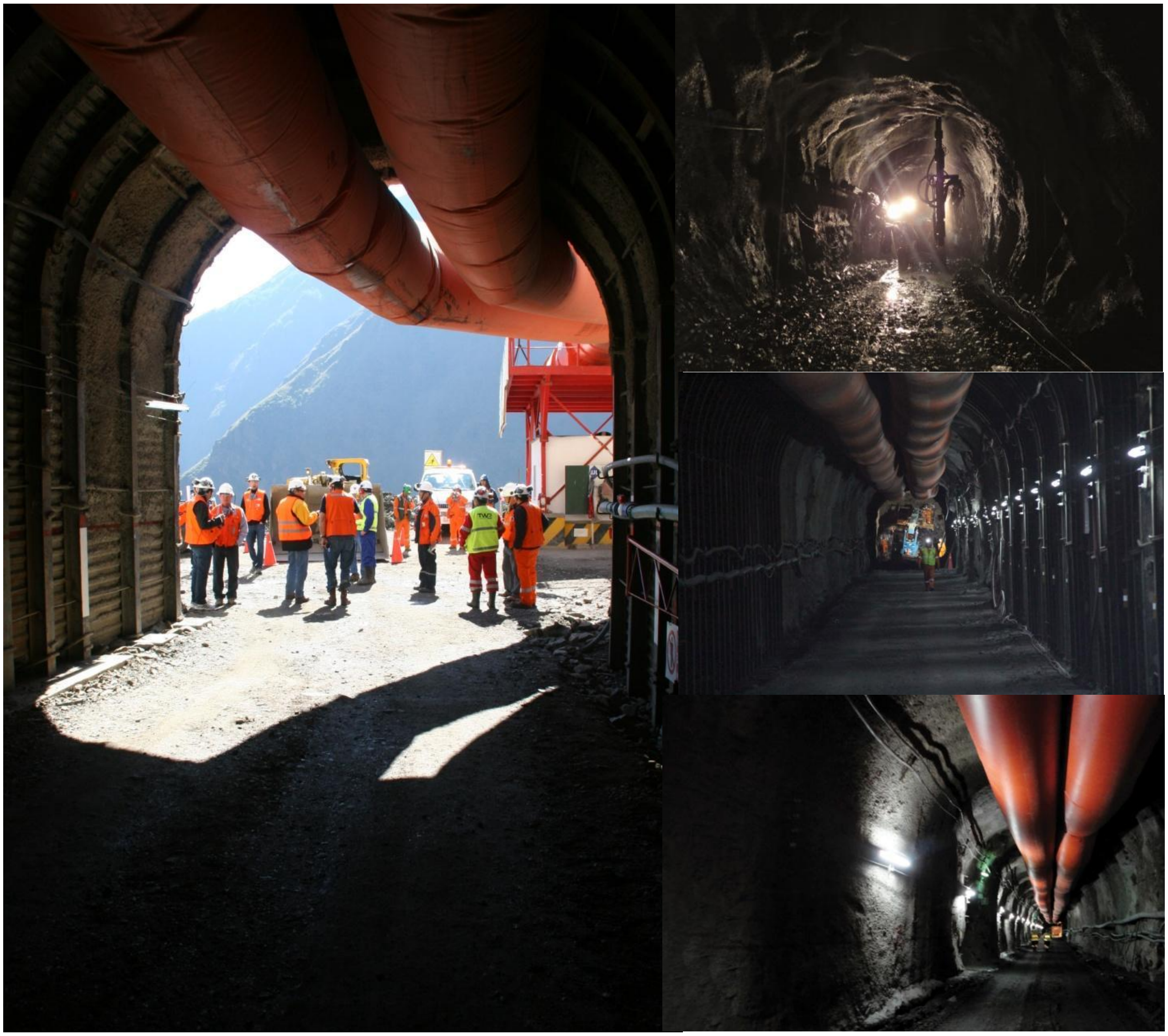
Minera Kuri Kullu S.A.
(Subsidiaria de Minera IRL Ltd)

amec

Ollachea Gold Project
PERU
NI 43-101 Technical Report on Feasibility Study



Prepared for:
Minera Kuri Kullu S.A.

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IMPORTANT NOTICE

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1 SUMMARY

The Ollachea Gold Project (the Project) is located in the Puno Region of southern Peru. Compañía Minera Kuri Kullu S.A. (MKK), a wholly owned subsidiary of Minera IRL S.A. (IRL) and which is a wholly owned subsidiary of Minera IRL Limited (MIRL), currently owns the Project and retained AMEC Peru S.A. (AMEC) and Coffey Mining Pty Ltd (Coffey Mining) to conduct a Feasibility Study (FS) on the viability of mining the deposit from underground and processing ore in a 1.1 million t/a facility to produce gold doré. Process plant design and project estimating were carried out by AMEC in Brisbane, Australia.

1.1 Property, Access and Permits

The Ollachea Project consists of 12 concessions covering an area of 8,698.98 ha (Table 4-1). map of the Ollachea Property is shown in Figure 4-2. The concessions are map-staked and defined and registered spatially by the location of their vertices.

Legal opinion provided supports MKK's interpretation that the Ollachea Property is in good standing, valid and in full force and effect, therefore giving MKK the right to explore and exploit the minerals existing in the titled area.

The mineralization included in the Mineral Resource and Mineral Reserves discussed in this Report occur within the Oyaechea 3 concession. The proposed plant site location will be located on the Oyaechea 2 concession. The portal location for the exploration access adit (currently being developed), which will serve as the main mine portal is located on the Oyaechea 2 concession. The Tailings Storage Facility is located approximately 2.5 km north of the mine portal and within the Oyaechea 9 concession.

A gap measuring approximately 3,000 m long by 130 m wide exists between the Oyaechea 2 and Oyaechea 3 concessions. This concession is not held by MKK. The proposed exploration drive and other mine infrastructure discussed in this report have been located to avoid this gap.

MKK has signed a 30 year surface rights agreement in June 2012 with the Community of Ollachea allowing MKK to use the property covering the area of interest of the Project. The agreement allows the Community of Ollachea to carry out artisanal mining activities on the property until MKK commences production.

MKK currently holds permits which allow it to continue exploration activities and to develop an exploration access drive as part of their exploration program.

The property is crossed by the Southern Interoceanic Highway providing year-round highway access to the provincial capital of Puno, the airports in Juliaca and Puerto Maldonado (each with daily scheduled flights to Lima), the Pan American Highway and the deepwater port of Matarani, located at Ilo on the Pacific coast of Peru. The location, access, climate and elevation of the Project will allow production activities to be carried out year-round.

The property position and surface rights are sufficient to allow MKK to continue to explore and develop the Project.

1.2 Geology and Mineral Resources

The Ollachea gold deposit is an orogenic or mesothermal-style gold deposit hosted in Devonian-aged carbonaceous metasediments on the eastern flank of the Cordillera Oriental of the Peruvian Andes. Gold mineralization is contained within seven discrete west-striking, north-dipping structures below the Minapampa Zone on the north side of the Oscco Cachi River, a narrow creek, and approximately 1,000 m west of the town of Ollachea.

Gold mineralization occurs within a shear zone in carbonaceous sediments cut by seven discrete, continuous packages of quartz-carbonate-sulphide veins and veinlets. Metasediments in the shear zone are characterized by well-developed slatey cleavage. Mineralized quartz and quartz-carbonate veins and veinlets occur within the slate in the shear zone and are broadly concordant with the slatey cleavage.

Alteration of the metasediments, slates and phyllites is weak. Mild sericitization is observed in the area but has no correlation with gold mineralization.

Gold mineralization is associated with a sulphide assemblage that consists of pyrrhotite with minor pyrite, arsenopyrite, and chalcopyrite. Coarse crystalline arsenopyrite and free gold are frequently observed in close association to one another within the central Minapampa zone.

The Ollachea deposit has been explored since the late 1990s. To date, exploration and resource drilling at Ollachea, by MKK (since 2008), consists of 208 diamond drill holes totalling 80,550 m in length (plus an additional 523 m in deviations from specific targets for a grand total of 81,073 meters of drilling). Drilling in the vicinity of Minapampa that was considered in the modelling process consists of 172 diamond drill holes totalling 67,298 m in length. Drilling intersections considered for the actual resource estimation process came from a total of 155 drill holes totalling 60,306 meters.

Samples were prepared and analyzed at Certimin (previously known as CIMM) Laboratories in Juliaca and Lima with blanks, standard reference materials, pulp duplicates, coarse crush reject duplicates, check assays and core twin samples included as part of a quality assurance and quality control (QA/QC) program to establish assaying accuracy and precision. QA/QC procedures consistent with industry best practices have been followed and verified by independent auditors. Drilling, sampling, sample chain of custody, preparation and assaying of samples in the mineral resource database are reasonable to support the estimation of Mineral Resources.

The three dimensional geological model constructed for the deposit serves to constrain gold mineralization in the estimate and is consistent with the genetic model and structural interpretation for the deposit. The geological model considers the continuity of geology and grade indicated by the diamond drilling and sampling in the current mineral resource database.

Mineral Resources have been estimated from domained 2 m composited samples, using ordinary kriging to estimate block grades into; 20 mE x 20 mN x 4 mRL parent blocks. The estimated parent blocks were divided into 2.0 m x 2.0 m x 0.4 m sub-blocks which were used to better define the volume of the individual mineralized zones. The composite length, sub-block size, estimation method and estimation parameters for composite selection in estimation and control of extreme grades are reasonable considering the deposit type, proposed mining method and geostatistical characteristics of the gold mineralization.

Mineral Resources for the Ollachea (Minapampa) Project at a 2.0 g/t Au lower cut-off grade consist of 10.6 Mt of Indicated Mineral Resources with an average grade of 4.0 g/t Au containing 1.4 million ounces gold and 3.3 Mt of Inferred Mineral Resources with an average grade of 3.3 g/t Au containing 0.3 million ounces gold. This Mineral Resource has been estimated by Doug Corley, MAIG, R.P.Geo, of Coffey Mining, Perth and Qualified Person under National Instrument 43-101 and has an effective date of 6 July, 2012 (Table 1-1). Mineral Resources are inclusive of the Mineral Reserves reported in Section 1.3.

Table 1-1: Mineral Resources for the Ollachea (Minapampa) Project

Mineral Resources above a 2.0 g/t Au Cut-off Grade	Tonnage (Mt)	Au Grade (g/t)	Contained Au (Moz)
Indicated	10.6	4.0	1.4
Inferred	3.3	3.3	0.3

Note:
Mineral Resources are estimated by Doug Corley, MAIG, R.P. Geo, QP, of Coffey Mining and have an effective date of 6 July, 2012. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources are inclusive of Mineral Reserves. Tonnages are metric tonnes and ounces of contained gold are troy ounces. Mineral Resources above a 2.0 g/t Au cut-off grade have reasonable prospects for economic extraction, based on mineralization continuity, shape and distribution and as demonstrated in this study.

A further Inferred Mineral Resource of 10.4 million tonnes grading 2.8 g/t containing 0.9 million ounces (above a 2.0 g/t Au cut-off) has been defined at Concurayoc (with an effective date of 31 August 2011). The Inferred mineral resource was announced in a press release by MIRL on the 7 September 2011 entitled “Substantial Maiden Mineral Resource Concurayoc Zone, Ollachea Project, Peru”.

Concurayoc is located approximately 400 m westward along strike from the Minapampa Zone. There have been no mining studies carried out on the Concurayoc Inferred mineral resource, and therefore it does not have any demonstrated economic viability.

Exploration targets on the Project include, but are not restricted to, the Concurayoc Zone the eastern extension of Minapampa, where drilling from surface is impractical for topographic reasons, and the down-dip extension of the Minapampa and Concurayoc Zones.

There are currently no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that would materially affect the mineral resource estimate for the Ollachea Project.

1.3 Mining and Mineral Reserves

The Ollachea deposit will be mined from underground using long hole open stoping (LHOS) with paste fill. All stopes will be accessed longitudinally and extracted on a level by level retreat basis. The mine will have two main mining areas, east and west. The west area contains multiple stacked lodes and contains the majority of the Mineral Reserves. The east area has a limited number of lodes and extends over a greater vertical distance than the west, resulting in a lower concentration of contained ounces. Multiple mining areas are required to be mined simultaneously in the east to support project economic objectives. All mining will use a bottom up mining direction. The minimum horizontal mining width is 2.6 m, including dilution.

Major mine development will be accessed via an exploration incline that is currently being developed. This will be followed by the development of a second portal (2nd means of egress), ramp developments, ventilation raises, level accesses and haulage drifts. All mine levels will be 15 m apart. In general, ore and waste will be transported from the underground mine by truck via the exploration incline, with waste being deposited at a waste dump located in close vicinity to the lower portal entrance and ore being deposited at the process plant ROM pad.

The primary ventilation system consists of the exploration incline, other incline and decline drives, four surface raises (two return air raises and two fresh air raises), and an internal return air system and connecting drive that services the eastern part of the mine. Primary fans will be located on the two surface return air raises. The expected peak airflow at full production will be 700 m^3/s.

A hydrogeological numerical model was developed by AMEC to understand the behavior of the groundwater system in the Ollachea project area. Initial flow rates during project ramp up will be up to 80 m^3/h, which then increase to approximately 120 m^3/h during full production. Due to the nature of the planned mine development mine dewatering will be predominately gravity assisted.

Coffey Mining completed the FS geotechnical assessment based on data from project geotechnical core logging and core photographs, structural studies, rock test results, and personal inspection. The *in situ* stress state was determined using the Deformation Rate Analysis (DRA) technique. The Modified Rock Mass Quality Index (Q’) was utilised to characterise the *in situ* rock mass. Non-linear finite element numerical modelling was undertaken using the ABAQUS/FEA program to assess rock mass damage due to mine design and proposed extraction sequence. Modelling results were used to provide criteria for FS mine design and extraction sequence.

Ground support requirements for mine development, including large excavations, have been recommended based on the Q Index. Maximum stable stope spans were determined using the stability graph method. Empirical analysis was used to assess crown pillar stability. Two possible locations for vertical surface ventilation shafts were subjected to a shaft stability exercise to provide an indication of raise boring risk and provide LOM support recommendations.

Recommendations arising from the FS geotechnical assessment have been used as the basis of the FS mine design and extraction sequence, and mine operating and capital costs.

Face mapping information from the current development of the exploration incline, which is now traversing the orebody host rock, has not been considered in the FS geotechnical assessment due to timing of data availability. It is recommended that this data be assessed on an ongoing basis and used to update the geotechnical analysis completed as part of the FS. This should form an integral part of the proposed project development.

Due to the "non-visual" nature of the mineralisation, diamond drilling will form a significant part of the mine grade control program. Holes will be drilled from planned hanging wall drives prior to ore development on a minimum grid pattern of 15 m by 15 m. Ore drives will then be driven primarily on survey control, backed by face and wall channel sampling. An onsite laboratory is planned and has been designed to provide a 24 hour turnaround of samples.

The LHOS mining method and extraction sequence adopted for the Project is reliant on the use of paste fill. Process plant total tailings will be used to produce the paste fill. Approximately 42% 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 paste plant will be located at Minapampa with tailings transported from the process plant to paste plant by truck via the underground mine.

Suitability of process plant tailings for production of backfill was determined from the analysis of particle size distribution, mineralogy and rheological properties. It was found that the tailings are rather fine and suitable for paste fill.

Over 200 paste UCS tests were conducted to investigate the impact of cement type, and cement and tailings content on the UCS over a testing period of seven to 182 days. Representative testing was conducted using locally sourced cements, ore tailings and tailings processing water produced during FS process plant testwork.

The mine is planned to be owner operated by MKK. Specialist contractors will be used for selected activities such as raise boring. The mine is planned to operate 24 hours per day, 365 days per year and all mine personnel will work a 14 days on, 7 days off roster. Shifts will be of 12 hours duration.

The planned mine will require a standard, medium scale, underground mobile production fleet of jumbos, LHDs, trucks and drills. All mobile and fixed plant equipment will be purchased, operated and maintained by MKK. The fleet of primary mobile equipment units was calculated directly from equipment productivity rates and scheduled mine physicals from the final mine design.

Industry standard software was used to design and schedule the planned mine. Consideration was given to geological and geotechnical design constraints determined as part of the FS.

A single LOM mine design cut-off grade (COG) of 2.0 g/t Au was used for the FS. This was selected based on a simple break even grade analysis, company strategic objectives and previous study financial outcomes.

Economic value has been identified in processing low grade development ore below the project COG at different stages of the project life. The material to be processed is sourced from stope access development that traverses through Indicated Mineral Resources but has been diluted below the Project COG of 2.0 g/t Au. As the mining cost for this material will have already been expensed, it is economic to treat through the plant. A mill COG of 1.0 g/t Au was used for the low grade development ore in the FS.

As per Table 1-2 Probable Mineral Reserves totalling 9.3 Mt grading 3.4 g/t Au and containing 1.0 million ounces of gold are declared based on the results of the FS and the application of appropriate mining factors, and taking into account relevant processing, metallurgical, economic, marketing, legal, environmental, socio-economic and government factors. Mineral Reserves are based on a gold price of US$ 1,300/oz, an exchange rate of 2.65 (Peruvian Nuevo Sole / US$), life of mine (LOM) average site operating costs of US$ 49.20/t ore and LOM average metallurgical recovery of 91%. Mineral Reserves have an effective date of November 29, 2012. The reported Mineral Reserve has been compiled under the supervision of John Hearne, FAusIMM (CP), and an employee of Coffey Mining, and who is recognized as a Qualified Person for the purposes of National Instrument 43-101. These Mineral Reserves are included within the Indicated Mineral Resources reported in Section 14.

