Plan is to ensure that the environment surrounding the mining unit re-establishes the same quality conditions it had before the start of mining activities. Therefore, it seeks to safeguard public health and safety of the populations directly or indirectly affected during the execution of progressive closure activities, guarantee the long-term physical and geochemical stability of closed facilities, reclamation of affected soils, maintain the balance of river basins, and minimize the economic impact from the end of mining operations.

Table 20-4 shows all certifications obtained related to the Mine Closure Plan.

Table 20-4: Ollachea Project - Environmental Management Instruments (Mine Closure Plan)

Environmental Management Instruments (Mine Closure Plan)		
Environmental Certification	**Date**	**Approved Instrument**
Director's Resolution 222-2015-MEM/AAM	May 26, 2015	Approval of Mine Closure Plan of the Ollachea Mining Unit (rectified by Director's Resolution 455-2015/MEM-DGAAM)
Resolution 0429-2016-MEM-DGM/V	July 20, 2016	The creation of financial guarantees for the PCM is postponed from 2017 to the start of operations
Director's Resolution 026-2020-MINEM-DGAAM	January 24, 2020	First Mine Closure Plan update of Ollachea Mining Unit

According to the mining regulations, PCM updates are required five years following initial approval. Therefore, this must be submitted in late 2024.

A bond amounting to US$542,191 was issued for the Ollachea Mining Project's PCM. This bond is renewed every year until the start of operations, as established by the Ministry of Energy and Mines through Resolution 0429-2016-MEM-DGM/V. From the year following the start of operations, the bond will be subject to an annual increase of approximately US$550,000 on average until reaching a total amount of US$4.52M.

The approved total mine closure cost is US$7.71M, including VAT. This amount is calculated as follows: US$3.18M pertains to progressive closure during the operations stage, US$3.87M pertains to final closure, and US$0.65M pertains to post-closure, which is extended for 5 years following completion of final closure.

Based on the above, the mine closure is approved and guaranteed and includes remediation and reclamation measures.

As mentioned above, following the EIA update pursuant to current regulatory mechanisms, such as the Supporting Technical Report ("ITS"), a new amendment to the Mine Closure Plan will be required based on the PEA results.

20.7 Comments on Section 20

Given the current status of the permits and the agreement with the community for the project, the environmental, archaeological, and social baseline work completed so far, and the well-established permitting process in Peru, there are currently no known social, environmental, or archaeological issues that may materially affect MKK's capacity to extract the mineralized material on the Property.

Environmental liabilities associated with artisanal mining activities are expected. MKK has a mitigation program in place, which entails regular monitoring of water quality both upstream and downstream of the mine to identify any possible contamination related to these mining activities.

Due to its nature and scope, the PEA does not imply a significant change and does not require an Amendment to the EIA. It only requires the use of mechanisms to update the environmental instrument ("IGA") by means of a Supporting Technical Report ("ITS"). This is because the social and environmental impacts of the PEA proposal are non-significant, and these additionally entail a reduced use of reagents and reaffirms the safe management of mine waste and tailings and comprehensive water management.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

This PEA considers the viability of a low-CAPEX start-up for Ollachea with an underground mine, gravity concentration and carbon in leach ("CIL") plant designed to treat 1500 tonnes per day ("tpd") over the first three years (Stage 1), ramping up to 3000 tpd during the fourth year (Stage 2).

The Start-up Capital Costs are defined as those costs required to achieve Stage 1 at 1500 tpd ("Start Up Capital"). The Expansion Capital Costs are defined as the costs to achieve Stage 2, and include the expansion of the waste storage facilities in year 2, and the processing capacity to 3000 tpd in year 3.

Capital cost estimates have been prepared by the following parties as part of the 2021 PEA:

a. Mining Plus: Mining related costs
b. JAT Metco: Process and onsite infrastructure costs
c. Envis: Mine waste disposal costs.

All currencies are reported in U.S. dollars (US$), unless otherwise specified.

21.2 Capital Cost Estimate Summary

A summary of the Ollachea capital cost estimates is shown in Table 21-1.

Table 21-1: Capital Cost Estimates

Description	US$
Start-up (Stage 1)[1]	
Mine	$27M
Process Plant [2]	$37M
Tailings and Waste Rock Disposal	$5M
Owner's Costs	$2M
Start-up Capital Costs Pre-Contingency	**$71M**
Contingency (25%)	$18M
Total Start-up Capital	**$89M**
Expansion (Stage 2)[3]	
Process Plant	$16M
Tailings and Waste Rock Disposal	$13M
Owner's Costs	$1M
Expansion Capital Costs Pre-Contingency	**$30M**
Contingency (25%)	$7M
Total Expansion Capital	**$37M**

(1) Includes mine development and plant construction with a design capacity of 1500 tpd.
(2) Includes EPCM costs. Also applicable to expansion.
(3) Includes Tailings Storage Facility construction and process plant ramp-up from 1500 tpd to the designed capacity of 3000 tpd.

21.3 Basis for the estimate – Capital Costs

The capital cost estimate has been developed to provide an estimate suitable for the 2021 PEA, including costs to design, procure, construct, and commission the facilities. The expected accuracy range of the capital cost estimate is +30%/-30%.

The PEA estimates an initial CAPEX of US$89M to start with a design production capacity of 1,500 tpd. A plant expansion is anticipated during the fourth year to increase production capacity to 3,000 tpd. The expansion capital cost estimate is approximately US$37M. Both estimates include a 25% contingency.

21.4 Mine Capital Costs

Mine capital costs consider the development required to achieve production at 1,500 tpd.

The development required to support a 1500 tpd operation is estimated to be 7,570 m of horizontal development and preparations, and 869 m of vertical development.

Considering estimated horizontal development including preparation costs of US$2,400/m, and vertical development including preparation costs of US$4,800/m, total CAPEX cost (development aspect only) is estimated at US$22.34M. The total CAPEX of US$27M also includes US$3.5M for initial primary ventilation fans and paste infrastructure.

The development required for ongoing mine production including ramping up the mine to 3000 tpd has been considered as OPEX.

21.5 Process Plant and Infrastructure Capital Costs (Stage 1)

The capital cost estimate for the process plant and associated infrastructure has been based on the following. Stage 1 capital process plant and infrastructure has been summarized in Table 21-2:

- Process flow diagrams developed from results of test work.
- Equipment list.
- General arrangement drawings and site layout plans.
- Specifications for major equipment.
- Budget quotations from vendors for major equipment, mostly as packages.
- Budget pricing for bulk materials.
- Geotechnical and hydrogeological reports as presented in a previous feasibility study and detailed design for the Co-Disposal Facilities.
- Regional climactic data.
- Project work breakdown structure (WBS).

Table 21-2: Stage 1 -Estimated Capital Cost for Capital Process Plant and Infrastructure

CAPEX	Stage 1 Capex (US$)
Accommodation	$0.255M
Crushing	$2.640M
Grinding	$3.303M
Gravity concentration and re-grind	$6.064M
Concentrate leaching	$1.934M
Carbon elution, electrowinning and smelting	$3.059M
Cyanide Destruction	$0.226M
Filtration	$3.067M
Water treatment	$0.986M
Reagent storage and make up	$1.487M
Paste fill preparation	$2.222M
Laboratory	$0.440M
Workshop	$0.250M
Offices	$0.297M
Water supply system	$0.688M
Site development	$2.871M
Electrical sub-station and power reticulation	$3.355M

CAPEX	Stage 1 Capex (US$)
Fuel storage	$0.066M
Communications	$0.064M
Site roads	$0.360M
Mobile equipment	$0.550M
Capital spares	$0.294M
First fills	$0.227M
Temporary construction	$0.054M
Total Direct costs	**$34.759M**
EPCM	$2.472M
Total	**$37.231M**

The previously completed 2012 Feasibility Study (2012 FS) had developed a site preparation plan which had been used to obtain permitting. The site preparation plan used in the PEA ("Technical Report") has been left largely unchanged, and material quantities used in the 2012 FS, with some additional allowances, have been used in the Technical Report. Unit rates have been updated based on the contractor rates at the company's current operating mine.

Concrete requirements (volumes) for the foundations of the mineral processing equipment have been estimated based on experience gained from similar projects. Estimated capital costs have been based on concrete prices for finished concrete supplied by a national contractor.