Table 1-2: Mineral Reserve Estimate (November 29, 2012)

Classification	Tonnes (Mt)	Au Grade (g/t)	Contained Gold (koz)
Ore (+ 2 g/t Au)	8.7	3.5	983
Low Grade Development Ore (+1 g/t to 2 g/t Au)	0.6	1.5	28
Probable Mineral Reserves	**9.3**	**3.4**	**1,011**

Notes:
Probable Mineral Reserves are included within Indicated Minerals Resources and are declared inclusive of mining dilution with an effective date of 29 November, 2012.
Tonnages are metric tonnes and ounces of contained gold are troy ounces.
Probable Mineral Reserves are declared based on a base case gold price of US$1300 / oz, a project COG of 2.0g/t, LOM project operating costs of US$49.2 /t ore and a mill recovery of 91.04%.
Low Grade Development Ore is sourced from development drives that traverse through Indicated Mineral Resources but has been diluted below the project COG of 2.0 g/t Au. As the mining cost for this material will have already been expensed, it is economic to treat through the plant. A mill cut-off grade of 1.0 g/t Au has been applied to this material.
Mineral Reserves were estimated under the supervision of John Hearne, BEng(Mining), MBA, FAusIMM, CP(Mining), of Coffey Mining, and who is recognized as a Qualified Person for the purposes of National Instrument 43-101.

A summary of the main parameters used in estimating the Mineral Reserves are shown in Table 1-3.

Table 1-3: Main Parameters used for the Mineral Reserve Estimate (November 29, 2012)

Description	Units	Value
Gold Price	US$/oz	1,300
Mine Design Au Cut-off Grade	g/t	2.0
Mill Au Cut-off Grade[1]	g/t	1.0
Mining Method		LHOS
Minimum Mining Width (excluding dilution)	m	2.0
Annual Production Rate	Mt /a	1.1
Mining Operating Cost	US$/ t ore	23.4
Milling Operating Cost	US$/ t ore	21.5
G&A Operating Cost	US$/ t ore	4.3
Mining Dilution - Development.	%	21
Mining Dilution - Stopes.	%	19
Mining Recovery (within mine design shape)	%	100
Mill Recovery	%	91.04
Project Capital Cost	US$M	177.5
Sustaining Capital Cost	US$M	41.5
Closure Cost	US$M	4.2
Royalties	US$M	27.0
Special Mining Tax (SMT) or Especial de Mineria (IEM)	US$M	14.2
Workers Profit Share	%	8
Corporate Income Tax	%	30

The LOM FS mine physicals based on extracting the Probable Mineral Reserves are shown in Table 1-4.

[1] Applied to low grade development ore.

Table 1-4: Ollachea LOM FS Physicals

Physical		Units	LOM
Lateral Dev.	Capital dev.	m	8,019
	Operating dev.	m	56,987
Vertical Dev.	Capital dev.	m	1,586
	Operating dev.	m	220
Production	Total Mined	Mt	11.9
	Waste	Mt	2.6
	Ore	Mt	8.7
	Contained Gold	koz	983
	Gold Grade	g/t	3.5
	Low Grade Development Ore	Mt	0.6
	Contained Gold	koz	28
	Gold Grade	g/t	1.5
	Cable drill	kdm	662
	Production drill	kdm	697
	Haulage	Mtkm	30.1
	Backfill Void (Paste)	km³	3,012

Totals may not sum due to rounding

The Project is scheduled to start in January 2013 with first stope production in January 2015. Full mine production is reached in December 2016 and extends until December 2021. Mine production will stop at the end of July 2024, giving mine production tail a duration of 31 months. Full mill production commences in May 2016 based on processing direct supply mine ore, direct supply mine low grade development ore and stockpiled low grade development ore. Full mill production continues until November 2022 with stockpiled low grade development ore used as supplementary feed source.

The underground mine has been designed and scheduled to produce ore at the rate of 1.1 Mt/a. Due to the nature of the mineralisation, grade distribution and geotechnical environment, multiple stopes are to be turned over on a regular basis to meet the production target. This requires the designed mine development to be completed at a rate of approximately 800 meters per month to support stope turnover. Stope turnover (inclusive of drilling, blasting, filling, curing and associated preparation) is estimated to take between 30 and 75 days dependent on stope size. Production continuity will be primarily dependent on maintaining a balance between development and stoping. The completion of a high number of parallel activities is required to meet planned production targets; this will require good operational management and technical application to ensure required mining efficiencies are maintained.

1.4 Metallurgical Testwork and Process Design

The interpretation of results from metallurgical testwork carried out during 2012 has been used to guide process plant design. Testwork suggests that crushing and grinding of ore to P_{80} of 106 µm with gravity concentration (GRG recovery), high mass pull gravity (HMPG) concentration, separate carbon-in-leach (CIL) treatment of HMPG concentrates, followed by CIL treatment of the re-combined whole of the ore stream can be used to achieve gold recovery of over 91% from the Ollachea mineralization.

The flowsheet applied will comprise two stages of open circuit crushing followed by overflow ball milling, with the mill circuit in closed circuit with hydrocyclones. A gravity circuit employing a centrifugal concentrator will treat a split of the hydrocyclone underflow, the concentrate from which will be further upgraded using a Gemeni table, prior to smelting.

Hydrocyclone overflow will undergo continuous gravity concentration in the HMPG circuit. HMPG concentrates will be pre-aerated prior to being leached in a dedicated Carbon in Leach (CIL) circuit for approximately 24 hours. HMPG tails will be pre-aerated prior to being recombined with gravity tails and leached for approximately 36 hours in another CIL circuit.

Use of CIL and blanking reagents are techniques that will be employed to reduce the influences of preg-robbing minerals in the ore, such that almost all of the leachable gold will be recovered by the activated carbon in the CIL. To ensure high activated carbon quality, the circuit will include acid washing and Anglo American Research Laboratories (AARL) elution followed by thermal carbon regeneration. Final doré production will be achieved on-site by electrowinning of the AARL eluate and smelting.

CIL tailings will be treated to detoxify cyanide using the SO_2/air process, followed by iron (cyanide) precipitation by zinc sulphate. After iron precipitation, tailings will be thickened using a high-rate thickener and filtered using press filters. The filter cake will be transported to a paste plant at Minapampa to produce pastefill, when backfill is required in the underground mine. When backfill is not required, the filter cake will be stacked on a load-out platform for reclaim and haulage to a dry-stack tailings storage facility.

Plant design includes a water treatment plant (pre-aeration followed by neutralization and thickening) that will treat mine drainage, site run-off, and process plant excess water prior to discharge to the environment. Reagents including hydrated lime for pH control, sodium cyanide for leaching, kerosene for blanking, hydrochloric acid for carbon washing, copper sulphate and sodium metabisulfite for cyanide detoxification, and zinc sulphate for iron precipitation will be prepared and distributed by reagent preparation circuits considered in the FS plant design.

Metallurgy and mineral process design has been supervised and reviewed by Marius Phillips, MAusIMM (CP) of AMEC, Brisbane, who is the Qualified Person under NI 43-101 for this work.

1.5 Tailings Disposal Facility

The tailings storage facility (TSF) has been designed for a capacity of 5.85 Mt of filtered tailings with overall ultimate slopes of 2.5H:1V and an ultimate height of approximately 145 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.

A starter buttress will be constructed at the toe of the TSF for stability and erosion protection. The starter buttress will be constructed of compacted colluvial soils sourced from within the TSF footprint.

The TSF design considers two zones for tailings placement: (i) perimeter TSF tailings shell (Zone A), and (ii) interior TSF zone (Zone B).

The compacted “Zone A” tailings zone will form a structural “shell” that will be located at the exterior perimeter area of the TSF. Zone A tailings will be placed in 0.3 m-thick lifts that will be compacted to achieve at least 95% of the maximum dry density as determined by the standard Proctor test (ASTM D698).

1.6 Project Operating and Capital Costs

1.6.1 Operating Cost Estimates

Operating costs include fixed and variable costs for mine production, plant production, tailings management and general and administrative services for the operation. Operating costs were estimated based on labour and productivity data from current Peruvian mine operations including the Corihuarmi Mine operated by MKK’s parent company IRL, from AMEC and Coffey Mining cost estimation databases, and from quotations for major reagents, consumables and wear parts.

A life-of-mine staffing schedule was developed and indicates a peak operating staffing of 364 personnel. As much as 25% of the workforce will be locally based, with the remainder being based nationally.

A consolidated unit operating cost schedule is shown in Table 1-5. Mine operating costs average US$23.4/t ore processed (costs include backfill). Plant operating costs average US$21.5/t ore processed (including tailings disposal) and G&A costs average US$4.3/t ore processed. Total site operating costs are US$ 49.2/t ore processed or US$499/oz of gold.

Table 1-5 Overall Operating Cost Schedule

Operating Cost		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	Total LOM
Process Plant															
Supplies	US$M			11.8	15.5	18.4	18.5	18.4	18.6	18.7	18.5	12.2	3.6		153.6
Labour	US$M			1.0	1.2	1.3	1.3	1.3	1.3	1.3	1.3	1.0	0.6		11.3
Maintenance supplies and Misc.	US$M			1.0	1.2	1.3	1.3	1.3	1.3	1.3	1.3	1.0	0.7	-	11.8
Total Process Plant	US$M			**13.3**	**17.9**	**21.0**	**21.1**	**21.0**	**21.2**	**21.3**	**21.1**	**14.2**	**4.8**		**176.7**
TSF	US$M			**1.9**	**2.4**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.0**	**1.0**		**23.3**
G&A	US$M			**4.1**	**4.3**	**4.5**	**4.5**	**4.5**	**4.4**	**4.3**	**4.0**	**3.8**	**2.1**		**40.5**
Mining	US$M														
Fuel	US$M			1.6	1.9	2.4	2.4	2.4	2.3	2.2	1.9	1.6	0.5		19.2
Explosives	US$M			2.3	2.6	3.0	3.0	2.8	2.5	1.5	1.0	0.9	0.2		19.8
Maintenance Supplies	US$M			3.2	3.9	5.0	5.1	5.0	4.4	3.9	3.5	2.8	1.0		37.9
Labour	US$M			4.7	5.0	5.6	5.7	5.5	5.3	4.1	3.9	3.1	1.1		44.0
Power	US$M			2.1	2.3	2.5	2.5	2.5	2.4	1.8	1.4	1.2	0.5		19.2
Consumables	US$M			6.1	7.5	9.5	8.7	8.8	8.9	8.3	5.4	4.3	1.2		68.7
Other	US$M			1.1	1.1	1.2	1.3	1.2	1.2	0.9	0.7	0.5	0.2		9.4
Total Mining	US$M			**21.1**	**24.4**	**29.1**	**28.7**	**28.3**	**26.9**	**22.7**	**17.8**	**14.4**	**4.6**		**218.2**
TOTAL	US$M			**40.4**	**49**	**57.3**	**57**	**56.5**	**55.2**	**51**	**45.6**	**34.4**	**12.5**		**458.7**

1.6.2 Capital Cost Estimates

The capital cost estimate for the processing plant and associated infrastructure was prepared by AMEC and the mining cost estimate was prepared by Coffey Mining. IRL provided the estimate for the Owner's capital cost. The accuracy of this estimate is within -10/+15%.

The initial capital is estimated to be $177.5 M and sustaining capital cost is estimated to be $45.7 M over the life of mine.

The total capital cost estimate is shown in the Table 1-6.

Table 1-6 Capital Cost Estimate Summary

CAPEX	US$(M)
Mine	55.1
Process Plant	72.0
EPCM	18.6
Below The Line Costs (Incl Contingency & Owners costs)	31.8
Total Capital Cost Estimate	**177.5**
SUSTAINING CAPITAL	
Mining Sustaining	38.3
Waste Dump Closure	2.0
TSF Closure	2.2
Process Plant Sustaining	3.2
Total Sustaining Capital Estimate (Life of Mine)	**45.7**
Project Total	**223.3**

1.7 Financial Analysis

A financial evaluation of the Project was undertaken using the discounted cash flow analysis approach. Cash flows were projected for LOM, which includes construction, operation and closure phases. The cash inflows were based on projected revenues for the LOM. The projected cash outflows, such as capital costs, operating costs, royalties and taxes; were subtracted from the cash inflows to estimate the net cash flows. A financial model (the Model) was constructed on a monthly basis to estimate the net cash flows (NCF) over the LOM. The NCF were summarised on an annual basis. The cash inflows and outflows were assumed to be on a constant 3rd quarter 2012 US dollar basis.

The Project was evaluated on a 100% project stand-alone and 100% equity-financed basis. The financial results, including Net Present Value (NPV) and Internal Rate of Return (IRR) do not take past expenditures into account; these are considered to be sunk costs. The analysis was done on a forward-looking basis from commencement of development in January 2013, with the exception of the sunk costs to date, which were taken into account for tax calculations as an allowable deduction. Any other expenditure after 31 December 2012 not related to the Project construction has not been included

The inputs and assumptions that form the basis of the Model include metal prices, mining schedule, mining inventory, processing throughputs, metallurgical recoveries, realisation costs, operating costs, capital costs, royalties and taxation parameters.

The base case gold price used in the financial evaluation was US$1,300/oz. The PEN/US$ exchange rate used was 2.65. The results of the economic analysis represent forward-looking information as defined under Canadian Securities Law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

A summary of the results of the financial analysis is shown in Table 1-7.

Table 1-7 Summary of Ollachea Financial Results

Parameter	Unit	Base Gold Price US$1,300/oz	Upside Gold Price US$1,600/oz
Net Cash Flow before tax	US$ M	489	749
NPV @ 5% real (before tax)	US$ M	309	497
NPV @ 7% real (before tax)	US$ M	256	422
NPV @ 10% real (before tax)	US$ M	192	331
IRR (before tax)	%	29.2	40.2
Payback (before tax)	Years	3.2	2.5
Net Cash Flow (after tax)	US$ M	325	486
NPV @ 5% real (after tax)	US$ M	194	310
NPV @ 7% real (after tax)	US$ M	155	258
NPV @ 10% real (after tax)	US$ M	108	194
IRR (after tax)	%	22.1	30.2
Payback (after tax)	Years	3.7	3.0

Note:
NPVs as at commencement of construction.
NPVs are based on mid-period discounting.
Before tax is before Special Mining Tax, Workers' Participation Profit of 8% and Income Taxes of 30%.
Payback starts from the commencement of production.
The financial results are on 100% Project basis and exclude the agreement with the community for a 5% participation in MKK on commencement of production and Second Additional Payment payable by MKK and due to Rio Tinto in accordance with Mining Claim Transfer Agreement dated 23 February 2007.

Sensitivity analysis was performed on the Base Case NPV, using a 7% discount rate, and IRR (Figure 1-1 and Figure 1-2). Positive and negative variations up to 15% were applied independently to each parameter: gold price, capital cost, operating cost and gold grade). The results demonstrated that the project is most sensitive to variation in gold grade and gold price, and least sensitive to variation in capital cost.

Figure 1-1 NPV at 7% real (post-tax) Sensitivity Chart

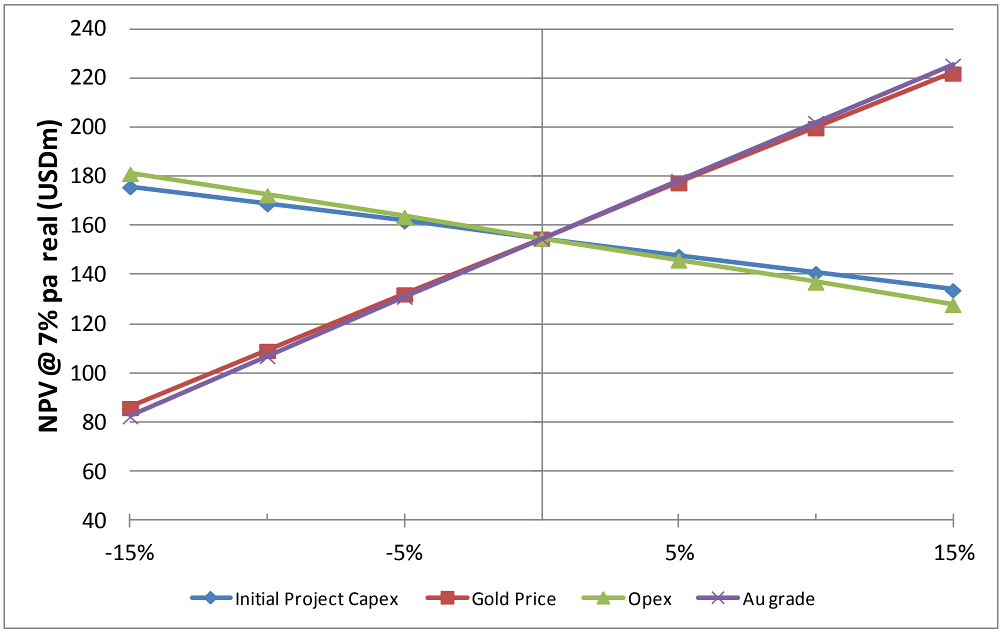
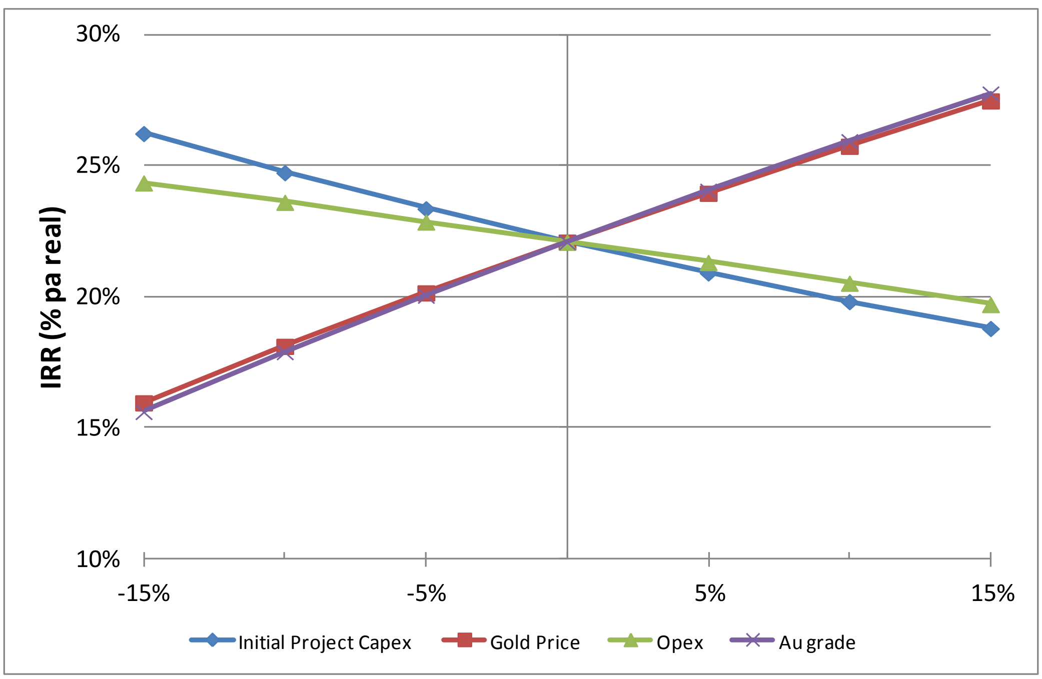


Figure 1-2 IRR real (post-tax) Sensitivity Chart



As with most gold projects, gold price is one of the most sensitive elements of the analysis. The breakeven point of the gold price for the NPV @ 7% real (after tax) is US$872/oz whereas the IRR real (after tax) reaches zero when the price of gold is US$679/oz. Table 1-8 shows the impact of different gold prices on Project returns.

Table 1-8 Sensitivity of Financial Returns versus gold price

Gold Price (US$/oz)	Pre-tax		Post-tax	
	NPV @ 7% (US$ M)	IRR (%)	NPV @ 7% (US$ M	IRR (%)
1,000	89	15.6%	48	12.2%
1,100	148	20.6%	84	15.8%
1,200	206	25.1%	120	19.1%
1,300	264	29.2%	155	22.1%
1,400	320	33.0%	189	24.9%
1,500	377	36.7%	224	27.6%
1,600	433	40.2%	258	30.2%
1,700	490	43.5%	292	32.6%
1,800	546	46.7%	325	35.0%
1,900	601	49.7%	359	37.3%
2,000	657	52.7%	392	39.5%

1.8 Conclusions

The results of the Definitive Feasibility Study indicate that the Ollachea Project, under the assumptions in the study, returns positive project economics.

1.9 Project Risks and Opportunities

1.9.1 Geology and Mineral Resources

Opportunities

Potential exists to increase the current Mineral Resource, both along strike and down dip from the current known Mineral Resource.

Potential exists for grade control drilling to delineate additional mineralisation; adjacent to known mineralised zones.

Potential exists for Inferred Mineral Resources located in close vicinity to the declared Mineral Reserves to be upgraded to the Indicated resource category. This is particularly true of the Concurayoc Zone, located 400m west of the Minapampa Zone, where 0.9 Moz of Inferred Resource has been outlined.

Risks

As the mineralisation is "non-visual", grade control delineation is critical to spatially locating mineralised zones. Grade control drilling on a minimum of a 15 m x 15 m grid is planned to allow ore development drives to be located optimally.

Grade control drilling and geological mapping will be required to identify any possible north – south fault offsets to known mineralised zones; not identified by the dominantly north – south orientated mineral resource drilling. Grade control drilling should be angled to intersect any potential north – south striking structures.

1.9.2 Mining and Mineral Reserves

Opportunities

Parts of the FS mine development design have been identified that could be optimised resulting in a reduction of metres, waste tonnes and ultimately cost and development time. These include:

- Diamond drilling platforms designed to provide coverage in the western part of the mine could be reduced such that a full platform on every second level would provide coverage for two levels.
- The twin surface return air raises require a lower and upper connection to allow the air flow to be balanced. Early development of the upper connection would negate the requirement for each and every level to have two connections to these raises.

Section 15 identifies several parts of the FS mine design where there is potential to recover economic material through further investigation with the current available data or when new geological and geotechnical data becomes available. These parts include:

- The dimensions of the crown pillar were determined based on limited geotechnical data. With further investigation and data collection, including mapping of the near surface artisanal mines, crown pillar dimensions could possibly be reduced. Alternatively, full crown pillar extraction using high strength paste backfill should be investigated.
- Economic stopes located in close vicinity to the widest part of a shear zone that obliquely cuts through the orebody.
- Economic stopes located in close vicinity to primary ventilation raisebore holes.
- Use of an alternative extraction sequence to negate the use of sill pillar located in the western part of the mine.

The FS geotechnical assessment has provided recommendations on stable stope dimensions based on a limited data set (but appropriate for a FS). As mining commences, capturing and analysing actual stope performance compared to this initial assessment will provide invaluable data that can be used to optimise stable stope dimensions. Development of a linear numerical modelling capability and calibrating models against observed conditions will also assist the stope performance assessment. Any increase in stope dimensions, especially along strike, will positively impact on mine production efficiencies.

Similarly, geotechnical data collected during the initial stages of mine development will enable ground support designs to be optimised to better suit the prevailing conditions. Any reductions in fibre reinforced shotcrete (recommended in the FS for capital development) would provide positive cost benefits to the Project.

Cement is one of the largest cost contributors to the mine operating costs through its use in paste backfill. FS testwork has indicated that there is scope for potential reductions in cement consumption based on increasing tailings solid content. The addition of sand and / or chemical additives should also be investigated as this may also provide cost savings by reducing cement requirements.

Risks

If stope performance were continually less than expected on a major scale, e.g. excessive hangingwall overbreak, stope turnaround times would be negatively impacted to the extent that planned production targets may not be readily achieved. This would also increase costs due to increased tonnage to be hauled and void to be filled. Excessive overbreak would also dilute mine head grade with a resultant decline in ounces produced. The development of early stopes, or trial stopes, is a means to mitigate this risk. Close monitoring and observation would allow stope dimensions and cable bolt support patterns to be optimised.

A diamond drilling grade control programme using a minimum grid spacing of 15 m by 15 m is required to allow ore drives to be optimally located for effective and efficient stope extraction. A highly variable hangingwall over short distances would present challenges to the placement of these ore drives. This may require an even smaller spacing to be drilled which would increase production cost and delay ore drive development. Increasing diamond drilling resources would mitigate time delays; however, this will further increase production costs.

The ground conditions around the widest part of the oblique shear zone have been identified as a risk resulting in economic stopes being removed from the Mineral Reserves. An increase in the size of this area would have a negative impact on the Mineral Reserves. The closely spaced diamond drilling grade control programme will provide an early indication of the scale and extent of this risk.

Four surface raisebore holes are planned as part of the primary ventilation system. A shaft stability assessment completed as part of the FS on two potential areas highlighted the potential for excavation stability risk during boring and for LOM. To mitigate this risk, specific diamond drill holes must be drilled early in each of the locations selected during the FS mine design exercise, with each hole geotechnically logged to an engineering standard. Shaft stability assessments should then be completed on each location, and if satisfactory, be used as part of the raise boring tender. An unsatisfactory outcome on any of the shaft stability assessments would require consideration to be given to alternative development techniques or a new location(s) to be identified. Any change in the location would require the FS mine design and schedule to be updated to reflect the location change(s) prior to implementation.

The FS mine development and production schedule is based on the continuous extension of the exploration incline. This requires the current development contract to be extended and necessary permits to be granted. Failure to achieve any of these in a timely manner will have a direct impact on the planned project ramp-up.

The mine has been designed and scheduled to produce ore at the rate of 1.1 Mt/a. Due to the nature of the mineralisation, grade distribution and geotechnical environment, multiple stopes are to be turned over on a regular basis to meet the production target. This requires a sustained high number of parallel development and production activities to be completed. Any issues encountered when completing these activities that leads to a reduction in operating efficiency will limit the mines ability to meet the desired production target with a likely reduction in ounces produced. Good operational management and technical application is required to mitigate this risk.

The project in considered technically and operationally complex due to the nature of the orebody, geotechnical environment and required high turnover of stopes to meet planned production targets. MKK and its parent IRL at the completion of the FS have limited in-house underground experience therefore will require a major recruitment drive in the short term to hire the necessary technical and operational personnel to implement and operate the underground mine at the designed capacity.

1.9.3 Metallurgy and Mineral Process Design

Opportunities

- Leaching at elevated temperature has the potential to partially mitigate the impact of carbonaceous material on gold extraction. This temperature affect has not been fully considered within the current process design. Marginal benefits may be expected as mill discharge slurry temperature will approach 40°C.

- Increasing the mass pull to the HMPG CIL circuit could potentially benefit final residue grade given the consistent leaching performance of this circuit.

- During the test work program, anecdotal evidence of increased viscosity, within the HMPG CIL, was observed. Consequently, the current design allows for the leach circuit to operate at 30% solids. The use of caustic soda, as a modifier, would potentially reduce the viscosity in this circuit, offering the potential for this circuit to operate at 38-40% solids which would allow optimisation of the HMPG CIL leach tank size.

- Ore Zones 5, 6 and 7 currently yield higher residue grades, and, consequently, lower extractions, than Ore Zones 1, 2 and 3. This conclusion is based upon limited data points, from the FS test work program. Further test work, focusing specifically on these three zones, is recommended with the aim of reducing reside grade through further optimisation of reagents and leach conditions.

- The LOM recovery provided is dependent upon the mine plan utilised for its derivation, and the proportion of respective ore zones, comprising the mine plan. A change in the proportion of ore from the respective ore zones would affect LOM recovery.

- Laboratory testwork cyanide consumption is normally much higher than what is seen in the full scale plant, due to the effect of fresh carbon addition. Cyanide consumption testwork to establish the impact of carbon on cyanide consumption and to quantify a realistic operating cyanide consumption is recommended

Risk

- Test work has indicated that leach extraction is variable, with different ore zones yielding variable final residue grades, with Ore Zones 1, 2, 3 and 4 yielding significantly better extraction results than Ore Zones 5 and 6. Consequently, this could lead to variable recovery in the process plant

- The LOM recovery provided is dependent upon the mine plan utilised for its derivation, and the proportion of respective ore zones, comprising the mine plan. A change in the proportion of ore from the respective ore zones would affect LOM recovery.

- Testwork has indicated that, to ensure a consistently low final residue grade, the recovery of gold to the HMPG circuit is to be maximised, with lower residue grades experienced when the HMPG CIL contribution to overall recovery increases. Consequently, this could lead to variable recovery and cyanide consumption

- Test work indicates that the comminution characteristics of the ores are variable. Similarly, the RWI:BWI differential indicates the potential for critical size material being rejected from the ball mill, as scats. Scats build-up would impact milling throughput and would require further processing through a pebble crusher. The FS design provides for as future pebble crushing installation, but is currently not included in the plant design

- The water quality of the feed to the water treatment facility requires verification. Currently, the water treatment facility has sufficient volumetric capacity and retains reagent dosing systems typical of a high density sludge neutralisation plant. Chemical requirements, with respect to reagent addition, and consequently operating cost may be negatively impacted, should water quality be poorer than assumed. However, as poor quality water from mine infiltration to date has not been observed, this is equally an opportunity for potential savings in operating costs.

1.9.4 Recommendations

Geology and Mineral Resources

Due to the non-visual nature of the mineralisation at Minapampa, it is recommended for grade control purposes diamond drilling is conducted on a minimum of a 15 m x 15 m grid to confirm the spatial location of the mineralised zones and determine stope boundaries. Estimated LOM cost to complete this task is US$4,600,000.

Samples for bulk density determination should be taken from mineralized zones, as an ongoing exercise. A target of 100 density determinations per zone (or more) should be taken to adequately characterize variability in the mineralized zone. Once a regular underground drilling program has begun, it would be recommended to take one density determination per sample collected, to develop a large enough database, so the bulk density can be estimated in a similar way as gold is. Estimated LOM cost to complete this task is US$200,000.

Mining and Mineral Reserves

Due to timing, and location, of available data no cognisance was taken of exploration incline face mapping during the course of the FS. This development is now located in orebody host rock and face mapping data is being captured and reported using a RMR classification system. FS geotechnical design was completed using the Q system and specific geotechnical design methodologies based exclusively on this system (Stope Stability Chart (Mathews-Potvin), Ground Support Selection Chart (Barton-Grimstad) and Raisebore Stability Analysis (McCracken-Stacey)). Coffey Mining therefore recommends that the Q system be adopted as the project standard for rock mass classification as this system is considered the most appropriate for designing and optimising this type of underground mine. Technical personnel currently onsite will be able to complete this at no additional cost.

To mitigate risk associated with the development of surface shafts using the raise boring technique diamond drill holes centred on the location of each of the four planned holes should be prioritised and planned for early completion. Estimated cost to complete this task is US$100,000.

Several opportunities have been identified with regard to FS mine design and schedule optimisation. These should be reviewed prior to project implementation to determine the scale and possible benefit of the opportunity, and whether the outcome would warrant a change to the base FS mine design and schedule. Estimated cost to complete this task is US$50,000.

The FS development and production schedule is reliant on the continuous extension of the exploration incline. Both permit approval and an extension to the current development contract must be expedited to adhere to the project implementation schedule. Estimated cost to complete this task is US$10,000.

For the FS no formal underground equipment tenders were issued. Budget quotes were sourced from OEM suppliers and agents that sell similar types of planned equipment. Formal underground equipment tenders require to be issued as part of the implementation phase of the project. Given the planned accelerated project implementation schedule underground equipment tenders are considered a priority activity. Estimated cost to complete this task is US$25,000.

Metallurgy and Mineral Process Design

The Ollachea FS metallurgical test work program has yielded sufficient information to develop a definitive metallurgical flow sheet, with quantifiable metallurgical outcomes, with respect to product and tailings quality. Recommendations for consideration include:

- Conducting additional cyanide detoxification test work to confirm the results achieved through inclusion of the ZnSO4 step, which is critical to ensuring discharge criteria are met. Estimated cost to complete this task is US$20,000.

- Conducting additional test work on Ore Zone 5 to establish whether altered process parameters, with respect to blanking reagent addition, carbon or cyanide concentration, will result in improved residue grades. Estimated cost to complete this task is US$40,000.

- Obtain additional information on the quality of the water to be treated by the water treatment plant, and perform water treatment testwork to ensure the chemical considerations are adequate. Estimated cost to complete this task is US$20,000

Project Infrastructure

- Slope monitoring at the TSF Cuncurchaca site should be carried out throughout the life-time of the proposed facility. Local failures, mostly due to the lack of erosion protection, on the exposed cuts along the Interoceanic highway, are a regular occurrence. Although no visible signs of general land mass movement were identified during the geohazard evaluation of the area, two inclinometers were installed immediately downhill of the proposed TSF to monitor potential ground movement. It is recommended that monitoring is conducted at least twice a year or more frequently if movement is measured. Estimated LOM cost to complete this task is US$200,000.

- Additional geotechnical investigation consisting of test pits and borings is recommended along the proposed TSF access road. Geotechnical borings are recommended for the proposed bridge crossing the Cuncurchaca River. Estimated cost to complete this task is US$150,000.

- Additional studies are recommended to identify and characterize material borrow sources in the vicinity of the project. In particular, sources of low-permeability soil for use in caps for closure of waste rock facilities were not identified in the feasibility study. Estimated cost to complete this task is US$40,000.

- On-going monitoring of piezometers installed at the TSF, waste rock dump and process plant site is recommended to characterize seasonal fluctuations of the phreatic surface. Monthly monitoring is initially recommended. Based on fluctuations of readings, the monitoring frequency may be revised. Estimated LOM cost to complete this task is US$40,000.

- Geotechnical characterization of waste rock, including grain-size distribution, durability and anticipated weathering, should be evaluated to confirm the criteria assumed for the waste dump design. Estimated cost to complete this task is US$15,000.

- A geotechnical exploration consisting of additional borings and test pits is recommended for the Minapampa access road. As it is currently proposed, the road design could require significant cuts in an area where little geotechnical information is available to determine batter angles or the need for retention systems. Estimated cost to complete this task is US$250,000.

- Results from a hydrological study of the Ollachea River basin should be used to determine the safe elevation/distance of any facilities to be constructed in the vicinity of the river such as the campsite or any other potential structures. Estimated cost to complete this task is US$40,000.