Single quotes have been obtained for the major pieces of mineral processing equipment, grouped into packages, typically inclusive of electrical switchgear, instrumentation, and some pipework. Based on experience gained from similar projects, additional allowances have been considered for electrical cabling and remaining pipework. Quotes have been obtained for the following equipment packages:

- Complete crushing plant.
- Ball mill and accessories, gravity concentration circuit, and regrind mill.
- Agitators, interstage screens.
- Tailing filtration system.

Installation costs have been estimated using local labour rates and man hours needed for each installation and have been based on experience gained from similar projects.

Estimated capital costs to construct buildings have been based on costs supplied by a local contractor and based on their experience from similar projects.

Estimated capital costs for the electrical supply, sub-station and distribution have been supplied by an experienced contractor.

Estimated capital costs for crane rental have been estimated based on prices paid by a contractor for similar projects.

21.6 Process Plant and Infrastructure Expansion and Sustaining Capital (Stage 2)

Sustaining capital cost estimates consider ongoing capital expenditures required to sustain operations and the expansion of production from 1,500 tpd to 3,000 tpd in year four of operation.

The expenditure for the expansion of the processing facilities from 1,500 to 3,000 tpd occurs in year 3 in preparation for year 4 when the ramp up to 3000 tpd is scheduled.

Expansion capital cost estimates have been summarized in Table 21-3:

Table 21-3: Stage 2 -Estimated Capital Costs for Process Plant and Infrastructure

CAPEX	Stage 2 CAPEX (US$)
Crushing	$0.528M
Grinding	$3.303M
Gravity concentration and re-grind	$6.064M
Filtration	$3.066M
Water treatment	$0.099M
Laboratory	$0.088M
Workshop	$0.125M
Offices	$0.059M
Site development	$0.207M
Electrical sub-station and power reticulation	$0.287M
Fuel storage	$0.033M
Communications	$0.032M
Site roads	$0.036M
Mobile equipment	$0.503M
First fills	$0.227M
Temporary construction	$0.054M
Total Direct costs	**$14.711M**
EPCM	$1.000M
Total	**$15.711M**

Total Sustaining Capital for life of mine maintenance and critical spares has been estimated at US$0.850M.

21.7 Tailings and Waste Rock Disposal Capital Costs (Stage 1)

Estimated initial capital costs consider tailings and waste rock disposal for the first 2.5 years of operation and are shown in Table 21-4. These costs have been updated based on the 2012 FS; and also a detailed-design and cost estimate from 2014 (Anddes 2014). The previous cost estimates were adjusted to reflect differences to the design concepts, and updates to current unit costing.

The costs shown pertain to the development of the Lower Portal Co-Disposal site, which will allow for storage.

Table 21-4: Tailings and Waste Rock Disposal Capital Costs (Stage 1)

Tailings and Waste Disposal Costing Item	Stage 1 (US$)
Early Works	$0.30M
Earthworks	$2.00M
Liner and Underdrains	$2.31M
Surface Water Diversion (Non-Contact)	$0.60M
Geotubes - Tailings Contingency	$0.10M
Total direct	**$5.31M**

21.8 Tailings and Waste Rock Disposal - Expansion and Sustaining Capital (Stage 2)

Sustaining capital costs consider the ongoing capital expenditures required to sustain operations. Planned expansion and related capital expenditures of tailings and waste rock storage facilities have been considered beginning in year 2 of operations. These costs have been updated based on the 2012 FS; and also a detailed-design and cost estimate from Anddes 2014. The previous cost estimates were adjusted to reflect differences to the design concepts, and updates to current unit costing.

By year 2.5 of operations, the Cuncurchaca Co-Disposal Area will be required. Estimated capital costs to develop the Cuncurchaca Co-Disposal Area are summarized in Table 21-5.

Table 21-5: Estimated Capital Costs to develop the Cuncurchaca Co-Disposal Area (Stage 2)

Cuncurchaca Co-Disposal Area - Years 2.5-11	Stage 2 (US$)
Early Works	$0.30M
Earthworks	$10.00M
Liner and Underdrains	$0.88M
Surface Water Diversion (Non-Contact)	$1.10M
Geotubes - Tailings Contingency	$0.85M
Total direct	**$13.13M**

21.9 Owner's Costs

MIRL provided estimated owner's costs based on their experience operating a mining project. Estimated owner's costs include insurance, camp costs, safety, and training.

21.10 Operating Cost Estimate Summary

The Technical Report (PEA) contemplates an underground mine from which mineralized material will be trucked to a gravity concentration and CIL plant located close to the main portal. The expected accuracy range of the operating cost estimate is +30%/-30%.

Life of Mine ("LOM") operating costs are summarized inTable 21-6.

Table 21-6: Estimated Life of Mine Operating Costs

Operating Costs	LOM (US$)	$/tonne leached	$/oz Au
Mining (1)	$406	$42.10	$464
Processing	$127	$13.11	$144
Tailings and Waste Rock Disposal	$35	$3.66	$40
Onsite G&A (2)	$35	$3.65	$40
Total Operating Costs	**$603**	**$62.52**	**$688**
Treatment & Refining Charges	$4	$0.44	$5
Government Royalty	$35	$3.63	$40
Royalties (3)	$41	$4.21	$46
Community Interest	$11	$1.14	$13
Total Cash Costs	**$694**	**$71.94**	**$792**
Sustaining Capital	$1	$0.13	$2
All-in Sustaining Costs (AISC)	**$695**	**$72.08**	**$794**

(1) Includes paste backfill, supervision and stope definition drilling costs.
(2) Includes mine closure bond.
(3) Includes NSR of 2.9%.

Operating cost estimates have been developed to provide an estimate suitable for the Technical Report (PEA), including costs for mining, processing, and waste disposal.

Contingencies have not been considered when estimating operating costs.

21.11 Mining Operating Costs

Average estimated LOM operating costs have been estimated at $42.10/tonne (t). Estimated operating cost ("OPEX") consider proposed mine plan, local cost benchmarking and experience from similar operations and local conditions. It has been envisaged that mining operations will be carried out by a contractor. Average estimated LOM operating costs have been summarized in Table 21-7.

Table 21-7: Estimated Mine Operating Costs

Mine Operating Costs	Total Cost (US$)	US$/tonne leached
Mining cost (US$/yr)	$348,382,148	$36.11
Paste Cost ($4.50/t)	$28,664,208	$2.97
Mine Supervision Cost	$11,096,349	$1.15
Stope definition drilling Cost	$18,000,000	$1.87
Total Direct Costs (US$/t)		***$42.10***

21.11.1 Mining Cost

Estimated mining costs consider the following aspects:

- Horizontal Development + preparation (US$/m)
- Vertical Development + preparation (US$/m)
- Stopes Production (US$/t).

A breakdown of the calculated costs is presented in Table 21-8.

Table 21-8: Estimated Mine Operating Costs

	Development (m)	Cost per meter (US$)	Total Cost (US$)
Horizontal Development meters	58,560	$2,400	$140,542,400
Vertical Development meters	1,466	$4,800	$7,035,200
Total (US$)			**$147,577,600**
	Stope Tonnes (t)	**Cost per tonne (US$)**	**Total Cost (US$)**
Stopes Production Tonnes (t)	7,753,071	25.9	$200,804,547
Total (US$)			**$200,804,548**
Total Mining Cost (US$)			**$348,400,000**
Total Mineralized Material (t)			**9,647,000**
Cost Per Tonne (US$/t)			**$36.11**

All costs are built up from first principals based on an indicative contractor quote, and crosschecked against internal Mining Plus benchmarking.

21.11.2 Paste Cost

Estimated paste costs are on internal benchmarking for paste production. Total cost is based on the m^3 of voids created needing to be filled. The estimated cost is US$29M over the life of mine.

21.11.3 Mine Supervision Costs

Estimated labour costs have been developed based on MIRL's experience running a mining operation in Peru. Estimated labour costs consider local labour rates and overheads.

Estimated mine supervision costs consider the following roles and a total of 53 people. Estimated mine supervision costs have been summarized in Table 21-9. The estimated cost is US$11M over the life of mine.

Table 21-9:Estimated Mine Personnel Costs

Department	Position	Count
Mine Management	Mine Manager	1
	Mine Superintendent	1
Geology	Geology Manager	1
	Chief Geologist	1
	Ore Control Geologist	3
	Assistant Geologist	2
	Ore Control	2
	Field Samplers (Drilling)	10
	Geotechnician	2
	Surveyor	1
	Assistant Surveyor	2
Projects	Head of Projects	1
	Assistant to the Head of Projects	1
	Field Supervisor	1
	Field Foreman	3
	Mason	2
	Laborer	6
Maintenance / HV	Maintenance Chief	1
	Maintenance Supervisor	2
	Welder	4
	Electrical HV Supervisor	2
	Electrician	3
	Instrument Technician	1
Total		**53**

21.11.4 Stope Definition Drilling

Stope definition drilling has been allocated to drill out stoping areas prior to production from dedicated underground platforms to improve the definition of mineralized bodies and to update and refine stope shapes. A total of US$18M has been allocated for stop definition drilling over the life of mine based on the estimated drill meters required.