2 INTRODUCTION

The Ollachea Gold Project (the Project) is located in the Puno Region of southern Peru. MKK, a wholly-owned subsidiary of IRL and which is a wholly own subsidiary of MIRL, currently owns the Property and retained AMEC Australia Pty Ltd (AMEC) and Coffey Mining to conduct a Feasibility Study (FS) on the viability of mining the deposit from underground and processing ore in a 1.1 Mt/y facility on the property to produce gold doré. The project location is included as Figure 2-1.

Figure 2-1: Ollachea Project Location

2.1 Terms of Reference

This Independent Technical Report was prepared to provide technical information to support the 29 November, 2012 press release issued by MIRL titled: Minera IRL Ltd Announces Definitive Feasibility Study, Ollachea Project, Peru.

2.2 Qualified Persons

Qualified Persons responsible for the content of this technical report are:

- Doug Corley, BSc (Hons) (Geology), MAIG, R.P.Geo. ; Coffey Mining Principal Resource Geologist; Mineral Resources QP; responsible for Items 10-12 (excluding items10.6 and 11.2), and 14

- Donald McIver, MSc (Exploration and Economical Geology), BSc (Hons) (Geology), BSc (Geology and Chemistry), FAusIMM; FSEG; IRL VP Exploration; Geology and Exploration QP; responsible for Items 6 – 9

- Tim Miller, BSc (Chemistry), Grad DipAppFin&Inv, MAppFin, MAusIMM, FFinsia, MIRL Chief Financial Officer and Company Secretary, responsible for Item 19

- John Hearne, BEng (Mining), FAusIMM; CP (Mining), Coffey Mining Regional Manager Western Australia; Mining and Mineral Reserves QP; responsible for Items 15, 16 (excluding 16.5 and 16.9)

- Vadim Louchnikov, FAusIMM, MEngSc (Geotechnical Engineering), BEng (Hons) (Mining Engineering); Coffey Mining Associate Mining Geotechnical Engineer; Geotechnical and Backfill QP; responsible for Items 16.5, 16.8

- Marius Phillips, P Eng, AMEC Brisbane, Process Consultant, MAusIMM(CP), responsible for sections 11.2, 13, 17, 21.2.3 and 26.3

- Grahame Binks, BEng (Hons Met), MEngSc, AusIMM(CP), AMEC Australia Pty Ltd, Strategic Business Manager, responsible for sections 1-5, 20-26 (excluding 21.2.3, 26.3)

- Brett Byler, MSc (Civil Engineering), BSc (Geological Engineering), P.E; AMEC (Peru) S.A., E&I Civil Engineer; Infrastructure QP; responsible for Item 10.6 and 18

- Jim McCord, PhD P Eng (Geological Science with dissertation in Hydrology), MSc (Hydrology), BSc (Civil Engineering, minor in Hydrology), AMEC (Peru) S.A , Principal Hydrogeologist, responsible for items 16.4 and 18.4.

2.3 Site Visits and Scope of Personal Inspection

Site visits was conducted by Marius Phillips (AMEC), Brett Byler (AMEC), Vadim Louchnikov (Coffey Mining), Neil Hastings (Coffey Mining) and Niresh Deonarain (AMEC) during January 2012.

Since the inception of exploration activities at Ollachea, Mr Donald McIver has had regular site visit with a frequency of at least one every four months. On Donald's last site visit (mid-November), Donald completed a 4-hour review of all geological developments and improved understanding of the mineralizing process coming out of the mentioned 50 Km of core re-logging. He additionally chaired a 3-hour discussion on the structural controls on mineralization. Beyond that Donald conducted a 2-hour visit to the underground access tunnel where he completed a geological evaluation of mapping procedure, data presentation, representative geochemical sampling and importantly, actual rock and water filtration conditions within the exploration access tunnel.

Niresh Deonarain has visited site in his capacity as the study manager of the feasibility study.

2.4 Effective Dates

The effective date of this report is taken to be the date of the finalization of the financial model for the Project on 29 November, 2012. The dates for critical information used in this report are:

- The database was closed out for estimating purposes on 23 April, 2012
- The updated Mineral Resource estimate and Mineral Resource block model were completed on 6 July, 2012
- The Mineral Reserve estimate for the project was completed on 29 November, 2012, The final FS mine plan was issued 19 October, 2012
- FS Mineral process engineering and capital cost estimation were completed 21 November, 2012
- The FS financial model was finalized 29 November, 2012.

At the effective date of the report, MKK were excavating an exploration decline. The depth of the 1.2 Km programmed incline (+1.5 percent slope), was 935 m at 07h00 on 11 December 2012.

2.5 Information Sources and References

This Report is based on information provided in the following key documents and files:

- FS Study Final Report (AMEC, 21.12.2012)
- PFS Study Final Report (AMEC, 2011d)
- Mineral Resource Block Model File - ol0712m.dm
- Mine Schedule File - EPS Report Ollachea_FS_2_Alt2_Rev4_201112_1355_Final.xlsx
- Capital Cost Estimate File - Cx_OllacheaGold_21112012_Final.pdf
- Exploration Access Drive Report (Geoservice, 2010)
- Geotechnical Site Investigations at the Proposed Ollachea Plant Site (Garcia, 2011)
- Interim Tailings Characterization Report (AMEC, 2011b)
- Financial Model File – 11_07_17_Ollachea_PFS_FinalDraft(17Jul11).xlsx
- 120902_E&YPeru_MineralRL-DescriptionofPeruvianMiningFiscalSystemReport_September2012.pdf

AMEC has also sourced information from appropriate reference documents as cited in the text and as summarized in Section 27 of this Report. Additional information was requested from, and provided by, MKK. AMEC has also relied upon other experts as outlined in Section 3.

3 RELIANCE ON OTHER EXPERTS

The QPs state that they are qualified persons for those areas as identified in the appropriate QP "Certificate of Qualified Person" attached to this Report. The authors have relied upon and disclaim responsibility for information derived from the following reports pertaining to mineral concession tenure, surface rights agreements, permitting, environmental and social impacts.

3.1 Exploration and Mining Concession Tenure

The Coffey Mining and AMEC QPs have not reviewed the mineral tenure nor independently verified the legal status, ownership of the Project area, underlying property agreements or permits. The QPs have fully relied upon and disclaim responsibility for information derived from legal experts for this information through the following documents:

Arevalo, M., 2011. Actual State of Ollachea Project. Unpublished internal IRL memorandum regarding the property status of the Ollachea Project prepared by Marco Arevalo for Diego Benavides dated 6 June, 2011. 3 p.

Tong, F., 2012. Certain mineral rights and permits held by Minera IRL S.A. and Compania Minera Kuri Kullu S.A. Memorandum prepared by independent legal studio Rodrigo, Elias & Medrano Abogados for RBC Dominion Securities Inc., Jennings Capital Inc., Haywood Securities Inc., Collins Stewart Europe Limited, Finncap Limited and Tim Miller, Chied Financial Officer & Company Secretary and dated 5 March, 2012. 32 p.

Qualified Persons have relied on, and disclaim responsibility for these opinions in Section 4.1, Section 4.2, Section 4.3 and 4.5.

3.2 Surface Rights

Information was provided by 2011 IRL Annual Information Form dated 29 March 2012 (IRL, 2012), a legal opinion memorandum from Estudio Rodrigo, Elias and Medrino (Tong, 2012), and direct communication from MKK (2012)

Qualified Persons have relied on, and disclaim responsibility for these opinions in Section 4.4.

3.3 Permitting

MKK currently holds exploration permits allowing them to conduct exploration drilling and the development of an exploration tunnel on the Property. Qualified Persons have relied on, and disclaim responsibility for, these opinions in Section 4.6.

3.4 Social and Environmental Impacts

A summary the environmental baseline of the Ollachea property was prepared by AMEC E&I Environmental Specialist Jorge Mera and Guillermo Pedroni with reference to documentation prepared in support of the modification of a semi-detailed environmental base line study (SDEIA) prepared for the Ollachea Property by Especialistas Ambientales S.A.C. in 2011 (Especialistas Ambientales, 2011) and approved in July 2011. Qualified Persons have relied on this opinion in Section 4.7.

3.5 Taxation Information

AMEC has fully relied upon and disclaims responsibility for taxation information provided by MIRL. All taxation information used throughout this document was provided by MIRL in consultation with their advisor, Ernst and Young Peru in the document “120905_E&YPeru_MineraIRLDescriptionOfPeruvianMiningFiscalSystemReport_September 2012.pdf”.

4 PROPERTY DESCRIPTION AND LOCATION

The Ollachea Gold Project is located near the town of Ollachea in the Ollachea District of the Carabaya Province in the Puno Region of southern Peru (Figure 4-1). The plant site will be located approximately 1,000 m north of the northern limit of the town and immediately west of the Interoceanic Highway. The Project is approximately 250 km southwest of the city of Puerto Maldonado and 250 km north of the City of Juliaca.

Figure 4-1: Ollachea Project - Location and Access

4.1 Property and Title in (Jurisdiction)

Information in this sub-section has been compiled from the Mining Guide to Peru (Ministry of Energy and Mines – General Mining Bureau, 2006).

The General Mining Law of Peru defines and regulates different categories of mining activities, prospecting, exploration, exploitation, and processing (D.S. No. 014-92-EM, 1992). Mining concessions are established using Universal Transverse Mercator coordinates using the WGS-84 system, and their areas usually exceed 100 hectares up to 1,000 hectares each. Mining titles are irrevocable and perpetual provided that the title is current with respect to payment of fees, any penalties that may be imposed, and that a mining operation has commenced within the title area. The amount payable as good standing fee is US$3 per hectare per year and must be paid prior to 30 June each year.

4.1.1 Exploration Concessions

The Ollachea Project consists of 12 concessions covering an area of 8,698.98 ha (Table 4-1). A map of the Ollachea Property is shown in Figure 4-2. The concessions are map-staked and defined and registered spatially by the location of their vertices.

Tong (2012b) concludes that the Ollachea Property is in good standing, valid and in full force and effect, therefore giving MKK the right to explore and exploit the minerals existing in the titled area. As at the date of Tong (2012), the validity fees of the Ollachea concessions have been paid with respect to all the years elapsed as from their filing, with the exception of the 2012 validity fees

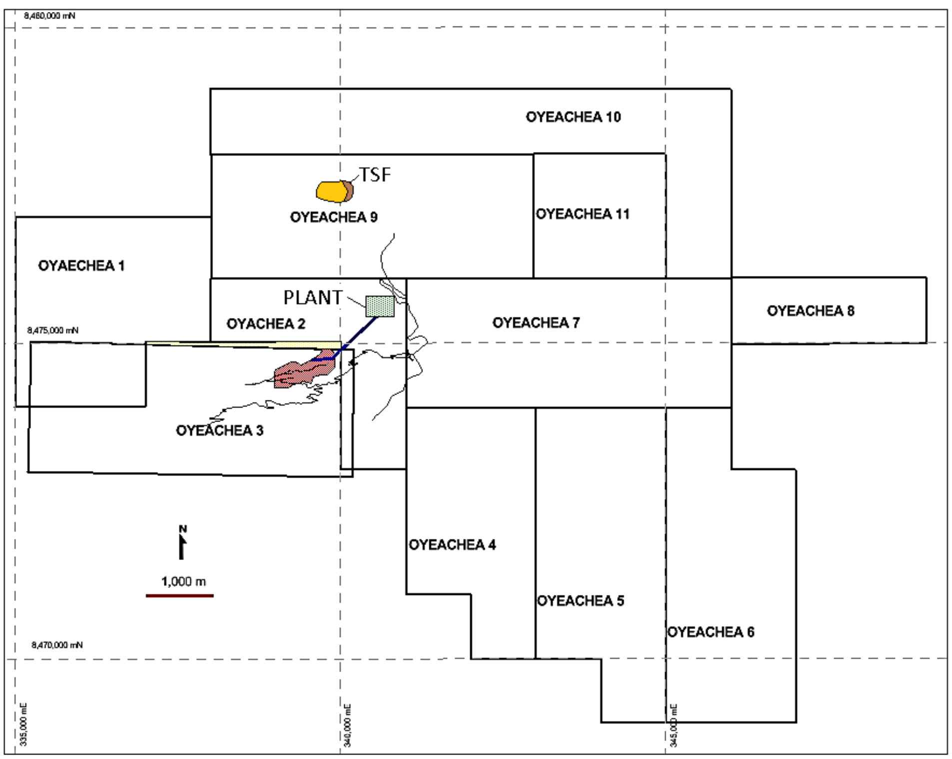
The mineralization included in the Mineral Resource and Mineral Reserves discussed in this Report occur within the Oyaechea 3 concession. The proposed plant site location will be located on the Oyaechea 2 concession. The portal location for the exploration access adit (currently being developed), which will serve as the main mine portal is located on the Oyaechea 2 concession. The Tailings Storage Facility will be located approximately 2.5 km north of the mine portal and within the Oyaechea 9 concession.

A gap measuring approximately 3,000 m long by 130 m wide exists between the Oyaechea 2 and Oyaechea 3 concessions (Figure 4-1). This concession is not held by MKK but by third-parties. The proposed exploration drive and other mine infrastructure discussed in this report have been located to avoid this gap.

Table 4-1: Ollachea Concessions

Concession Name	Concession Number	Concession Holder	Area (ha)	Registration Date
OYAECHEA 1	10215003	Compañía Minera Kuri Kullu SA	800	22/05/2007
OYAECHEA 2	10215103	Compañía Minera Kuri Kullu SA	500	22/05/2007
OYAECHEA 3	10218103	Compañía Minera Kuri Kullu SA	998.98	22/05/2007
OYAECHEA 4	10215203	Compañía Minera Kuri Kullu SA	700	22/05/2007
OYAECHEA 5	10215303	Compañía Minera Kuri Kullu SA	900	22/05/2007
OYAECHEA 6	10215403	Compañía Minera Kuri Kullu SA	900	22/05/2007
OYAECHEA 7	10389907	Compañía Minera Kuri Kullu SA	1000	14/05/2009
OYAECHEA 8	10389807	Compañía Minera Kuri Kullu SA	300	07/05/2009
OYAECHEA 9	10139909	Compañía Minera Kuri Kullu SA	1000	18/02/2010
OYAECHEA 10	10140009	Compañía Minera Kuri Kullu SA	1000	11/02/2010
OYAECHEA 11	10140109	Compañía Minera Kuri Kullu SA	400	11/02/2010
OYAECHEA 12	10167809	Compañía Minera Kuri Kullu SA	200	08/04/2010

Figure 4-2: Ollachea Exploration Concession Map



Note: The red polygon is the surface projection of Indicated Mineral Resources in the Minapampa Zone. The green polygon is the footprint of the mineral processing plant proposed in this feasibility study. The yellow polygon between the Oyaechea 2 and Oyaechea 3 concessions is a wedge-shaped gap in the MKK tenure holdings, and is owned by third-parties. The proposed exploration access drive is marked as a blue line and roads are marked as thin black lines. The proposed TSF is marked as an orange area in the Oyaechea 9 concession.

4.2 Mineral Tenure

The Oyaechea 1 to Oyaechea 6 concessions were originally registered by Rio Tinto Mining and Exploration Limited Sucursal del Peru (Rio Tinto) during its exploration activities at Ollachea beginning in 2006. On 1 September 2006, MIRL and IRL signed an agreement with Rio Tinto to acquire the original Ollachea concessions. On 23 February, 2007 the agreement was ratified and the Rio Tinto concessions were transferred to MKK (Tong, 2012). These transfers were officially recorded on 22 May, 2007.

From 2007 to 2009 MKK filed applications for the Oyaechea 7 to Oyaechea 12 concessions. These concessions were officially transferred during a period extending from 14 May, 2009 to 8 April, 2010. These concessions together with those previously held by Rio Tinto constitute the Ollachea Project.

4.3 Surface Rights

The following is summarized from the 2011 IRL Annual Information Form (IRL, 2012), a legal opinion memorandum from Estudio Rodrigo, Elias and Medrino (Tong, 2012), and direct communication from MKK (2012):

MKK negotiated a surface rights agreement with the Community of Ollachea covering an area of 5,998.9848 ha of the Oyaechea 3 concession, which was signed on 25 November 2007. This agreement was originally drafted for a maximum of five years. However, on 30 May, 2012 it was extended for a period of 30 years. MKK will make payments for surface rights access at a rate of 100,000 Nuevos Soles (approximately US$37,736) each year for 2013 and 2014. In addition, MKK agreed to pay 150,000 Nuevos Soles (US$56,604) for subsequent years and until the contract remains valid. MKK also commits to making contributions to sustainability projects and to social responsibility programs for the community totaling 3,360,000 Nuevos Soles (US$1,267,924) for the 2013 and 2014 years. This agreement is set to be revised for the contributions to social and community programs by 2015 when the mine is fully operational. The agreement also includes a contribution for technical support to artisan miners of US$300,000 over the life of the agreement. As a part of the agreement, upon the commencement of commercial production, MKK will transfer a participation of 5% of the share capital of MKK to the Community of Ollachea, giving them a participating interest in the project.

4.3.1 Agreements and Royalties

The following is summarized from the MIRL Annual Information Form for 2011 (IRL, 2012) and is supported by Tong (2012).

In September 2006 IRL was granted an option to acquire the property rights and a 100% interest in the Oyaechea 1 to Oyaechea 6 concessions from Rio Tinto for an initial payment of US$250,000 plus progressive payments totaling 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 IRL successfully negotiating a surface rights agreement with the local community within 120 days.