21.12 Processing Operating Costs

Average estimated processing costs have been developed based on the design process flowsheet, and consider labour requirements for the processing plant and planned throughput rates. Estimated consumption rates of reagents, consumables, electricity, and maintenance have also been considered.

Estimated labour costs have been developed based on MIRL's experience running a mining operation in Peru. Estimated labour costs consider local labour rates and overheads.

Estimated power consumption has been developed based on a list of electrical equipment with estimated power requirements for each, and projected hours of operation. Power costs have been determined based on unit rates provided by a utility supplier.

Quotes have been acquired for major consumables, including delivery to site. Levels of consumption have been determined based on typical values or test work contained within previous studies, primarily the 2012 FS.

Mobile equipment will be rented. Rental costs for mobile equipment have been acquired from a major dealer. Fuel costs have been determined by projected hours of operation and manufacturer stated fuel consumption.

Maintenance cost of plant have been determined using 5% of the purchase cost of the plant equipment.

A summary of operating costs has been provided in Table 21-10.

Table 21-10: Estimated Processing Costs

Processing Costs (US$/t)	Stage 1	Expansion
Labour	$ 2.30	$ 1.45
Consumables	$ 3.15	$ 3.16
Electrical Power	$ 4.11	$ 3.71
Mobile Equipment	$ 2.27	$ 1.13
Plant Maintenance	$ 2.25	$ 3.41
Total Processing Cost	**$ 14.08**	**$ 12.86**

The average processing cost (total processing cost / total processed tonnes) across the two stages is US$13.11/t.

21.13 Tailings and Waste Rock Disposal Operating Costs

Tailings and waste rock operating costs relate primarily to the transport of filtered tailings and waste rock in 15 m^3 haul trucks. During the initial 2.5 years, these materials will be placed at the Lower Portal Co-Disposal Facility; during the remaining mine life, they will be transported approximately 4 km (on surface) to the Cuncurchaca Co-Disposal Facility.

As a result, operating costs for transport have been calculated on a per-cubic-meter basis, as required to haul the materials from their source locations (Lower Portal or Process Plant for waste and filtered tailings, respectively) to each of the two placement locations.

Haul costs were estimated based on recent experience at similar project sites and settings in Peru.

21.14 Onsite G&A

Onsite General and Administrative ("G&A") costs have been estimated for the following work areas:

- Administration
- Environmental Affairs
- Community Relations
- Security - 8 posts plus supervisor
- Communications allowance
- Office supplies, office maintenance etc
- Light vehicles (2 for mine supervision, 1 for each G&A dept.)
- Social relations costs
- Insurance.

The estimated number of staff for administration, environmental affairs and community relations is 47 people for stage 1, and 60 people for stage 2. The Onsite G&A costs are estimated at approximately US$150,000 to US$200,000 per month dependent on the stage of the project.

G&A costs associated with mine closure have also been considered.

21.15 Treatment & Refining Charges

The refining cost is estimated at 0.3% of gross revenue.

21.16 Government Royalty

The government royalty is 2.5% of net revenue.

21.17 Royalties Payable to Third Parties

Royalties on Gold production include Osisko, Macquarie and Sherpa and at the follow rates:

- Osisko Royalty – 1.0% of net revenue
- Sherpa Royalty – 0.9% of net revenue
- Macquarie Royalty – 1.0% of net revenue.

21.18 Community Interest

Pursuant to the surface rights agreement, the community was granted a participation of 5% in MKK. This commitment involves a payment estimated at US$1M per year based on the profit generated by the project, and is effective upon commencement of commercial production and continues throughout the life of the Ollachea project.

Given the financing repayment period, it is expected that the project will generate profits after the third or fourth year. However, the community will receive payment as from the first year of production.

22 ECONOMIC ANALYSIS

22.1 Cautionary Statements

Certain information and statements contained in this section and in the Technical Report are "forward looking" in nature.

All forward-looking statements in this Technical Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted.

Material assumptions regarding forward-looking statements are discussed in this Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Technical Report are subject to the following assumptions:

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the project assumptions as discussed in this Technical Report and may result in changes to the calendar timelines presented.

The preliminary economic analysis is partly based on Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the PEA based on these Mineral Resources will be realized.

22.2 Methodology Used

The financial analysis was carried out using a discounted cash flow (DCF) methodology. Net annual cash flows were estimated projecting yearly cash inflows (or revenues) and subtracting projected yearly cash outflows (such as capital and operating costs, royalties, and taxes). These annual cash flows were discounted back to the date of beginning of capital expenditure at mid-year 2022 and totaled to determine the NPV of the project at selected discount rates. A discount rate of 7% was used as the base discounting rate.

In addition, the IRR, expressed as the discount rate that yields an NPV of zero, and the payback period, expressed as the estimated time from the start of production until all initial capital expenditures have been recovered, were also estimated.

Sensitivities to variations in gold price, initial capital costs and operating costs were carried out to identify potential impacts on NPV and IRR.

22.3 Principal Assumptions

A financial model was completed based on the mine plan, which assumes the commencement of production in 2023, in addition to other inputs such as mining inventory and rates, processing throughputs and metallurgical recoveries, capital and operating costs, royalties, government royalty and taxation parameters.

22.3.1 Mineral Resource, Mine Plan, and Mine Life

The PEA mine plan is based on the subset of Mineral Resources stated in Section 14, with the subset discussed in Section 16.

Indicated Mineral Resources account for 95.9% of total mill feed and Inferred Mineral Resources for 4.1% of total mill feed.

The forecast mine and mill feed schedules were included in Table 22-5.

22.3.2 Metallurgical Recoveries

Metallurgical recoveries used for the financial analysis are 90.3% during the first three years, and 86.2% over the remaining LOM as discussed in Section 13.

22.3.3 Government Royalty

The government royalty is 2.5% of net revenue.

22.3.4 Royalties Payable to Third Parties

Royalties on Gold production include Osisko, Macquarie and Sherpa and at the following rates:

- Osisko Royalty – 1.0% of net revenue
- Sherpa Royalty – 0.9% of net revenue
- Macquarie Royalty – 1.0% of net revenue.

22.3.5 Metal Prices

A base case gold price of US$1,600/oz has been used in the financial modeling and is discussed in Section 19.

22.3.6 Discount Rate

The net present value ("NPV") was calculated from the cash flow generated by the project using a base discount rate of 7%. The discount rate was selected based on a benchmark analysis of recent mining project reports in Peru and other gold projects in Latin America. It considers risks associated with the project, commodity prices and country risks.

22.3.7 Capital Costs (including Expansion Capital)

Capital cost assumptions are outlined in Section 21. A construction period of 14 months was considered (starting in 2022) for the overall project implementation. Year 2023 corresponds to the first year of production. Capital costs were applied in the financial model excluding IGV/GST (IGV is the General Sales Tax in Peru).

Initial Capital Costs include mine development and plant construction with a design capacity of 1500 tonnes per day ("tpd").

Expansion Capital Costs include the Tailings Storage Facility construction and process plant expansion in order to achieve a production rate of 3000 tpd.

Capital Costs (including Expansion Capital) are shown in Table 22-1.

Table 22-1: Capital Costs (including Expansion Capital)

Description	US$
Initial Capital Costs	
Mine	$26,840,000
Process Plant Direct	$37,231,000
Tailings and Waste Rock Placement	$5,310,000
Owners Costs	$1,899,000
Total Capital Cost Pre-Contingency	**$71,281,000**
Contingency Costs	$17,820,000
Expansion Capital Costs	
Process Plant	$15,711,000
Tailings and Waste Rock Placement	$13,130,000
Owners Costs	$950,000
Total Expansion Capital Pre-Contingency	**$29,791,000**
Contingency Costs	$7,448,000
Total Capital Costs	**$126,339,000**

*Numbers may not sum due to rounding.

22.3.8 Sustaining Capital

Sustaining Capital costs have been applied to the Mining and Processing aspects of the project for critical spares and ongoing maintenance only. All mine development beyond the initial capital costs is considered an operating cost.