A surface rights agreement was reached in February 2007 (Section 4.4) and the Oyaechea concessions were transferred to MKK in accordance with Mining Transfer Agreement dated 23 February 2007.

Rio Tinto's clawback right lapsed in 2009 and on 15 December 2009, Rio Tinto notified IRL 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$3.81 million.

For the second additional payment to Rio Tinto, IRL and MKK has committed to making an additional cash payment of 30% of the net present value of the Ollachea Project (at a 7% discount rate) based on the results of the feasibility study, less 30% of the sunk costs determined after the exercise of this option. The second additional payment may be paid in three installments. The first installment is 34% of the second additional payment and is due 90 days after reception of notice from independent appraisers on the valuation of the FS. The second installment is 33% of the second additional payment and is due 12 months after reception of notice from independent appraisers. The third installment is 33% of the second additional payment and is due 24 months after reception of notice from independent appraisers. The second additional payment must be paid with a minimum of 20% cash with the balance in ordinary shares of MIRL. The second and third installment shall accrue an annual interest rate of 7%.

The Peruvian government currently levies a royalty based on gross profit per trimester from mining operations that ranges between 1% (for profits between 0% and 10%) and 12% (for profits greater than 80%). In addition, there is a royalty that is exclusive to mining activities that is based on operational margin (ratio between profit over sales per trimester), which ranges from 2% (for margins between 0% and 10%) to 8.4% (for margins above 85%).

4.4 Permits

MKK currently holds exploration permits allowing them to conduct exploration drilling and the development of an exploration tunnel on the Property. Additional permits will be required to support Project development. Permitting is discussed in more detail in Section 20.

4.5 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.

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.

Archaeological surveys have also been carried out as part of baseline study semi-detailed environmental impact assessment studies prepared to support applications for exploration permitting. These archaeological surveys, and others carried out as part of construction of the Interoceanic Highway have identified two isolated, minor sites in the vicinity of the Challuno area which has been proposed as the plant site area. Consideration of these sites has been taken in the Project layout.

The archaeological studies supervised by the Ministry of Culture allowed the rescue of tools and pieces of ceramics of ancient cultures which will be exhibited in a Cultural interpretation Center to be installed by the company in the Community of Ollachea with the authorization of the Culture authorities. The two intangible archaeological areas previously identified by the constructor of the Interoceanic Highway in the area of Challuno have been isolated, and they are not part of the design of the plant project.

4.6 Social License

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

In MKKs' 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.

Since most of the Project activities are underground, the transition from exploration to operation activities should not affect significantly 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.) should help mitigate negative impacts into the socioeconomic dynamic of the general population and result in the continuing acceptance of the project. The social initiatives comprise the retraining of old artisanal miners to productive activities that would allow an improvement in their economic incomes, besides being incorporated as trained labor for the company's mining activities.

4.7 Comments on Section 4

Information from legal and MKK experts support that all mineral concessions, permits and community agreements are currently in good standing and can support estimation of Mineral Resources, Mineral Reserves and the assumptions in the FS.

To the extent known, there are no other significant factors that may affect access, title or the right to perform work on the property. There is sufficient area within the permits to site all required infrastructure for project development and operation.

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

5.1 Accessibility

Road access to the Ollachea Project is by the Interoceanic Highway, which is an international, transcontinental highway in Peru and Brazil, runs immediately east of the proposed plant site for the Project. The stretch of the Interoceanic Highway in the vicinity of the project consists of a two-lane asphalt-paved road connecting the Brazilian highway system with the south of Peru and the port of Matarani at the city of Ilo on the Pacific Coast of Peru. Portions of the highway between Macusani in the highlands to the town of Ollachea and for approximately 5 km from the town of Ollachea towards San Gaban are currently unpaved and are undergoing civil works to improve the stability of slopes over the highway. Road conditions in this interval of the highway are currently very good to moderate, with occasional closures for construction and road clearing activities.

A series of unpaved roads connect the town of Ollachea to the Minapampa area and the Oscco Cachi valley. These are currently used to support exploration drilling on the Project.

The Project can be reached by driving approximately four hours north from the airport at Juliaca, or five hours southwest from the airport at Puerto Maldonado. Both airports have daily commercial flights of one to two hours duration from Jorge Chavez International Airport in the District of Callao, immediately north of the National Capital city of Lima.

The closest deep water port is at Matarani, which is at the Pacific end of the Interoceanic Highway and is located approximately 600 km southwest of the Property. Matarani is located at the city of Ilo which is also on the Pan American highway that, except for an 87 km gap in Colombia, runs from North to South America.

5.2 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) with an average of 1,235.4 mm. 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.

5.3 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 and development activities. The involvement of the community in the recent 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 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 Rafael tin mine, owned by Minsur, a Peruvian mining company, is the nearest underground mine of reasonable size to the Ollachea Project. There are several other important underground mines in southern Peru, including mines in Arequipa and Apurimac. In general, there is a well-developed underground mining work force in Peru.

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.

Due to the construction of the Interoceanic Highway, there is excellent access to the Project for the purpose of delivery of equipment supplies and labour from the Juliaca Airport, the deep water port of Matarani and the international airport and deepwater port at Callao.

MKK has permits to draw water for exploration activities from the Oscco Cachi River and the Manticuyoc Cujo spring above Minapampa. During operation, water for plant operations is anticipated to come from mine drainage and to a lesser extent from the Oscco Cachi and Ollachea rivers.

5.4 Physiography

The Project is located at between 2,500 m and 3,450 m elevation on the eastern flank of the Cordillera Oriental of the Peruvian Andes. The physiography of the property is typical of this elevation in the Andes and consists of relatively narrow, alluvia and colluvium-filled, first order, river valleys fed by narrow *quebradas* or ravines with seasonal to year-round water flow. *Pampas* or flat areas are relatively uncommon and are frequently occupied by settlements or towns making the location of Project infrastructure a challenge. Minapampa, the site above the Minapampa mineralized zone, appears as a light green-coloured pasture at the lower right of the photograph on the left of Figure 5-1. The proposed plant site is located on another relatively flat area called Challuno, which is in the foreground of the photograph on the right of Figure 5-1.

Figure 5-1: View of Minapampa (Left) and the Proposed Plant Site Location at Challuno (Right)

5.5 Comments on Section 5

The Ollachea Gold Project is easily accessible and well connected to the national highway system and has reasonable access to resources at the local and national level to carry out exploration, mine development and operation.

The climate at Ollachea is moderate and exploration and development activities can be carried out year round.

The physiography of the Ollachea Property is challenging; however, well-located sites with favourable topography for the construction of a mineral processing plant and mining and tailings infrastructure have been identified and surface rights for these areas have been acquired by MKK.

MKK has negotiated surface right agreements with the Community of Ollachea covering an area of 5,998.9848 ha of the Oyaechea 3 concession for a period of 30 years starting 30 May, 2012. MKK is currently in negotiations for surface rights for the proposed dry-stack tailings storage facility. Thus far, up to 30% of the proposed tailings area surface rights have been negotiated.

The project schedule assumes all surface rights can be obtained, and while MKK considers the acquisition to be doable, if any major delays are encountered, the project schedule assumptions could be affected.

6 HISTORY

The earliest evidence of mining on the Ollachea Project is attributed to Spanish colonial activity during the 18th century. Informal mining activity has been pursued in the area since at least the 1970's and probably considerably earlier.

Between 1998 and 1999, Peruvian Gold Ltd., a publicly-traded Canadian exploration company, drilled five diamond drill holes on the Project and encountered low-grade gold mineralization but did not do any further work.

In May 2003, Rio Tinto re-discovered the area while following-up a regional stream sediment sampling program. Between 2003 and 2004, Rio Tinto carried out surface sampling, encountering encouraging surface sample gold assays but in 2006 elected to farm out the project.

IRL started negotiations with Rio Tinto in 2006, which were followed by the negotiation of an Agreement of Use of Surface Lands and another related to Artisanal Mining Exploitation with the Community of Ollachea, signed in November 2007, after which exploration works started over the property.

In 2007, the Community of Ollachea and MKK worked to formalize mining at Minapampa under the national Act of Formalization and Promotion of the Little and Artisanal Mining Industry and its regulations (Tong, 2010b). MKK granted the Community of Ollachea right to exploit near surface mineralization at a part of the Minapampa area for five years in exchange for surface rights to carry out exploration activities on a portion of the property (Tong, 2010b). On 30 May, 2012, this surface rights agreement was extended for a period of 30 years. Small-scale artisanal mining continues on the Project (Figure 6-1).

Beginning with field activities in early 2008, MKK carried out bedrock sampling, geochemical sampling, mapping and structural geology based on aster image interpretation (Telluris, 2009). By the end of September 2009, 71 diamond drill holes totaling 26,026 m had been drilled, and a Mineral Resource estimate and Preliminary Assessment was carried out for the Project by Coffey Mining (Coffey, 2010).

MKK continued diamond drilling and, in mid-2010, contracted AMEC to assist with a Prefeasibility Study for the Project. By November 2010, an additional 60 drill holes for a total of 131 drill holes totaling 51,062 m had been drilled and the Mineral Resource estimate for the Property was updated (Coffey, 2011a).

Figure 6-1: Artisanal Mine Workings at Minapampa - October, 2010

Between October 2010 and May 2011, MKK completed 26 more core drill holes totalling 11,143 m. At this stage, a Prefeasibility Study Mineral Resource estimate for the Minapampa Zone, based on 120 drill holes totalling 46,404 m, was completed. The results of the Ollachea Prefeasibility Study were announced in an IRL Press release dated 18 July, 2011.

An extended period of exploration drilling from May 2011 was followed by another infill drill campaign by MKK on the Minapampa Zones to end of March 2012, which added another 49 core drill holes totalling 17,904 m. By this time, 206 drill holes totalling 80,109 m had been completed on the Ollachea Project. The database provided to Coffey Mining for the Feasibility Study resource update included information taken from this drill hole database. Subsequent to the provision of the resource data to Coffey Mining, 2 additional drill holes were completed for a project total of 208 diamond drill holes totalling 81,073 m in length.

This Feasibility Study Report includes an updated Mineral Resource estimate for the Minapampa Zones (effective date 6 July, 2012) based on the Minapampa Mineral Resource database to end-April 2012 (151 drill holes for 59,509 m). The results of the Ollachea Resource Upgrade to be used for the Minapampa Feasibility Study were announced in an IRL Press release dated 18 July, 2012

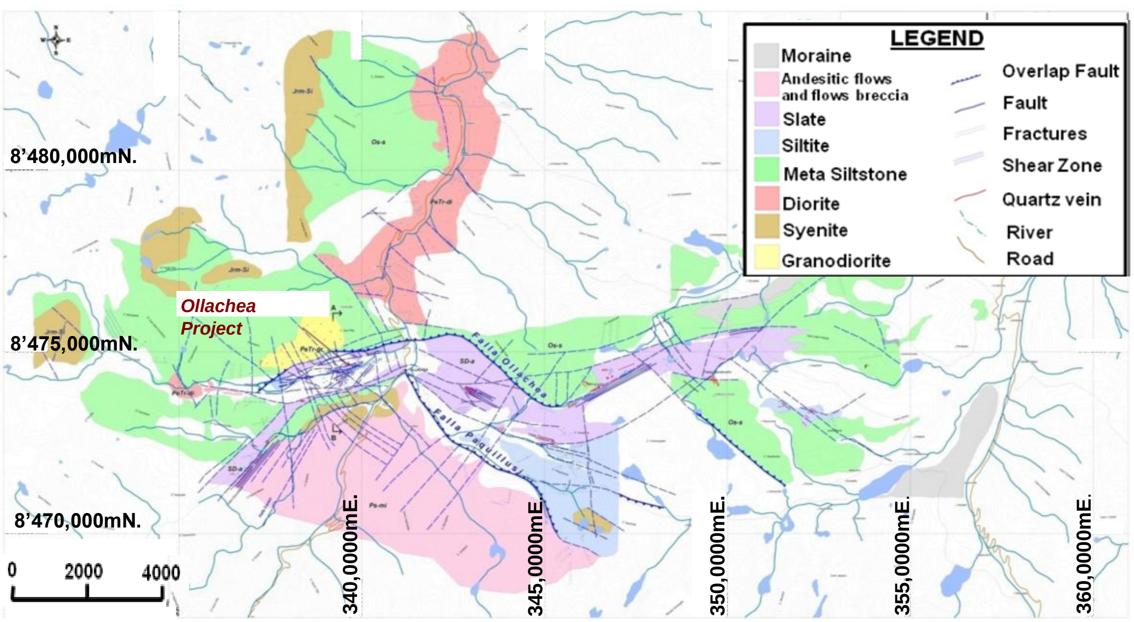
7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The regional setting of the Ollachea Project is characterized by a significant change in the strike of the Andean range, whereby the stratigraphy is locally aligned approximately east-west, as opposed to the dominant northwest Andean trend. This deflection is postulated to have resulted from significant compression and thrusting to accommodate a prominent portion of the adjacent Brazilian Shield located to the east.

On a regional scale, high-grade gold deposits occur almost exclusively in slates/phyllites, (usually carbonaceous), and rarely in more arenaceous sediments but only when they lie adjacent to mineralized phyllites. This suggests that there may be a regional control on pre D1 syngenetic gold in sulphides that has been upgraded in areas of strong overprinting D1 deformation. Figure 7-1 shows the regional setting with respect to the Project.

Figure 7-1: Regional Geology of the Ollachea Project

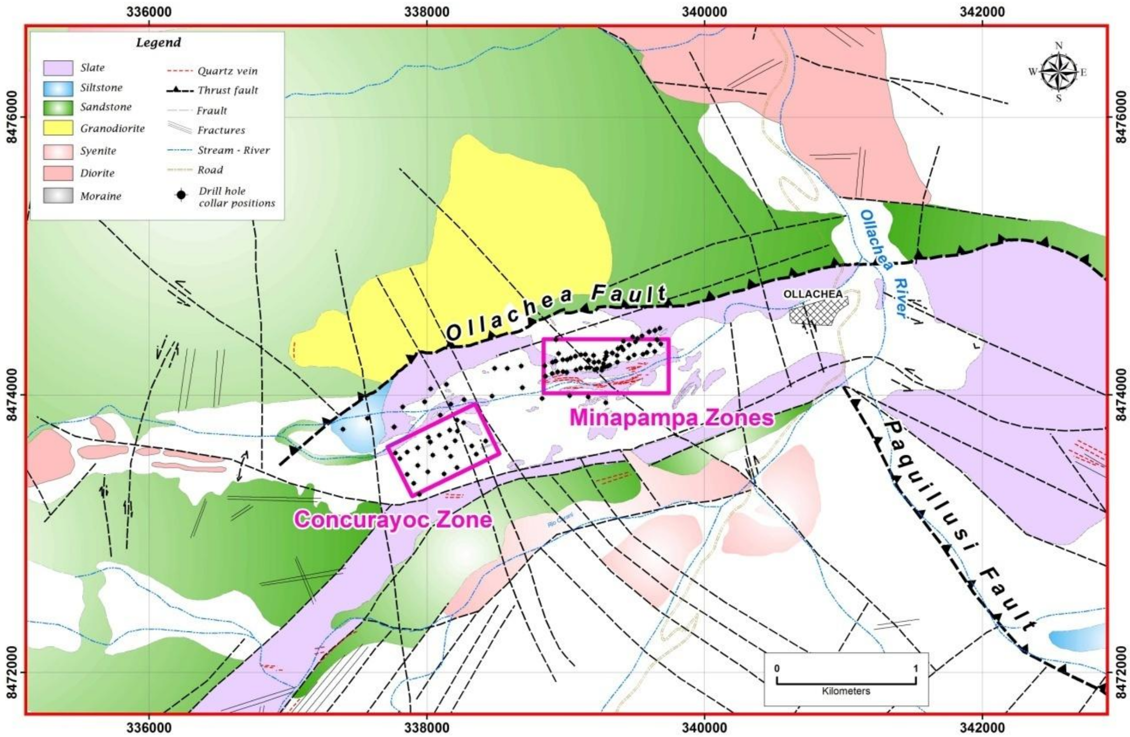


(after Ing. Valdivieso, Y., MKK, 2008. Regional Map of the Ollachea Project. 1:50,000 scale) Based on previous PSAD56 grid)

7.2 Project Geology

The geology of the Ollachea Project is dominated by phyllites of the Devonian Sandia Formation, and variably bedded graphitic slates and shales of the Siluro-Devonian Ananean Formation. Figure 7-2 shows the surface geological plan and limits for the area including the Minapampa and Concurayoc deposits.

Figure 7-2 Surface geological plan and limits for Ollachea area deposits



Andesitic volcanic rocks crop out south of the sedimentary units and both the sedimentary and volcanic rocks are intruded by nepheline syenite to the south and granodiorite to the north. Intra-formational contacts and a strong penetrative cleavage in the sedimentary package of rocks are oriented approximately east-west and are parallel to two regional-scale thrust faults that bound the phyllitic slates which play host to the gold mineralization at Ollachea (Figure 7-3).

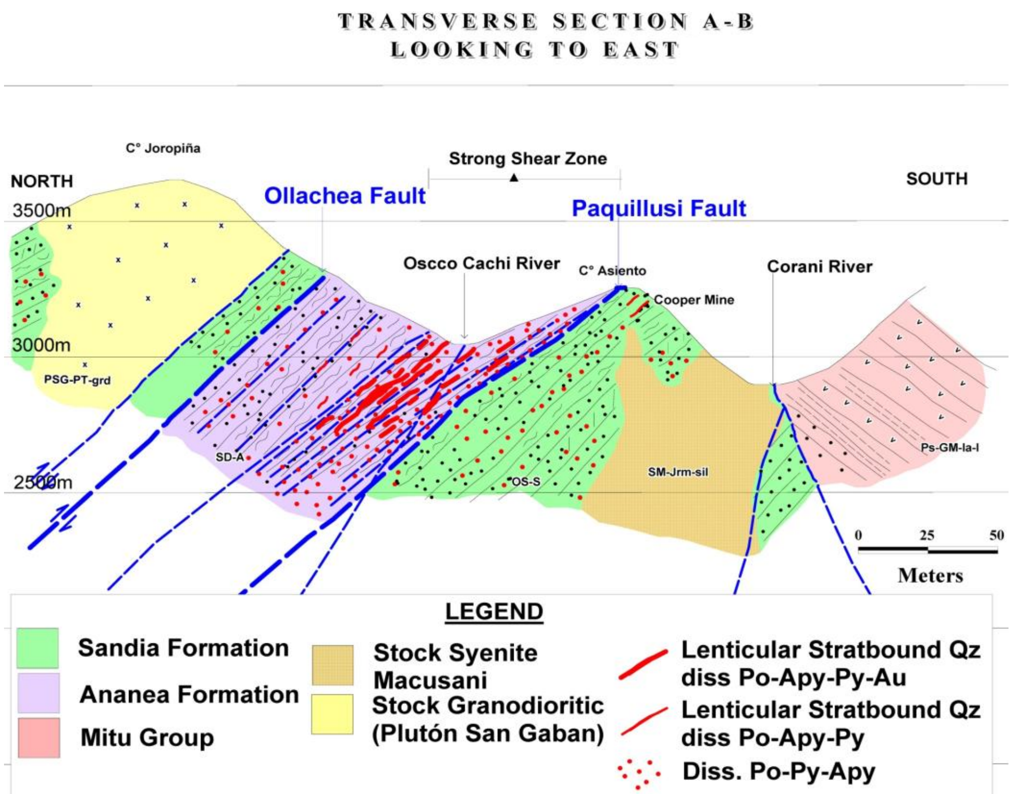
The gold mineralization at Ollachea is broadly strata-bound within northeast to east-west-trending, north-dipping carbonaceous phyllites. Two principal tectonic events are recognized in the Ollachea District:

- D1 – this first event is the deformation of the slate sequence resulting in the localized thrusting of the underlying Sandia Formation over the Ananea Formation.
- D2 – the second phase of deformation is the start of the deformation of the Andean belt (late-Triassic approx. 220 ±10 Ma).

The D1 event consisted of northwesterly - to southeasterly-directed compression forming northeasterly striking zones of shearing, folding and thrusting. Gold mineralization is associated with the D1 event.

The Ollachea and Paquillusi Faults occur as district-scale northward-dipping thrust faults which bound the prospective generally east-west striking sheared sedimentary sequences that host the Ollachea mineralized gold deposits. The Ollachea thrust fault has been intersected in exploration drill holes and tunnel access development and is known to attain widths of up to 20 m in thickness. Although intensely sheared, the fault zone consists of compressed slate material which is relatively impermeable to water infiltration.

Figure 7-3: Schematic Cross Section of the Ollachea Deposit



(after Ing. Valdivieso, Y., MKK, 2008. Schematic Transverse Section looking East, Ollachea Project. 1:50,000 scale) based on previous PSAD 56 grid Qz = quartz; Po = pyrrhotite; Apy = arsenopyrite; py = pyrite; diss. = disseminated.

A later deformation event (D2) consisted of a prolonged stage of compression oriented north-northeast to south-southwest that formed principally reverse faults striking west-northwest, which folded the Ollachea District into the form of a dome structure and changed the orientation of the slates in the central area of the District to an almost east-west strike.

7.3 Deposits

7.3.1 Mineralisation

The principal known Minapampa zone of mineralization comprising the Ollachea Prospect is being extensively worked from surface by artisanal miners (Figure 7-4).

Delineated gold mineralization occurs within seven discrete east-striking, north-dipping structures below Minapampa and on the north side of the Oscco Cachi River. Mineralization has been traced continuously for 900 m along strike. Gold mineralization has also been encountered to the west of the Minapampa Zone in a zone on the south side of the Oscco Cachi River that is referred to as Concurayoc, located some 400 m west of Minapampa. The known mineralized zone at Ollachea, including both the Minapampa and Concurayoc Zones as well as the intervening 400 m of strike between the two mentioned deposits, is approximately 1,900 m long, up to 200 m thick and has been traced in places to over 400 m below surface and remains open along strike as well as at depth.

An extensive shear zone hosts the gold mineralized horizons (Figure 7-4). The shear zone is characterized by a well-developed slaty cleavage, with quartz-sulphide veins and veinlets, broadly concordant with the slaty cleavage. Quartz-sulphide veins and veinlets vary from a few millimetres to centimetres wide, up to a maximum of 40 cm, but do not always contain gold mineralization. The gold mineralization is hosted both within as well as along contacts of the quartz -sulphide veins and veinlets. The veins can be strongly boundinaged, resulting in the development of packages of irregularly mineralized veins and veinlets within discrete mineralized horizons, incumbent to the sheared slate package.

The phyllitic carbonaceous slate rocks hosting the Ollachea gold mineralization had previously (Prefeasibility Study) been categorized as a series of discrete lithological horizons (Pz1-Pz6), on the basis of the presence and nature of quartz and sulphide content. Current re-evaluation of the Ollachea core recognizes that the slate lithology is a constant and thus sulphidic and quartz content is now included as a variant within appropriate columns on the relevant logging sheets.

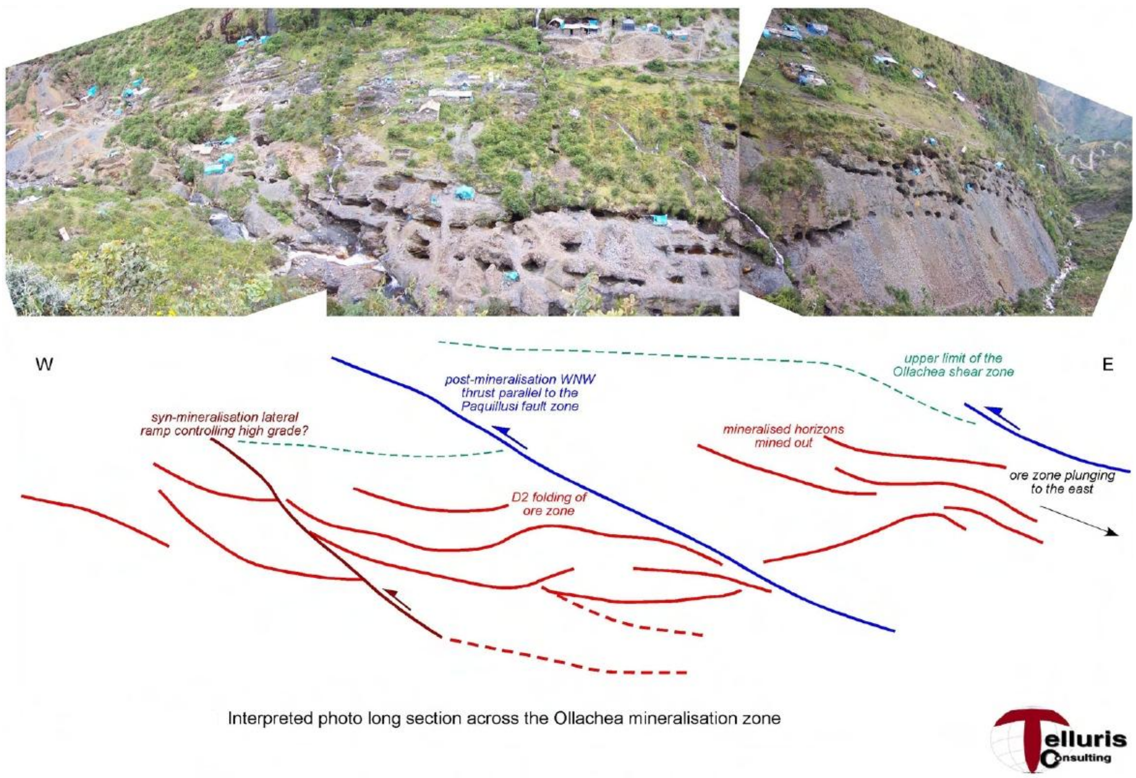
As a consequence of this refinement, the slate horizons previously classified as Pz1 – Pz6 (Ollachea Prefeasibility Study), are now grouped into only one horizon. A three-dimensional array of erratically occurring, generally narrow, discontinuous lenticular horizons of more carbonaceous or graphitic content are hosted within the metamorphosed package of fine-grained slates.

Slates previously classified as Pz5 are now classified as a fine-grained siltstone horizon, also with inconsistently formed, generally narrow, lenticular horizons of carbonaceous or graphitic content, but of higher relevant content than the over-lying slates.

The lithological base to the mineralized component of the Ollachea deposit, when it has been intersected where deeper exploration drill holes have traversed the entire mineralized package, has been assumed by a compact dark green-grey quartzite horizon consisting primarily of quartz grains and granuloblastic mica. This lithological horizon may correlate with the upper Sandia Formation.

The mineralized slate package also hosts an irregular assemblage of intercalated schistose, narrow sill-like structures. There is doubt surrounding the origin of these structures that occur as greenish-grey, granoblastic to lepidoblastic textured features comprised of around 40% chlorite, 5-7% quartz and some sericite. An incipient carbonate alteration superimposes the chloritic alteration of amphiboles.

Figure 7-4: Structural Interpretation of Minapampa Zone



(Telluris Consulting Ltd, 2009)

Alteration of the slates and phyllites is weak. Mild sericitisation is observed in the area, but has no apparent correlation with gold mineralization.

Gold mineralization is associated with a sulphide assemblage consisting predominantly of pyrrhotite with minor pyrite, arsenopyrite and traces of chalcopyrite. Coarsely crystalline arsenopyrite and free gold are frequently observed in close association with one another within the central Minapampa zone. The occurrence of coarse pyrite without other sulphides is often a counter-indicator of gold mineralization.

At Concurayoc, as opposed to Minapampa, only six discrete gold mineralized horizons have been delineated by drilling. The mineralized structures strike essentially southwest-northeast and dip towards the north. Characteristics of the gold mineralization are very similar to those observed at Minapampa with a sole exception being that the economically mineralized horizons are not as wide as some of those at Minapampa. Mineralization has been traced continuously for 700 m along strike and up to 400 m in depth. The mineralized horizons remain open-ended in strike as well as at depth.

Data on faults and fractures from the structural logging of the drill holes has been adequately interpreted in order to obtain a good structural correlation.

An orientated drill core (DC) study, on 18 DC (DDH10-102 to DDH10-119) was completed; the test was run from 50 metres before the projection of the mineralized zone, as identified in the project area, to the end of the hole. Then the alpha and beta angles of the foliations, faults, fractures, veinlets, micro veinlets and other outstanding structures were recorded over the core.

The results of azimuths and dips from oriented core mostly match those as recorded from surface exposures. A high predominance of structures have azimuths between 270° - 300° and dips between 40° - 60°.

There is an alignment of the mineralization relative to the foliation where favourable horizons continue. This information was also used to help interpret the mineralized zones.

8 DEPOSIT TYPES

The deposit model guiding exploration targeting is mesothermal quartz vein style gold mineralization. Dr. Noel White (2011) and other consultants (e.g. Telluris, 2009) have also described the Ollachea deposit as a member of the class of orogenic gold deposits, with the possibility of local syngenetic gold enrichment playing a role in the location of the mineral deposit. This variety of gold deposit can also go by the name slate belt gold deposit and can be both very large and very rich.

MKK is using and actively modifying an exploration model developed by Telluris Consulting (2009) which describes the main stage of gold mineralization at Ollachea as being associated with a D1 event comprising shearing and folding and largely confined to the weaker carbonaceous shales along a brittle-ductile shear zone. The absence of main stage D1 mineralization outside the graphitic phyllonites of the Ananea Formation and comparison with other deposits in the region suggests that there may be some degree of possible pre-shearing concentration of gold within the syn-sedimentary pyrite.

Exploration drilling by MKK targets mineralization along strike and down dip within the sheared carbonaceous shale package.

8.1 Comments on Section 8

This model is supported by field evidence at Ollachea and is suitable to guide continued exploration efforts. Furthermore, the model is considered suitable for application in the resource and reserve estimation process.

9 EXPLORATION

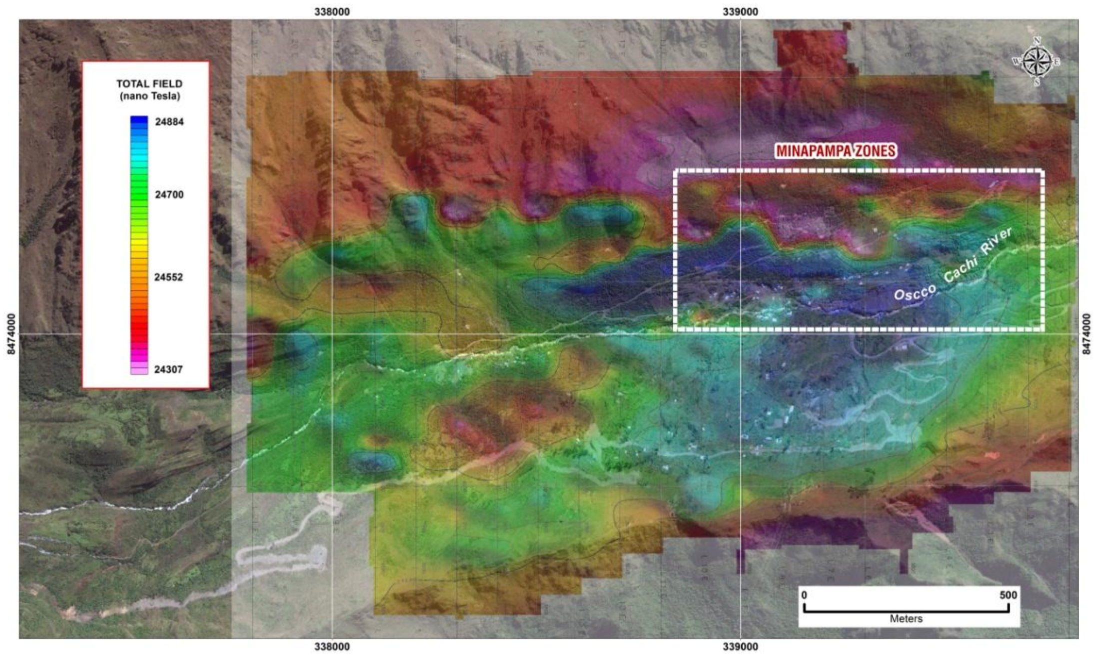
Core drilling has been the dominant exploration tool of MKK. During 2008, geological mapping, geochemical sampling and geophysical ground magnetic studies, together with an Aster satellite imaging and a structural geology targeting exercise completed by Telluris Consulting in September 2009, have additionally contributed to understanding of the potential of the Project. Concerning the definition of the true potential of the Ollachea deposits, no exploration results have been more convincing than those obtained from the results of exploration diamond drilling.

To illustrate a point, it was not until analytical results from an initial line of exploration drill holes had been received, that MKK staff could begin to obtain an appreciation for the major potential of this gold project. Prior to drilling in late-2008, several routine early-stage exploration activities and studies were completed. These included detailed geological mapping over an area of 784 hectares and geochemical sampling over an area of some 630 hectares. Geochemical sampling included the collection of 362 rock chip samples from surface outcrop and systematic geochemical sampling of some 1,623 meters of trenching and surface channel sampling (on an approximate 50 m spacing), across the main body of outcropping structures, as they were recognized at surface. All geochemical samples were analysed at independent analytical laboratories for gold (50g aliquot by FA with gravimetric finish) as well as a full suite of 36 elements (by ICP).

In isolation, the results obtained from the above-mentioned activities were in reality insufficient to justify a decision to undertake a serious exploration investment into diamond drilling. As part of the company’s 2008 exploration strategy, during the completion of the mentioned exploration activities, tests were conducted in the field to evaluate whether IP Chargeability/Resistivity studies would provide a useful response from the mineralized structures. It transpired that the host carbonaceous slate-belt sediments were highly conductive and absorbed all emitted IP energy which resulted in inadequate response to IP testwork.

At this time, due to the presence of magnetic pyrrhotite observed in association with mineralized structures which were exposed to surface, a decision was taken to conduct a ground magnetic survey over the central area of interest at Minapampa. The prospective carbonaceous slate sediments provided a strongly anomalous magnetic response (Figure 9-1). The results of this 24.1 Km survey (21 lines at 100 m spacing) transpired to be the turning point in MKK’s appreciation of whether or not to drill and of where to locate a first line of exploration drill holes. First exploration holes were located, on a spacing which varied between 100 to 200 m, along the high magnetic anomaly obtained on the Northern side of the Oscco Cacchi River.