22.3.9 Operating Costs

Operating cost assumptions are outlined in Section 21. For the purpose of this PEA, it has been assumed that the mine will be operated by a contractor. Operating costs were applied in the financial model excluding IGV (IGV is the General Sales Tax in Peru).

22.3.10 Closure Costs and Salvage Value

A provision of US$10.6M was included to account for closure costs. Closure costs have been considered as part of the G&A cost and are contributed to annually.

No salvage value was considered.

22.3.11 Financing

The Cashflow, Net Present Value ("NPV") and Internal Rate of Return ("IRR") before-tax and after-tax in the PEA do not include any debt service payments.

22.3.12 Inflation

No escalation or inflation has been applied. All amounts are in real (constant) terms.

22.4 Economic Analysis

The economic analysis shows that using a base case gold price of US$1,600/oz, the Pre-Tax NPV discounted at 7% ("NPV7%") is US$327M with a 54% IRR, and the after-tax NPV7% is $189M with a 38% IRR.

Start-up CAPEX is estimated at $89M (including 25% contingency), with an after-tax payback period of 2.5 years.

Average annual production over a four-year ramp-up period is approximately 66,000 ounces of gold at 1500 tpd, with an estimated peak of 111,000 ounces in year five following an expansion to 3000 tpd.

Average gold recovery is 90.3% during the first three years, with average recovery of 86.2% over the remaining LOM.

22.4.1 Taxes and other government levies

The following outlines the main taxation considerations applied in the financial model, according to the Peruvian tax regime for mining companies:

(i) A standard corporate tax rate of 29.5% is applied to taxable income,

(ii) A special mining tax is applied to operating profit resulting from the mining activity; the effective rate is calculated based on the operating margin and ranges between 3 to 4 %; and

(iii) Workers' profit participation of 8% is applied to taxable income.

22.4.2 Cash Flow Forecasts

The cash flow forecast on an annual basis is presented in Table 22-5.

The NPV, IRR, and payback period are presented in Table 22-2.

Table 22-2: The net present value (NPV), internal rate of return (IRR), and payback period

Economics Summary	Pre-Tax (US$)	After-Tax (US$)
NPV7%	$326,737,962	$189,261,914
IRR	54%	38%
Payback	2.04	2.55
LOM Cash Flow	$580,261,428	$352,956,955

The Operating Cost Estimate per tonne and per ounce are presented in Table 22-3.

Total Operating Costs consist of mining and processing costs, tailings and waste rock disposal and on-site G&A.

Total Cash Costs consist of operating costs plus treatment and refining charges, government and NSR royalties and community interest (5%).

All-in Sustaining Costs (AISC) consist of cash costs plus sustaining capital (mining and processing).

Table 22-3: Operating Cost Estimate per tonne and per ounce

Operating Costs	LOM (US$)	$/tonne leached	$/oz Au
Mining	$406,161,345	42.10	463.5
Processing	$126,521,754	13.11	144.4
Tails and Waste Rock Placement	$35,269,644	3.66	40.3
Onsite G&A(1)	$35,183,045	3.65	40.2
Total Operating Costs	**$603,135,787**	**62.52**	**688.4**
Treatment & Refining Charges	$4,205,762	0.44	4.8
Government Royalty	$35,048,017	3.63	40.0
Royalties(2)	$40,655,699	4.21	46.4
Community Participation	$11,000,000	1.14	12.6
Total Cash Costs	**$694,045,265**	**71.94**	**792.1**
Sustaining Costs	$1,275,000	0.13	1.5
All-in Sustaining Costs (AISC)	**$695,320,265**	**72.08**	**793.6**

The Production and Cost Profile by Year are presented in Table 22-4 and Figure 22-1.

Table 22-4: Production and Cost Profile by Year

Year	0	1	2	3	4	5	6	7	8	9	10	11
Gold Production (Ozs)	**0**	**54,616**	**68,027**	**73,424**	**69,907**	**110,690**	**89,792**	**83,977**	**85,068**	**81,475**	**90,533**	**68,692**
Total Cash Costs (US$)	$ 2,683,054	$ 60,577,695	$ 46,359,158	$ 51,607,272	$ 66,417,741	$ 75,169,248	$ 66,768,822	$ 82,331,914	$ 65,295,128	$ 64,280,517	$ 64,574,855	$ 47,979,861
Total Cash Costs (US$ / Oz)	-	**$ 1,109**	**$ 681**	**$ 703**	**$ 950**	**$ 679**	**$ 744**	**$ 980**	**$ 768**	**$ 789**	**$ 713**	**$ 698**
Total AISC Costs (US$)	$ 2,683,054	$ 60,577,695	$ 46,284,158	$ 51,532,272	$ 66,342,741	$ 75,019,248	$ 66,618,822	$ 82,181,914	$ 65,145,128	$ 64,130,517	$ 64,424,855	$ 47,829,861
Total AISC Costs (US$ / Oz)	-	**$ 1,109**	**$ 680**	**$ 702**	**$ 949**	**$ 678**	**$ 742**	**$ 979**	**$ 766**	**$ 787**	**$ 712**	**$ 696**

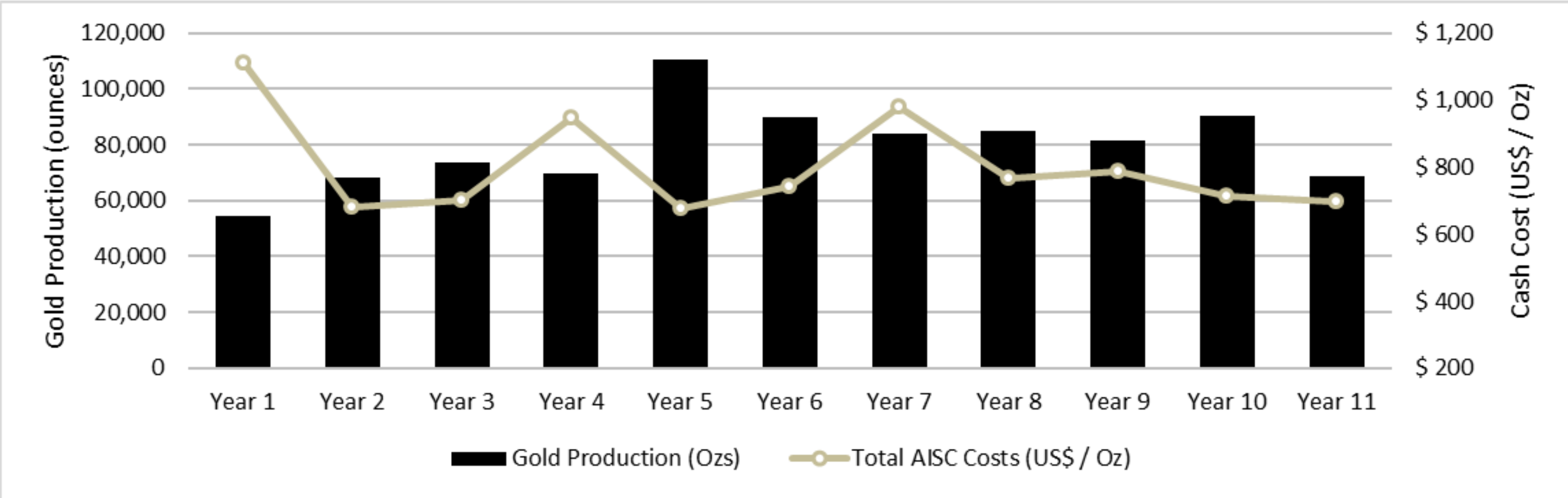

Figure 22-1: Production and Cost Profile By Year

Table 22-5: Cash Flow Forecasts

	Units/Year	0	1	2	3	4	5	6	7	8	9	10	11	Total
Mine Production														
Total Mineralized Material	Tonnes		449,920	542,594	539,631	821,254	1,088,230	1,084,916	1,082,080	1,045,761	1,101,030	1,084,974	806,776	**9,647,166**
Gold Grade	g/t		4.18	4.32	4.68	2.99	3.67	2.99	2.80	2.83	2.71	3.01	3.09	
Contained Metal	Ounces		60,483	75,334	81,311	78,909	128,411	104,167	97,421	98,687	94,518	105,027	79,689	**1,003,957**
Mining Rate tpd	tpd		1,250	1,507	1,499	2,281	3,023	3,014	3,006	2,905	3,058	3,014	2,241	**26,798**
Waste Tonnes	Tonnes	410,028	850,408	191,088	494,782	282,219	194,727	35,760	423,781	32,280	18,211	0	5,013	**2,938,297**
Plant Production														
Recovery	%		90.3%	90.3%	90.3%	88.6%	86.2%	86.2%	86.2%	86.2%	86.2%	86.2%	86.2%	**87.3%**
Gold price	USD/oz		1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	**1,600**
Royalty	%		2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	2.5%	
Processing Cost	USD/Ton		14.08	14.08	14.08	13.57	12.86	12.86	12.86	12.86	12.86	12.86	12.86	
Cash Operating Cost	USD/oz		-249	-611	-630	-837	-581	-637	-841	-658	-676	-611	-598	
Gold Produce Au	**Oz**		**54,616**	**68,027**	**73,424**	**69,907**	**110,690**	**89,792**	**83,977**	**85,068**	**81,475**	**90,533**	**68,692**	**876,200**
Cash Flow														
Initial Cash Flow														
Net Revenue	**USD**		**87,385,760**	**108,842,563**	**117,478,133**	**111,850,877**	**177,104,451**	**143,667,126**	**134,363,043**	**136,109,110**	**130,359,226**	**144,853,238**	**109,907,144**	**1,401,920,672**
Costs														
Refining Cost (0.3% gross)	USD		-262,157	-326,528	-352,434	-335,553	-531,313	-431,001	-403,089	-408,327	-391,078	-434,560	-329,721	**-4,205,762**
Government Royalty	USD		-2,184,644	-2,721,064	-2,936,953	-2,796,272	-4,427,611	-3,591,678	-3,359,076	-3,402,728	-3,258,981	-3,621,331	-2,747,679	**-35,048,017**
Mining Cost (USD/yr)	USD		-40,219,836	-21,167,120	-32,118,857	-36,554,470	-36,452,837	-29,966,530	-46,036,020	-29,475,610	-28,516,677	-28,100,827	-19,792,003	**-348,400,787**
Paste Opex ($4.50/tonne)	USD		0	-1,691,053	-1,681,818	-2,559,527	-3,391,586	-3,381,258	-3,372,419	-3,259,227	-3,431,479	-3,381,438	-2,514,404	**-28,664,208**
Mine Supervision	USD	-829,405	-829,405	-829,405	-829,405	-893,646	-983,583	-983,583	-983,583	-983,583	-983,583	-983,583	-983,583	**-11,096,349**
Stope Definition Drilling	USD		-2,000,000	-2,000,000	-2,000,000	-2,000,000	-2,000,000	-2,000,000	-2,000,000	-2,000,000	-1,000,000	-1,000,000	0	**-18,000,000**
Processing Costs (less G&A)	USD		-6,335,238	-7,640,163	-7,598,442	-11,146,348	-13,995,193	-13,952,573	-13,916,101	-13,449,020	-14,159,807	-13,953,319	-10,375,551	**-126,521,754**
Tails and Waste Rock Placement	USD		-2,837,106	-3,421,490	2,757,678	-3,215,049	-4,260,208	-4,247,235	-4,236,132	-4,093,950	-4,310,318	-4,247,462	-3,158,371	**-35,269,644**
General and Administration	USD	-1,853,649	-2,375,121	-2,405,901	-2,440,175	-2,673,202	-2,990,887	-3,048,617	-3,128,965	-3,275,518	-3,448,177	-3,651,592	-3,891,241	**-35,183,045**

	Units/Year	0	1	2	3	4	5	6	7	8	9	10	11	Total
Ollachea Community Participation	USD		-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	**-11,000,000**
Royalties Mcquarie & Sherpa & RioTinto	USD		-2,534,187	-3,156,434	-3,406,866	-3,243,675	-5,136,029	-4,166,347	-3,896,528	-3,947,164	-3,780,418	-4,200,744	-3,187,307	**-40,655,699**
Sustaining Costs														
Mining	USD			-25,000	-25,000	-25,000	-50,000	-50,000	-50,000	-50,000	-50,000	-50,000	-50,000	**-425,000**
Process	USD			-50,000	-50,000	-50,000	-100,000	-100,000	-100,000	-100,000	-100,000	-100,000	-100,000	**-850,000**
Net Operating	**USD**	**-2,683,054**	**26,808,065**	**62,408,405**	**65,795,861**	**45,358,136**	**101,785,203**	**76,748,304**	**51,881,129**	**70,663,983**	**65,928,709**	**80,128,383**	**61,777,283**	**706,600,407**
CAPEX														
PROCESS CAPEX	**USD**													
Directs	USD	-26,069,144	-8,689,715		-14,710,868									
EPCM	USD	-1,854,330	-618,110		-1,000,000									
Owners Cost	USD	-1,424,508	-474,836		-949,672									
TAILINGS CAPEX	**USD**													
Directs	USD	-1,593,000	-3,717,000	-13,130,000										
MINE CAPEX	**USD**													
Pre-Prod Dev	USD	-17,318,400	-8,021,600											
Paste Infrastructureplace	USD	-1,500,000	0											
Contingency	USD	-12,439,846	-5,380,315	-3,282,500	-4,165,135									
Sub Total		**-62,199,228**	**-26,901,576**	**-16,412,500**	**-20,825,676**									**-126,338,979**
Cash Flow Pre-Tax	**USD**	**-64,882,281**	**-93,511**	**45,995,905**	**44,970,186**	**45,358,136**	**101,785,203**	**76,748,304**	**51,881,129**	**70,663,983**	**65,928,709**	**80,128,383**	**61,777,283**	**580,261,428**
P&L														
Revenues		0	87,385,760	108,842,563	117,478,133	111,850,877	177,104,451	143,667,126	134,363,043	136,109,110	130,359,226	144,853,238	109,907,144	**1,401,920,672**
Costs		-829,405	-54,668,387	-39,796,823	-44,760,231	-59,500,864	-66,042,332	-58,553,859	-74,306,420	-57,072,445	-56,051,922	-55,722,520	-39,901,312	**-607,206,521**
Gross Margin		**-829,405**	**32,717,373**	**69,045,740**	**72,717,902**	**52,350,013**	**111,062,119**	**85,113,268**	**60,056,623**	**79,036,665**	**74,307,303**	**89,130,719**	**70,005,831**	**794,714,151**
G&A		-1,853,649	-2,375,121	-2,405,901	-2,440,175	-2,673,202	-2,990,887	-3,048,617	-3,128,965	-3,275,518	-3,448,177	-3,651,592	-3,891,241	**-35,183,045**
Sustaining Capital Costs		0	0	-75,000	-75,000	-75,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	-150,000	**-1,275,000**
Ollachea Community Participation		0	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	-1,000,000	**-11,000,000**
Special Mining Tax (SMT)		0	-824,147	-2,524,774	-2,629,448	-1,553,531	-4,125,150	-2,996,206	-1,743,986	-2,716,646	-2,504,231	-3,185,656	-2,478,083	**-27,281,858**

	Units/Year	0	1	2	3	4	5	6	7	8	9	10	11	Total
Workers' Profit Participation		0	-2,072,713	-3,269,716	-3,464,060	-2,055,309	-6,603,057	-4,923,732	-3,267,847	-4,925,974	-4,797,457	-6,112,228	-4,755,936	**-46,248,030**
Operating Margin		**-2,683,054**	**26,445,391**	**59,770,349**	**63,109,219**	**44,992,971**	**96,193,025**	**72,994,712**	**50,765,824**	**66,968,527**	**62,407,438**	**75,031,243**	**57,730,571**	**673,726,218**
Royalties		0	-2,534,187	-3,156,434	-3,406,866	-3,243,675	-5,136,029	-4,166,347	-3,896,528	-3,947,164	-3,780,418	-4,200,744	-3,187,307	**-40,655,699**
Interest Expenses		0	0	-19,012,184	-19,865,661	-18,038,238	-15,121,843	-12,205,449	-9,289,055	-6,372,660	-3,456,266	-539,871	0	**-103,901,227**
Income Tax		0	-5,513,637	-11,092,511	-11,751,824	-6,972,637	-22,400,870	-16,703,760	-11,086,171	-16,711,367	-16,275,373	-20,735,735	-16,134,513	**-155,378,399**
Net Profit		**-2,683,054**	**18,397,567**	**26,509,221**	**28,084,868**	**16,738,421**	**53,534,283**	**39,919,156**	**26,494,070**	**39,937,335**	**38,895,382**	**49,554,893**	**38,408,751**	**373,790,893**
Cash Flow from Investment Activities														
CAPEX	USD	-62,199,228	-26,901,576	-16,412,500	-20,825,676	0	0	0	0	0	0	0	0	-126,338,979
Project Capital Costs – IGV/GST only	USD	1,119,586	484,228	0	0	0	0	0	0	0	0	0	0	1,603,814
Project Cash Flow After Tax	**USD**	**-63,762,695**	**-8,019,780**	**29,108,904**	**27,124,853**	**34,776,659**	**68,656,126**	**52,124,605**	**35,783,125**	**46,309,995**	**42,351,648**	**50,094,764**	**38,408,751**	**352,956,955**

22.5 Sensitivity analysis

Sensitivities of pre-tax and post-tax NPV and IRR to gold prices per ounce are presented in Table 22-6.