Figure 9-1 Anomalies resulting from ground magnetic survey at Ollachea



The first exploration drilling started at the beginning of October 2008 and by the end of November 2008, the Company had advised the investment market of the encouraging gold intersections being obtained from the Ollachea drilling. By the end of 2008, 17 diamond drill holes had been completed at Ollachea for a total of 4,787 mfetres. A short time after this a significant gold discovery had been confirmed and the Company emitted a press release in April 2009 confirming the Ollachea gold discovery on the back of 26 drill holes for a total of 8,706 m of drilling.

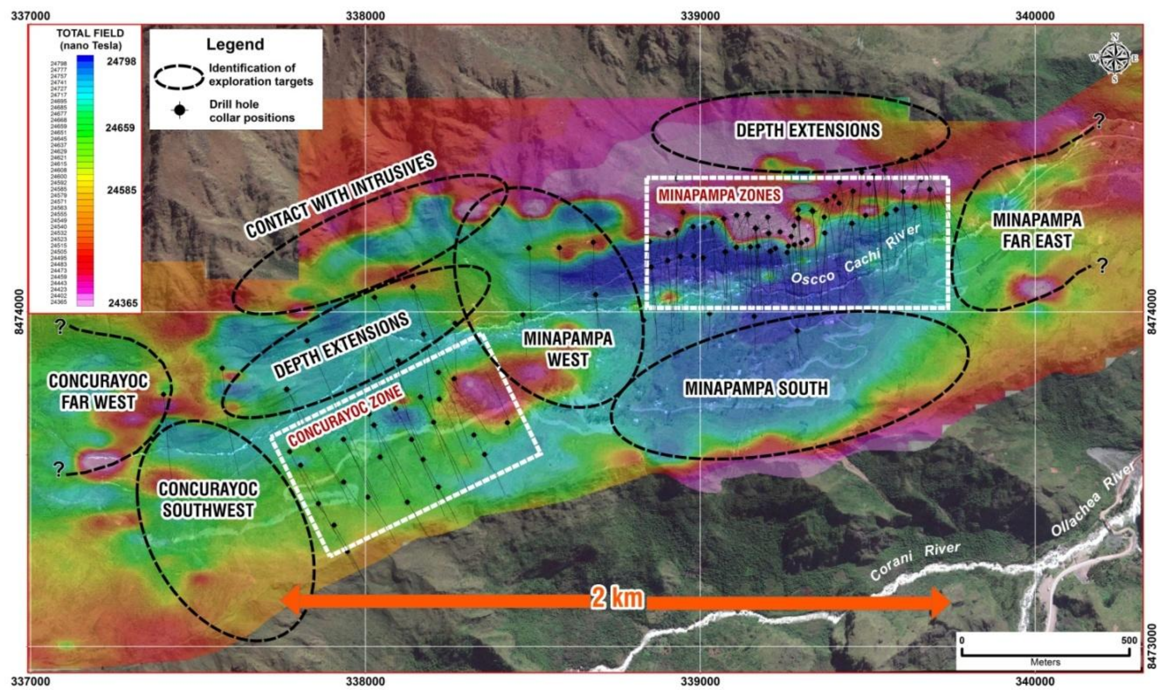
Since that time exploration diamond drilling has been the main exploration tool at Ollachea.

Exploration surveys and interpretations completed to date within the Project have largely been planned, executed, and supervised by national MKK personnel, supplemented by consultants and contractors for more specialised or technical roles. The data is considered to be of good quality.

9.1 Exploration Potential

Coffey Mining considers that the exploration targets identified to date (Figure 9-2), including extension, step-out and conceptual targets, justify further follow-up. The deeper, down-dip potential of Ollachea and the eastern extension of Minapampa will be better targeted from the underground exploration drive (in progress), as diamond drilling from surface will require >1 km holes due to the elevated topography north of the main northward-dipping mineralization.

Figure 9-2 Ollachea exploration targets including extension, step-out and conceptual targets located nearby the Minapampa and Concurayoc Zones.



New discoveries such as the Concurayoc Zone, displaced by some 400 m from the main Minapampa Zone, have additional exploration potential. All mineralization discovered to date at Ollachea remains open-ended along strike as well as down-dip.

10 DRILLING

At the database closure date on 23 April, 2012, there were a total of 172 holes totalling 67,298 m in the area around the Minapampa zone. A total 155 of these drill holes immediately adjacent to the Minapampa zone, totalling 60,306 m, were used to construct the Mineral Resource model used in this study.

Going back to Concurayoc, the database closure date was 08 August, 2011. At that time there were a total of 45 drill holes totalling 16,943 m in the area of Concurayoc. All drill hole information from these holes was used in the construction of the Inferred Mineral Resource model as applied to Concurayoc.

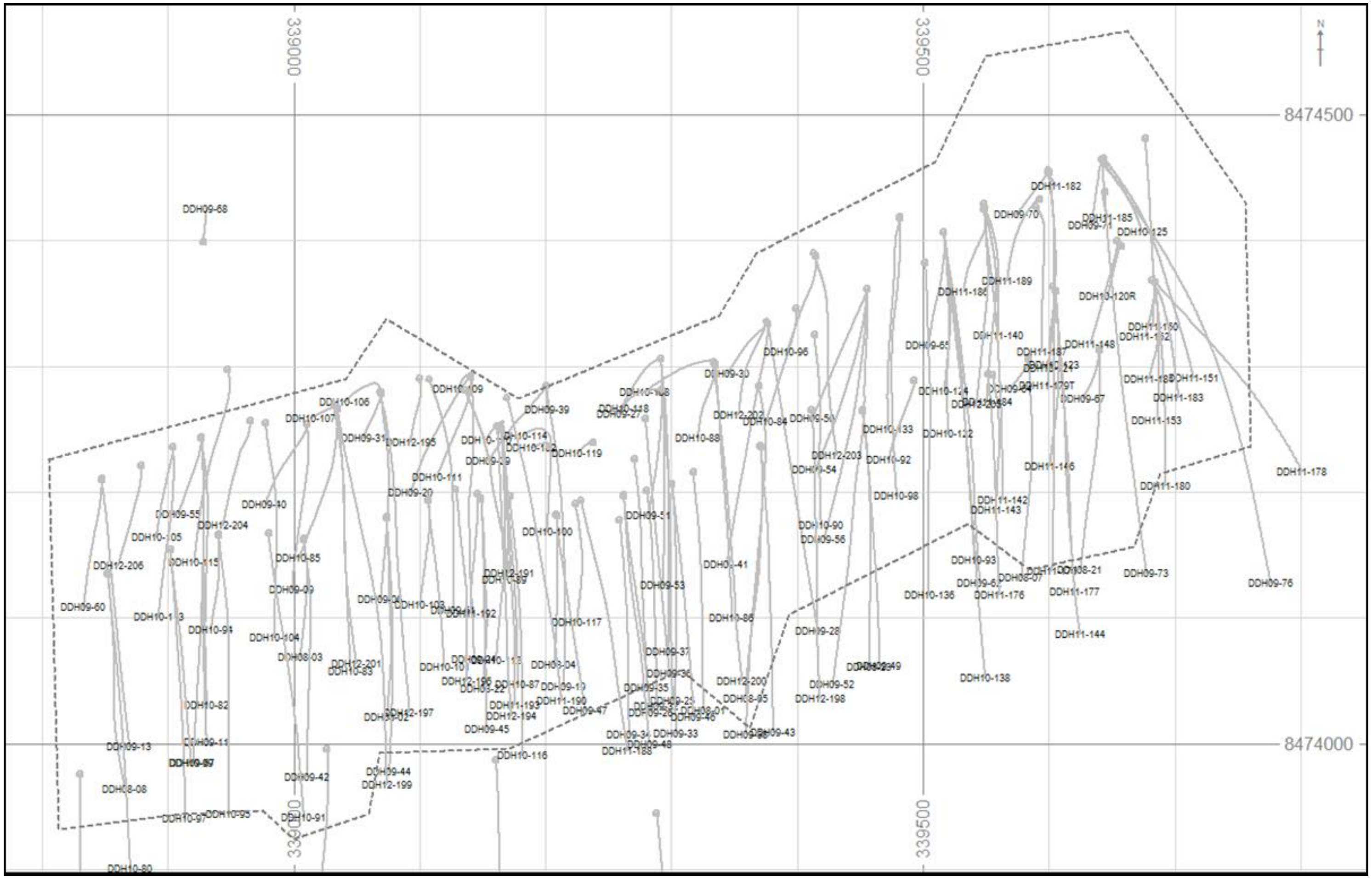
The principal methods used for exploration drilling at Ollachea have been diamond core drilling (DC) by MDH SAC (drilling company), using standard wire-line diamond drilling of HQ diameter then reducing to NQ then BQ as ground conditions dictate. Core recovery was very good (greater than 99%), except in large fracture zones where recovered core is noticeably fractured, but these zones are not expected to have a material impact on the accuracy and reliability of the results for the mineral resource estimate.

All surveying, plotting and mineral resource modelling utilises the National Geodesic Network grid in the UTM WGS 84 (Zone19S) coordinate system. The grid was changed from the previous PSAD-56 coordinate system to conform to Peruvian Law. It was also noted that magnetic declination correction was not applied to the previous down hole drilling survey data over the years; this has been corrected in the current drilling database used for this updated Mineral Resource estimate. All previously mineralised zones were re-interpreted to the new spatial location, resulting in minimal volumetric changes to previous interpretations.

Limited information is known of the late-1990's Peruvian Gold drill program, and the information has not been used in the Mineral Resource estimate.

Figure 10-1 shows a plan of the drill traces of exploration and resource infill drill holes in the vicinity of the Minapampa Zone.

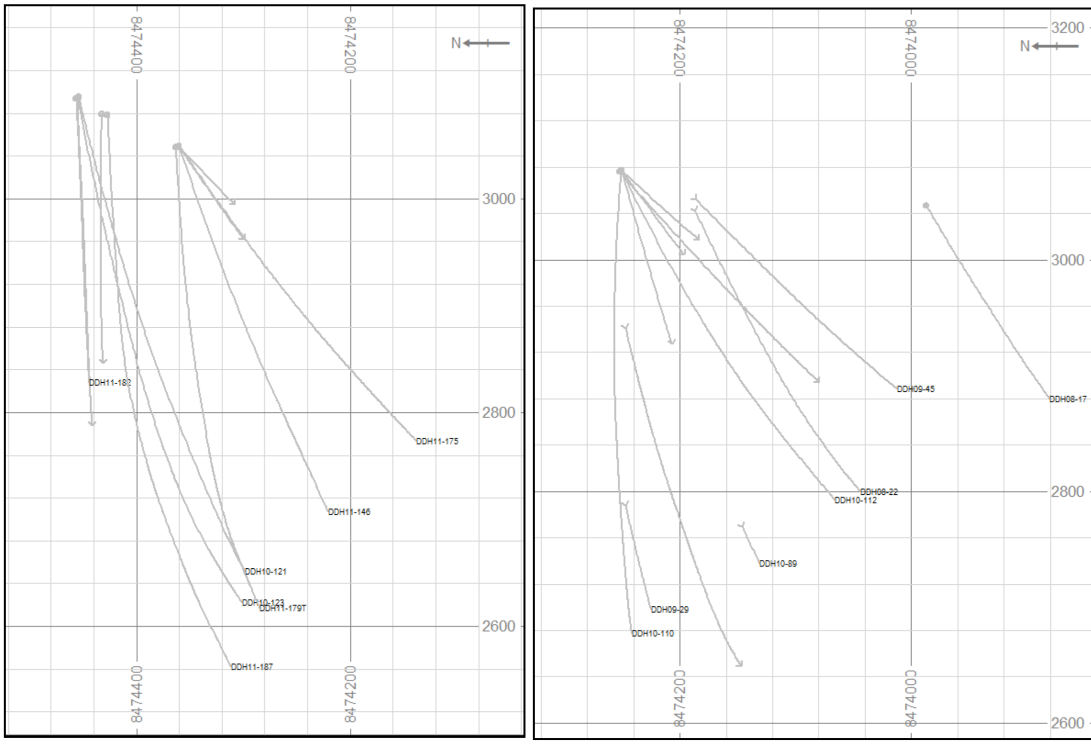
Figure 10-1: Plan View Minapampa Exploration and Resource Infill Drill Hole Location Map



Note: Drill collars are marked with grey dots. Drill hole traces are solid grey lines. Drill hole names are marked at the toe of the holes. A plan projection of the Mineral Resource limits is marked with the dashed line

Figure 10-2 shows two representative sections of drill traces of exploration drill holes in the Minapampa Zone.

Figure 10-2: Representative Sectional View - Exploration and Resource Infill Drill Hole



Note: left plot is at 339600mE (+/- 10m) and the right plot is at 339160mE (+/- 10m). Grid shown on plot is 40 m x 40 m

Figure 10-3 shows drill hole collar locations for exploration, resource database, resource and metallurgical sample drill holes, geotechnical drill holes and pits and drill holes for hydrogeology. Table 10-1 summarizes pertinent drilling statistics. The Minapampa zone has been drilled at a nominal spacing of 30 m by 30 m.

Figure 10-3: Drill Hole Location Map

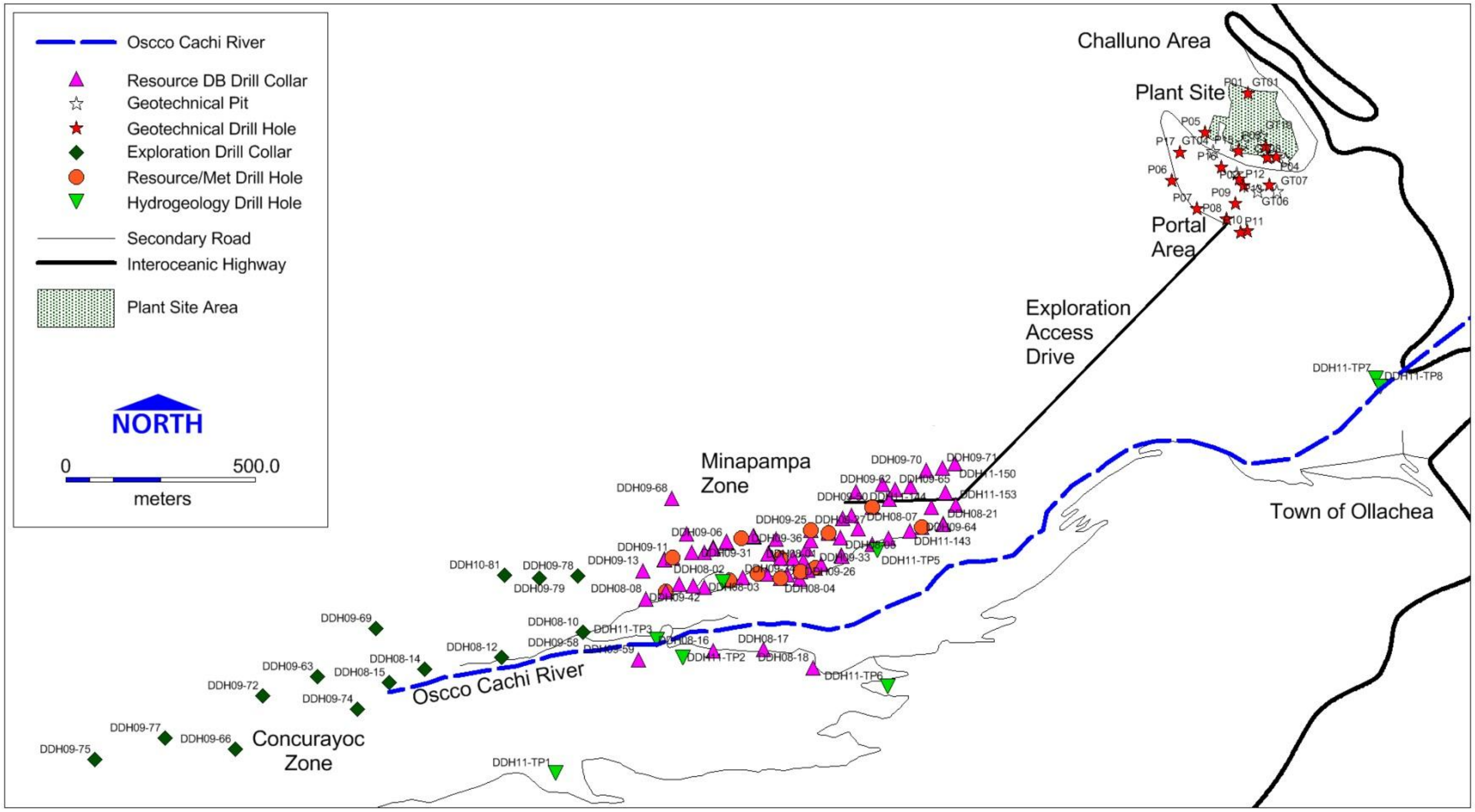


Table 10-1: Drilling and Sampling Campaigns at Ollachea (Minapampa & Concurayoc Areas)

	Peruvian Gold Ltd.	MKK 2008	MKK 2009a	MKK 2009b	MKK 2010	MKK 2011	MKK 2012
Period	1998-1999	September 2008 - April 2009 DDH08-01 to DDH09-33	April 2009 to June 2009 DDH09-34 to DDH 09-43	June 2009 to October 2009 DDH09-44 to DDH09-73	October 2009 to August 2010 DDH09-74 to DDH 10-125	October 2010 to April 2011 DDH10-120R, DDH10-133 to DDH11-153	August 2011 to March 2012 DDH11-175 to DDH12-206
Drill Holes	5	33	10	30	52	14	32
Total Length (m)	501	11,773.4	3,270.05	12,138.8	20,636.55	6,372.4	13,106.6 (NOTE: does not include 420.15m drilled in deviations from the principal drill holes during this period).
Resource DB	No	Yes	Yes	Yes	Yes	Yes	Yes
Sample length	Unknown	Running 2 m samples	By lithology			By lithology	
Standards	Unknown	8001, 8002, 8003, 8004*	8001, 8002, 8003, 8004*	8002, 8003	8006, 8007, 8008, 8009	8006, 8007, 8008, 8009	8006, 8007, 8008, 8009
Blanks	Unknown	8005	8005	8005	8005, commercial blank	commercial blank	8017, commercial blank
Field Duplicates	Unknown	Yes, 1/4 core, not blind	Yes, 1/4 or 1/2 core, blind	Yes, 1/4 or 1/2 core, blind	Yes, 1/4 or 1/2 core, blind	Yes, 1/4 or 1/2 core, blind	Yes, 1/4 or 1/2 core, blind
Preparation Duplicate	Unknown	None	From DDH09-42	Yes	To DDH09-80	No	No
Pulp Duplicate	Unknown	Yes	Yes	No	No	No	No
Referee Lab Analysis	Unknown	Inspectorate	Inspectorate	ALS	ALS	ALS	ACTLABS
Re-sampling	Unknown	The majority of samples greater than 1 m length in mineralized zones were re-sampled at 1 m lengths to hole DH10-119				No	Re-sampled at 1 m lengths (some mineralised zones greater than 1m)
Screen Fire Assay	Unknown	122 samples to DH09-25	244 assays of samples with greater than 1 g/t Au from DH09-26 to DH11-107			No	No

10.1 Drill Methods

All diamond drilling used in previous Minapampa and Concurayoc mineral resource estimates and specifically in the July 2012 Minapampa Mineral Resource estimate was completed by the MKK contractor. Most diamond core holes were drilled using HQ and reducing to NQ diameter. There was one BQ diameter hole drilled, only one (2 m interval) sample was located within the Minapampa defined mineralized zone.