Table 22-6: Economic Sensitivity to Gold Price

Gold Price ($/oz)	US$1400	US$1600	US$1800
Pre-Tax $NPV_{7\%}$	$223M	$327M	$430M
Pre-Tax IRR	40%	54%	68%
Pre-Tax Payback	2.5 years	2 years	1.7 years
After-Tax $NPV_{7\%}$	$125M	$189M	$253M
After-Tax IRR	28%	38%	47%
After-Tax Payback	3 years	2.5 years	2.2 years

The After-Tax Economic Sensitivity to discount rate is shown in Table 22-7.

Table 22-7: After Tax Economic Sensitivity to discount rate

Parameter	Unit	Amount (US$)
Net Cash Flow before tax		
NPV @ 5% real (before tax)	US$	$383,278,808
NPV @ 7% real (before tax)	US$	$326,737,962
NPV @ 10% real (before tax)	US$	$258,572,798
IRR (before tax)	%	54%
Payback (before tax)	Years	2.04
Net Cash Flow after tax		
NPV @ 5% real (after tax)	US$	$225,733,462
NPV @ 7% real (after tax)	US$	$189,261,914
NPV @ 10% real (after tax)	US$	$145,356,478
IRR (after tax)	%	38%
Payback (after tax)	Years	2.55

The After-Tax Economic Sensitivity to Gold Price, Operating and Capital Costs is shown in Figure 22-2.

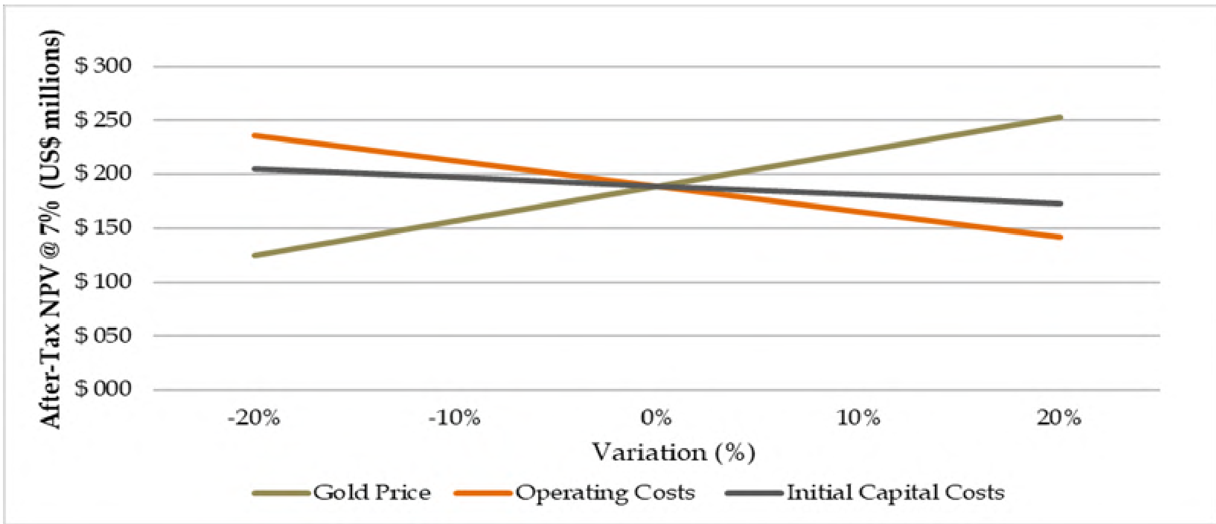

Figure 22-2: After-Tax Economic Sensitivity to Gold Price, Operating and Capital Costs

23 ADJACENT PROPERTIES

There are no properties adjacent to the Ollachea Property that are of relevance to this Technical Report.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data and information.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineral Resources

Best practices in geological data capture, storage and interpretation were implemented early at the Ollachea Property and have been maintained diligently. Throughout the project, the QAQC results have demonstrated the reliability of the sampling and assaying procedures, and the Mineral Resources have been estimated by independent Qualified Persons.

The Qualified Persons undertook a review of the database underpinning the Mineral Resource Estimate ("MRE") and reviewed the site procedures. Some minor issues were identified; however, these were not considered to be material for the MRE.

Experimentation by the Qualified Persons with various interpretation scenarios during the 3D modelling, showed that the tonnes and grade estimates were not particularly sensitive to these aspects. This was due to the good correlation between drillhole sections and consistent geometry and orientation of the mineralization, which left little room for deviation. Similarly, scenario testing various variographic, search and estimation parameters did not have a material impact on the global tonnes and grade estimates.

Technically, the risks to the MRE are mainly related to the uncertainty in the mining depletion as the QPs understand that there is no way to safely undertake a comprehensive survey of the artisanal underground workings at this time. The depletion could be material, although the current extent of the artisanal workings is unknown.

25.2 Mining and Mine Plan

Edgard Vilela (QP) considers that long hole open stoping ("LHOS") with paste fill is the optimal mining method for the mineralization reported at the Property. Edgard Vilela (QP) notes that mineralization reported at the Property has good continuity along strike, and that he has seen LHOS successfully applied to numerous mines with mineralization with a similar geometry.

Stopes will be accessed longitudinally (along strike) on each level by, one, two or three strike drives, dependent on lode thickness.

The direction of mining for the deposit will be from the bottom up. In general, as each mining level is completed, the next level will start using the backfilled stope voids as the mining platform.

To achieve a reasonable mining rate, the mine will be split into multiple mining panels that can be mined simultaneously (as is common practice). The lowest level of each of these mining panels requires an artificial sill pillar to be created using high strength paste fill. This high strength sill pillar allows mineralization located directly beneath it to be completely extracted. This artificial pillar is to contain a higher cement content than ordinary paste fill to be applied to the remaining stopes in the mine plan, in accordance with the geotechnical and backfill assessments.

This study has indicated that there is a defined area where the mineralized material is amenable to a higher cut-off grade. The mineralized material can be mined at an elevated cut-off grade in the first 3 – 4 years without breaking it up into isolated stopes (which are significantly less economic to mine).

The update block model with a high-grade domain has facilitated scheduling of high-grade material earlier in the proposed mine life than the 2012 Feasibility Study ("2012 FS").

The grades are higher than the 2012 FS in the early years of the schedule primarily because the mine plan is focused on the high-grade zone only where a cut-off grade of 3 g/t has been applied. With the 1500 tpd mining rate, the mine plan can source the majority of the material required from only the high-grade zone, allowing the mine plan to have consistently higher grades.

In year four, the processing facility increase its capacity to 3000 tpd. In line with the proposed ramp up and increased production volume, the cut-off grade for the material outside of the high-grade zone is set at 2.1 g/t Au, which results in a decrease in mined grade, but a rise in total ounces per year due to the increased volume of material mined.

The revised mine plan presented offers an opportunity for a low start-up CAPEX, whilst still maintaining reasonable revenues.

Significant opportunity still exists with respect to Minapampa Far East, and the inclusion of that material in the mine plan and financial model. Further work will need to be completed with respect to waste storage options to increase the mine life significantly, but the mineralized material is present (the inferred resource in Minapampa Far East), and it is a direct extension of the Minapampa area.

It will be important to survey the existing workings of the artisanal miners to understand the extent of their activities once formal mining begins, and those artisanal workings are abandoned. The crown pillar and the resource in the vicinity of the crown pillar will need to be re-assessed at that point.

25.3 Metallurgy and Mineral Process Design

Using the results of the two gravity concentration tests reported by Met-Solve 2017 and 2021, with head grades of 3.29 and 4.35 g/t Au, respectively, with CIL leaching of all the tailings from the re-grind circuit, predicted overall recoveries of gold are presented in Table 25-1.

Table 25-1: Summary of Overall Gold Recovery

Head Grade g/t Au	3.29	4.35
Gold Recovery	86.2 %	90.3 %

The assumptions used are:

- Recovery of gold from high-grade concentrates using a shaking table is 50%
- Tailings grade after recovery of a high mass pull concentrate (15%) is 0.4 g/t Au
- Tailings grade after CIL leaching (Ammtec 2013) is 0.3 g/t Au
- Overall process losses in smelting, solution losses in CIL is 1.0 %.

Test work on several samples taken from the Minapampa high-grade zone has shown that high recoveries of gold can be obtained to a gravity concentrate with approximately 15% of the feed mass, significantly reducing the size of downstream processing. Tests on samples from other zones of the mineralized material are needed, together with leaching tests on this concentrate, as extensive leach results are available but on higher grade gravity concentrates and gravity concentration tailings.

25.4 Project Infrastructure

Infrastructure such as workshops, laboratory, administrative offices etc. are straightforward. Confirmation of the availability of accommodation in the town is expected to be adequate but should be confirmed. Power supply requires only a short connecting line and a proposal for this and the substation indicate little risk. Water supply should be more than adequate from the mine, but a permit to extract water from the river would give added security.

25.5 Tailings and Waste Rock Management

The Ollachea mine waste management concept has been developed to minimize the impacts of tailings and waste rock materials. The concept includes the following key aspects:

- 43% of tailings to be returned to the mine as paste backfill.
- Remaining 57% of tailings to be filtered to a low moisture content and stacked in a system of co-disposed mine waste rock and filtered tailings product.
- Co-disposal will occur at two locations: the Lower Portal Co-Disposal Facility ("Lower Portal CDF") and the Cuncurchaca Co-Disposal Facility ("Cuncurchaca CDF").

25.6 Operating and Capital Cost Estimates

25.6.1 Operating Cost Estimates

Power and labour costs are well defined but recent increases in transport costs may affect the cost of reagents, especially sodium cyanide which is imported. Grinding balls and lime, the other two main consumable items, are manufactured in Peru.

Paste fill estimates could be optimized with the addition of potential co-disposal and optimization of the density and cement content.

25.6.2 Capital Cost Estimates

Quotes have been obtained for all major equipment but again, the cost of transport may affect the installed cost. Piping and electrical installation has been estimated by factoring, and detailed design will be necessary to improve accuracy of these costs.

The capital costs for the mining were established with the assistance of a contractor. The capital cost estimates for the mining are significantly lower than the 2012 FS primarily due to the use of a contractor rather than being an owner operator. As such, much of what was formally CAPEX has been moved to OPEX.

25.7 Financial Analysis

The financial analysis has shown that the project offers strong project economics with a low upfront CAPEX requirement, and a quick payback period.

The economic analysis shows that using a base case gold price of US$1,600/oz, the Pre-Tax Net Present Value discounted at 7% (“NPV7%”) is US$327M with a 54% Internal Rate of Return (“IRR”), and the after-tax NPV7% is $189M with a 38% IRR.

Start-up CAPEX is estimated at $89M (including 25% contingency), with an after-tax payback period of 2.5 years.

26 RECOMMENDATIONS

26.1 Geology and Mineral Resources

The Company plans to conduct additional exploration activities in order to add to the existing Mineral Resource, although there is no timeline placed on any exploration work or any update to the Mineral Resource Estimate ("MRE") at present. The QPs recommend a survey of artisanal workings be completed; however the cost of this potential work is unknown.

26.2 Mining and Mine Plan

Further work should be completed to optimise the mine plan, making minor modifications to cut-off grade early in the mine life to maximise the ounces produced. This should then be followed by a redesign of the stopes that remain in the high-grade zone after the ramp up to 3000 tpd to a cut-off grade of 2.1 grams per tonne maximising the resource recovery (increase of the ounces in the mine plan).

A revised ventilation study should be completed to understand the potential for two return air-raise versus existing three return raises (potential reduction in vertical development meters). Also, a single fresh air intake raise versus existing two fresh air raises (potential reduction in vertical development meters).

The vertical development needs to be optimized using VentSim considering the decreased production rate in the early years (optimization and potential deferral).

It is likely that not all stopes will need to be paste filled, and there may be opportunities in non-critical areas of the mine to use rock fill in place of the paste fill, reducing cost. This should be investigated further.

It is recommended that the location of the upper portal location be assessed as it is currently placed in areas with significant artisanal mining activity.

The vast majority of the above work is desktop work. Dependent on what aspects are advanced, and the level of detail of the work completed, the cost would be in the vicinity of US$200,000. This could then lead onto a PFS or FS level study to define reserves.

26.3 Metallurgy and Mineral Process Design

Gravity concentration tests on samples from other zones of the mineralized material are needed, together with leaching tests on the concentrates produced.

A budget estimate of US$300,000 should be allocated to source and test sufficient samples, although there is further work to be completed to define the drilling and sampling locations.

26.4 Project Infrastructure

Confirmation of the availability of accommodation in the town is expected to be adequate but should be confirmed. Water supply should be adequate from the mine, but a permit to extract water from the river would give added security.

It is recommended that the location of the paste plant be assessed as it is currently placed in areas with significant artisanal mining activity.

The cut and fill volumes for the platforms could be optimized to reduce costs potentially and could be looked at to understand if that aspect can be optimized.

26.5 Tailings and Waste Rock Management

An opportunity exists to eliminate the imported clay / geosynthetic clay liner at the Lower Portal Co-Disposal Facility ("LPCDF"). This is contingent upon demonstrating that the filtered tailings will act as a low-permeability element, as for the Cuncurchaca Co-Disposal Facility ("CCDF"). Further, the concept would need to be presented to regulators for approval. This could reduce the time and cost associated with constructing the LPCDF, as well as simplifying operation.

A further opportunity exists to increase the placement of tailings solids as underground paste backfill. For this PEA, relatively conservative values were used for the solids content of the backfill mix. This would reduce the required storage on-surface.

It is recommended that the mixed placement of the filtered tailings together with the waste rock be planned in detail, prior to beginning operation.

Further information on the geotechnical characteristics of the waste rock should be determined, as inputs to stability and seepage analyses.

Stability analyses must be done on the Co-Disposal Facilities, in order to confirm that assumed design slopes are safely achievable.

The proposed contingency for off-spec tailings to be discharged into geotextile geotubes, should be trial-tested at site, prior to full commissioning, using smaller, test-size geotubes, to confirm the type of geotextile and flocculant, if required.

The cost estimate to investigate these specific opportunities is US$100,000.

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- CIM, 2019. CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. Prepared by the CIM Mineral Resource and Mineral Reserve Committee. Adopted by the CIM Council on November 29, 2019. https://mrmr.cim.org/media/1129/cim-mrmr-bp-guidelines_2019.pdf

28 CERTIFICATES OF QUALIFIED PERSONS

I, Mr. Edgard Vilela, AusIMM CP(Min) do hereby certify that I am the author of Sections 1, 2, 3, 4, 5, 15, 16, 19, 20, 21.1 to 21.4, 21.9 to 21.11, 21.14 to 21.18, 22, 23, 24, 25.2, 25.6, 25.7, 26,2 and 27 of the Technical Report titled "Ollachea Gold Project - NI 43-101 Technical Report (Preliminary Economic Assessment)" with the effective date of August 27th, 2021:

1. My current address is: 5957 Republica de Panamá Av, Miraflores, Lima, Perú.
2. I am currently a full-time employee of Mining Plus.
3. I am a graduate of the Pontifical Catholic University of Peru and received a Bachelor of Science Degree in Mining in 2000.
5. I am Chartered Professional in the discipline of Mining and a registered member in good standing of the Australasian Institute of Mining and Metallurgy. MAusIMM CP (Mining) membership number 992615.
6. I have practiced my profession continuously since January 2001. My relevant experience includes over 21 years' experience working in relevant undergrounds mines in Perú as Volcan, Pan American Silver and Fortuna Silver developing positions since Operation Supervisor, Planning Superintendent, Technical Manager and Projects and Reserves Manager. Also, I have worked as mining consultant evaluating projects in Perú and South America in all their levels of study: Scoping, PFS and FS.
7. I have read the definition of "qualified person" set out in National Instrument 43-

   101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
8. I am responsible for Sections 1, 2, 3, 4, 5, 15, 16, 19, 20, 21.1 to 21.4, 21.9 to 21.11, 21.14 to 21.18, 22, 23, 24, 25.2, 25.6, 25.7 and 26,2 of the Technical Report.
9. As of the effective date of the Technical Report, August 27th, 2021, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
10. I have read N1 43-101 and Form 43101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. Applying all the tests in section 1.5 of N1 43-101, I am independent of Minera IRL and all its affiliates.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
13. I visited the Ollachea Property for the purposes of the report between 19th and the 20th of May 2021.

Dated August 27th, 2021.

Edgard Vilela B.Sc., MAusIMM, CP(Mining)

I, Dr. John Alan Thomas, P.Eng do hereby certify that I am the author of Sections 13, 17, 18.1 to 18.4, 21.5, 21.6, 21.12, 25.3, 25.4, 26.3 and 26.4 of the Technical Report titled "Ollachea Gold Project - NI 43-101 Technical Report (Preliminary Economic Assessment)" with the effective date of August 27th, 2021:

1. My current address is: 5766 Goldenrod Crescent, Delta, BC V4L 2G6, Canada.
2. I am currently a full time employee of JAT Metconsult Ltd.
3. I graduated with Chemical Engineering degrees (B.Sc. M.Sc. and Ph.D.) from the University of Manchester in 1969, 1971 and 1973 respectively.
5. I am a member of the self-regulating Association of Professional Engineers and GeoScientists of British Columbia (#125986)
6. I have practiced my profession for over 48 years. My relevant experience includes 48 years in the mining industry, and includes, process development, engineering, construction management and operation.
7. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
8. I am responsible for Sections 13, 17, 18.1 to 18.4, 21.5, 21.6, 21.12, 25.3, 25.4, 26.3 and 26.4.
9. As of the effective date of the Technical Report, August 27th, 2021, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
10. I have read N1 43-101 and Form 43101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. Applying all the tests in section 1.5 of N1 43-101, I am independent of Minera IRL and all its affiliates.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
13. I have NOT visited the Ollachea Property for the purposes of the report.

Dated August 27th, 2021.

J. A. Thomas

John Alan Thomas Ph.D, P.Eng

PROFESSIONAL
PROVINCE OF
Dr. J. A. THOMAS
34665
BRITISH COLUMBIA
ENGINEER

I, Mr. Donald (Don) Hickson, P.Eng. do hereby certify that I am the author of Sections 18.5, 21.7, 21.8, 21.13, 25.5, and 26.5 of the Technical Report titled “Ollachea Gold Project - NI 43-101 Technical Report (Preliminary Economic Assessment)” with the effective date of August 27th, 2021:

1. My current address is: Calle German Aparicio Gomez Sanchez 320, Miraflores, Lima, Peru.
2. I am currently a full-time employee and partner in Envis.
3. I graduated from the University of Waterloo, in 1991 with a Bachelor of Applied Science degree.
5. I am a registered professional engineer (P.Eng.) of Alberta, Canada (APEGGA).
6. I have practiced my profession continuously for over 25 years. My relevant experience includes over 25 years of experience in tailings and waste management projects of similar or greater complexity than Ollachea.
7. I have read the definition of “qualified person” as set out in National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a “qualified person” for the purposes of NI 43-101.
8. I am responsible for Sections 18.5, 21.7, 21.8, 21.13, 25.5, and 26.5.
9. As of the effective date of the Technical Report, August 27th, 2021, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
10. I have read N1 43-101 and Form 43101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. Applying all the tests in section 1.5 of N1 43-101, I am independent of Minera IRL and all its affiliates.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
13. I visited the site between the 16th and 20th of January 2012.

Dated August 27th, 2021.

Don Hickson B.Sc., P.Eng.

I, Dr. Andrew Fowler, MAusIMM CP(Geo) do hereby certify that I am the author of Sections 10.3, 11.2, 12.2, 14, 25.1 and 26.1 of the Technical Report titled "Ollachea Gold Project - NI 43-101 Technical Report (Preliminary Economic Assessment)" with the effective date of August 27th, 2021:

1. I am currently employed as a Principal Geologist with Mining Plus, Lv 17, 127 Creek Street, Brisbane, Queensland, Australia;

2. This certificate applies to the Technical Report titled "Ollachea Gold Project - NI 43-101 Technical Report (Preliminary Economic Assessment)" (the "Technical Report") prepared for Mineral IRL SA ("the Issuer"), which has an effective date of August 27th, 2021 – the date of the most recent technical information;

3. I am a graduate of the University of Melbourne (Ph.D., 2004). I am a Chartered Professional in the discipline of Geology and a registered member in good standing of the Australasian Institute of Mining and Metallurgy (#301401). I have practiced my profession continuously since November 2004. My relevant experience includes two years as Exploration Geologist with a junior greenfields explorer, Mithril Resources, two years as Project Geologist/Head Geologist with the Costerfield gold-antimony mine operated by AGD Operations, eight years as a Senior Geologist with AMC Consultants Pty Ltd, 1 year as Manager of Mineral Resources at MMG Las Bambas and three years as Principal Geologist at Mining Plus. I have read the definition of a "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101;

4. I completed a personal inspection of the Property from the 31st August to the 2nd September, 2016;

5. I am responsible for Items 10.3, 11.2, 12.2, 14, 25.1 and 26.1 of this technical report;

6. I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

7. I have not had prior involvement with the property that is the subject of the Technical Report;

8. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

9. As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading;

Effective Date: August 27th, 2021
Signing Date: August 27th, 2021

Andrew Fowler, Ph.D., MAusIMM, CP(Geo)

I, Mr. Doug Corley, MAIG R.P. Geo (Mining) do hereby certify that I am the author of Sections 10.1, 10.2, 11.1, and 12.1 of the Technical Report titled "Ollachea Gold Project - NI 43-101 Technical Report (Preliminary Economic Assessment)" with the effective date of August 27th, 2021:

1. My current address is: C/O Mining Plus, Level 17, 127 Creek Street, Brisbane QLD 4000, Qld.
2. At the time of the site visits and technical work performed I was a full time employee of Coffey Mining (2010) and GHD (2014).
3. I am a graduate of James Cook University, Townsville and received a Bachelor of Science degree (with Honours) in Geology 1991.
5. I am Chartered Professional in the discipline of Geology and a registered Member in good standing of the Australasian Institute of Geoscientists. MAIG R.P Geo (Mining) membership number 10109.
6. I have practiced my profession continuously since 1991. My relevant experience includes over 30 years' experience working for a variety of major to junior mining companies and consultancies focused on precious and base metals in a variety of geological settings around the world.
7. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
8. I am responsible for Sections 10.1, 10.2, 11.1, and 12.1.
9. As of the effective date of the Technical Report, August 27th, 2021, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
10. I have read N1 43-101 and Form 43101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. Applying all the tests in section 1.5 of N1 43-101, I am independent of Minera IRL and all its affiliates.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
13. I visited the Ollachea Property for the purposes of the report on two occasions. Between the 21st and 22nd of June 2010, and between 13th and 22nd of January 2014.

Dated August 27th, 2021.

Doug Corley B.Sc.(Hons), MAIG, R.P. Geo(Mining)

I, Mr. David Seers, AusIMM CP(Geo) do hereby certify that I am the author of Sections 6, 7, 8, and 9 of the Technical Report titled "Ollachea Gold Project - NI 43-101 Technical Report (Preliminary Economic Assessment)" with the effective date of August 27th, 2021:

1. My current address is: 6 Margaret Terrace, Pocklington, York, UK.
2. I am currently contracted to Mining Plus on a casual basis.
3. I am a graduate of the University of Leicester and received an undergraduate Master's degree (MGeol) with Honours in Applied Geology in 2003.
5. I am Chartered Professional in the discipline of Geology and a registered member in good standing of the Australasian Institute of Mining and Metallurgy. MAusIMM CP (Geo) membership number 991014.
6. I have practiced my profession continuously since September 2003. My relevant experience includes over 16 years' experience working for junior explorers focused on precious and base metal exploration, including orogenic deposits in southern Peru.
7. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
8. I am responsible for Sections 6, 7, 8, and 9 of the Technical Report.
9. As of the effective date of the Technical Report, August 27th, 2021, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
10. I have read N1 43-101 and Form 43101F1, and the Technical Report has been prepared in compliance with that instrument and form.
11. Applying all the tests in section 1.5 of N1 43-101, I am independent of Minera IRL and all its affiliates.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
13. I have NOT visited the Ollachea Property for the purposes of the report.

Dated August 27th, 2021.

David Seers MGeol, MAusIMM, CP(Geo)

CHARTERED PROFESSIONAL
AusIMM
THE MINERALS INSTITUTE
• GEOLOGY •
DAVID SEERS