Based upon inspection of various core trays available on site and review of the available reports, Coffey Mining considers that diamond core drilling has been carried out to expected industry standards.

10.2 Geological Logging

Diamond core was logged in detail for geological, structural and geotechnical information, including RQD and core recovery. Whole core was routinely photographed. Review by Coffey Mining of selected geological logs against actual core showed no significant discrepancies or inconsistencies.

Diamond core logging of geological, structural and geotechnical information has been completed using conventional and industry-standard methods, and is appropriate to the deposit type.

10.3 Recovery

High drill core recovery has been generally noted at the project (greater than 95%). Some shear zones have recorded lower recoveries, but these zones are typically less than 1 m wide and poor core recovery where it has occurred in these zones is not expected to affect the mineral resource estimate.

10.4 Collar Surveys

Drill hole collars were surveyed by MKK surveyors using total station instruments. Survey accuracy is reported as +/-0.5 m.

Accuracy of the survey measurements meets acceptable industry standards.

While on site in 2010, Coffey Mining chose several drill collars and verified their location using a hand-held GPS unit. All drill holes checked were within +/- 5 m of the reported location (within the accuracy limits of the device).

The UTM WGS84 datum is used to plot drill holes and other information in the Mineral Resource Estimate and Feasibility Study except where noted.

10.5 Down hole Surveys

Down hole surveys have been undertaken by the contract driller utilising both a Reflex single shot and a multi-shot survey tool, with readings taken on average at 20 m down hole interval depths.

On validating the database, the original survey certificates for holes DDH08-01 and DDH08-02 were not located. The survey coordinates within the database provided by MKK were used. On inspecting these holes spatially, there was good correlation from surrounding drilling and correlation of results. They were, therefore, used for the resource estimation.

Magnetic declination adjustment has been applied to all down hole surveys, on the basis of the year drilling was completed.

Accuracy of the down-the-hole survey measurements meets acceptable industry standards.

10.6 Geotechnical and Hydrological Drilling

10.6.1 Geotechnical Drilling

In 2012, eight geotechnical drill holes were drilled to depths of 30 to 40 m and nine test pits were excavated in the proposed plant site and lower portal waste dump areas. Additionally, six test pits were excavated around the proposed paste plant and upper portal area.

10.6.2 Hydrogeological Drilling

Eight hydrogeological drill holes were drilled in 2011 (Table 10-1). Hydrogeological holes range from 20 m to 150 m in depth and were drilled in the Oscco Cachi valley and above Minapampa.

Table 10-1: Hydrogeological Monitoring Drill Hole Locations

Borehole	East (m)	North (m)	Dip (º)	Elev. (msl)	BoreholeDepth (m)
DDH11-TP1	338,847.2	8,474,094.1	90	3,216.77	95.25
DDH11-TP2	339,190.7	8,474,390.1	90	3,037.97	20.00
DDH11-TP3	339,123.0	8,474,437.7	90	3,041.00	20.00
DDH11-TP4	339,300.8	8,474,586.7	90	3,066.13	150.00
DDH11-TP5	339,709.4	8,474,659.7	90	3,040.53	150.00
DDH11-TP6	339,726.8	8,474,299.6	90	3,055.85	103.00
DDH11-TP7	341,033.4	8,475,078.0	90	2,680.53	20.00
DDH11-TP8	341,043.4	8,475,055.2	90	2,681.84	20.00

* The coordinates are referenced in PSAD 56 Datum, Zone 19S and were provided by MKK.

In order to characterize the hydrogeology of the Minapampa area, three additional vertical boreholes were drilled in 2012 to a depth of approximately 150 m, along with two horizontal boreholes to lengths ranging from 200 to 300 m. Table 10-2 shows the locations and general characteristics of the additional Minapampa hydrogeological boreholes.

Table 10-2: Vertical and Horizontal Hydrogeological Drill Hole Locations

ID		East (m)	North (m)	Elevation (msl)	Azimuth (°)	Dip (°)	Length (m)
TP-09		339 293.7	8 474 213,8	3,060	-	-90	150
TP-10		339 378.3	8 474 261,1	3,055	-	-90	140
TP-11		339 282.2	8 474 302,5	3,076	-	-90	160
BHH-01	Inicio	339 582.5	8 474 151,3	2,923	290	0	300
	Final	339 301	8 474 254				
BHH-03	Inicio	339 582.5	8 474 151,3	2,923	330	0	200
	Final	339 483	8 474 325				

* The coordinates are referenced in UTM WGS84, Zone -19.

10.7 Metallurgical Drilling

Three metallurgical sampling campaigns were carried out to support process flow sheet development. Metallurgical samples were taken from exploration drill holes. The first campaign was composed of samples from mineralized horizons in drill holes:

- DDH08-04
- DDH08-22
- DDH09-25
- DDH09-26

The second metallurgical sampling program consisted of samples from drill holes:

- DDH09-44
- DDH09-45
- DDH09-46
- DDH09-52
- DDH09-53
- DDH09-54
- DDH09-57
- DDH09-61

The third metallurgical sampling campaign was composed of samples from drill holes:

- DDH09-64
- DDH10-86
- DDH10-97
- DDH10-100
- DDH10-129

10.8 Sample Length/True Thickness

Assay samples for the mineral resource database have been taken at 0.3 m to 5 m lengths within the known mineralized zones (samples of 2 m to 5 m lengths have been taken in the surrounding non-mineralized areas) and have an average length of 1.33 m (the median length is 1 m).

Exploration drill holes used in the mineral resource estimate were generally drilled to the south at between 40 degrees to 90 degrees dip. At different depths below the surface, holes were targeted to perpendicularly intersect the main trend of mineralization. Given the access from surface to deeper sections of mineralization, several of the deeper intersections are oblique to mineralization. The deeper sections of Ollachea will need to be targeted from underground or via >1 km surface directional drilling. The Minapampa zone has been drilled at a nominal spacing of 30 m by 30 m.

The relationship between exploration drilling used in the mineral resource estimate and mineralization is defined in further detail in Section 14. Drill holes typically intersect mineralization orthogonally, and the mineralized intercepts range between 60% and 100% of the true mineralized thickness (averaging approximately 90% of true mineralized thickness). There is no other drilling, sampling, or recovery factors known that could materially affect resource estimation.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

Sampling has been carried out using a series of different procedures since MKK began drilling on the Project. Sampling lengths have varied from fixed 2 m lengths within mineralized zones and fixed 5 m lengths outside mineralized zones to sampling lengths 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 are still a minor amount 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 lengths. 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 taking half-core samples at 1.0 m lengths, 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. Remaining core, from mineralized intervals that are identified as being of potential metallurgical interest, is currently stored at temperatures that are maintained at below -5˚C in refrigerated containers at MKK's Juliaca core storage facility. Storage at -5˚C limits oxidation of the core and maintains the core in semi-pristine condition in preparation for additional studies if required.

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

11.1.1 Sample Preparation and Analysis

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.

Chemical analysis is conducted at the Certimin Lima laboratory and consists of fire assay (FA) with atomic absorption spectrometry (AAS) finish on the 50 g pulp aliquot. A 32-element suite was also analysed 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.

Smee (2009) completed an audit of the preparation laboratory and identified serious sample preparation issues.

- 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

The issues identified by Mr. Smee (2009), were found not to be corrected on a subsequent audit by Mr. Smee (2011). From January 2012 through to March 2012, all samples were sent directly to Certmin Lima, for sample preparation, until corrections at the preparation laboratory in Juliaca were made. These issues have since been rectified.

Coffey Mining has not visited the preparation laboratory to verify the issues have been rectified, but has received email confirmation from the laboratory manager detailing the date of the corrections and the work completed. Based on the information provided, Coffey Mining considers the data is adequate for use in the mineral resource estimation.

BSI Inspectorate laboratories, certified under ISO65 and certAll; ALS Chemex Lima, certified under ISO 9001:2008, ISO 17025:2005 and IQNet; and Actlabs, Chile, certified under ISO 9001:2008, ISO 17025:2005, were used as secondary laboratories during the assaying campaigns from 2008 to 2012.

11.1.2 Adequacy of Procedures

Coffey Mining has been advised that the main issues identified by Smee (2009, 2011), have been rectified. This includes:

- 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.

Coffey Mining has not been able to independently verify that the recommendations by Smee have been implemented at the Juliaca sample preparation laboratory and is relying on information provided by MKK and Certimin.

Coffey Mining considers that the sample preparation and security are adequate and appropriate for use in Mineral Resource estimation.

11.2 Metallurgical Sampling

11.2.1 Summary

For the Feasibility Study, a metallurgical testwork program was designed to confirm the optimal conditions obtained from previous testwork and to generate process design data. Two types of composites were required for the testwork program: *master composites* (one for leaching and one for comminution) to examine the metallurgical response which would be representative of the ore body and several *variability composites* to investigate the effect of variability on metallurgical performance.

Samples selected for these composites were gold-bearing intervals from drill holes located in the Minapampa Zone. The preparation of samples used in this testwork program was based on composite recipes which were developed from set criteria.

The master composites involved the combination of several intervals based on the following criteria: spatial distribution, the masses of the main three ore zones proportional to the mine plan, and a target head grade of 3.69 g/t Au.

Variability composites are intervals of sample, from various drill holes, along different strikes, at various depths, gold grades and from different ore zones. The main criteria for selection of variability samples included:

- the number of samples for each of the seven zones reflecting the percentage of each ore zone mined according to the PFS mine plan,
- samples spatially distributed across the ore body (location and depth),
- the number of samples from each location (easting section) matching the frequency distribution curve of gold and head grades ranging from 1 to 12 g/t Au.

For the master testwork program, 121 kg of samples were required for leaching testwork and 50 kg of sample was required for comminution testwork. Due to the availability of samples, two separate master composites were obtained for leaching (125 kg) and comminution (62 kg). Variability leaching testwork required 7 kg of sample for each test, whilst variability comminution testwork required 15 kg of sample for each test. Recipes for 37 variability composites were composed with enough samples for 31 leaching tests and 10 comminution tests.

Composites were created at Ammtec, during the period between January 2012 and July 2012.

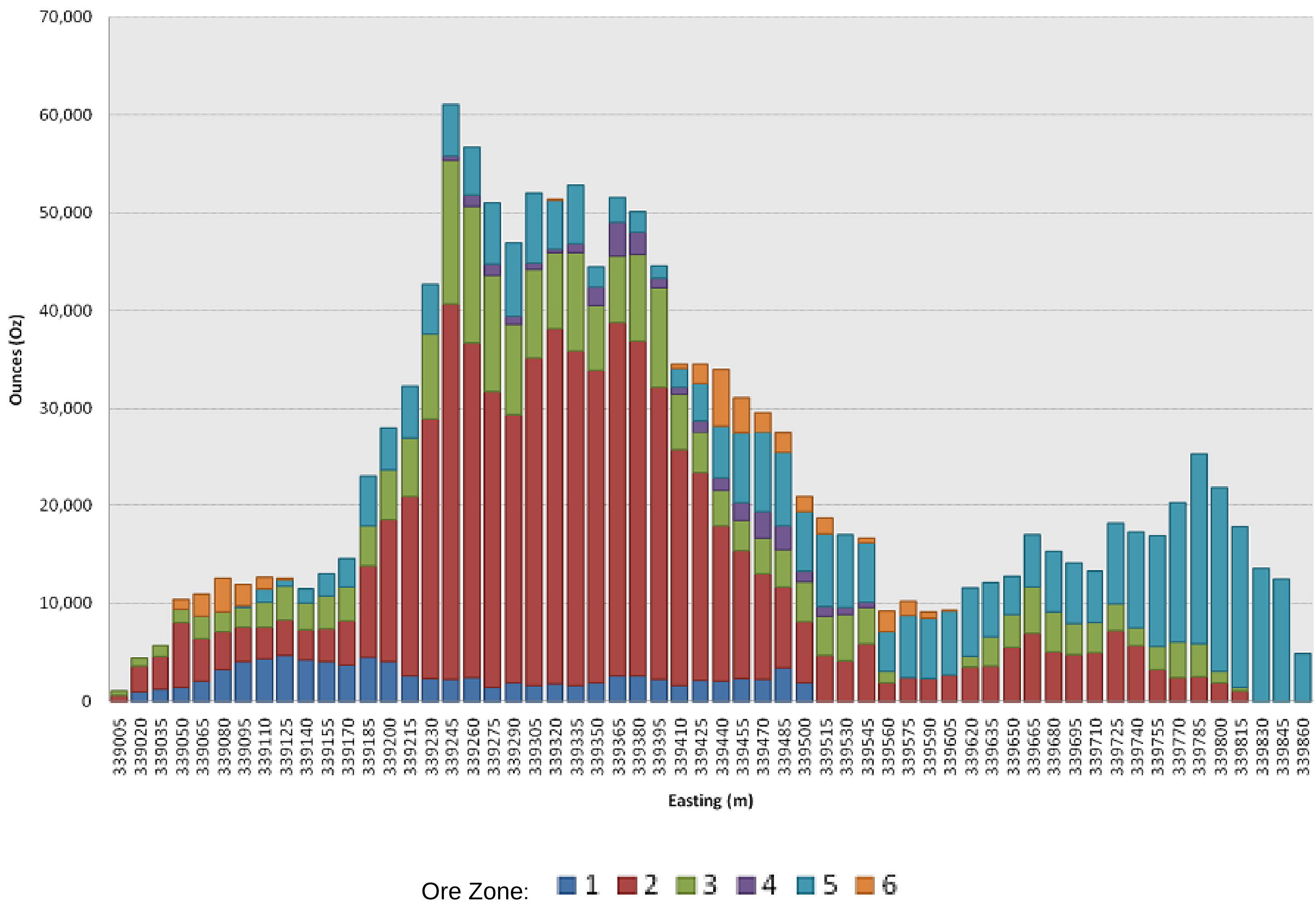
11.2.2 Background

Composite recipes were formed early in the study, when the Feasibility Study mine plan was not available. Therefore, composite recipes were developed based on the PFS mine plan.

According to the PFS mine plan, seven ore zones were identified for the Ollachea deposit, and the majority of the ore (~90%) would originate from Zone 2, Zone 3 and Zone 5.

The Ollachea drilling plan and exploration areas are described in Figure 10-3. Distribution of gold, by easting sections, is seen in Figure 11-1

Figure 11-1 Frequency distribution of Gold per 15m Easting Step (PFS)



11.2.3 MASTER SAMPLE SELECTION

Master composites were used in the FS master testwork program to investigate the metallurgical response representative of the ore body.

The master composite was prepared from drill core samples stored at Ammtec that had not been used during the PFS testwork program. The master composite involved the compositing of a number of continuous and non-continuous gold bearing intervals from the three main ore zones: Zone 2, Zone 3 and Zone 5.

To obtain a composite which would be representative of the whole ore body, the master composites involved the combination of samples based on the following criteria:

- The weight proportion corresponding to each of the three main ore zones was selected to match the proportion of each main ore zone mined according to the mine plan.
- A target overall bulk weighted average grade of 3.69 g/t Au to represent approximately the projected LOM average gold grade.
- Spatial distribution. Samples were selected from various drill holes which were spatially distributed through the ore body. Samples were also selected from intervals which were located from various locations along strike and from various depths (true depth).

Due to the availability of samples, an initial total master composite mass of 125 kg was obtained. This only met the sample requirements for the master leaching test work program (121 kg) and would not have been sufficient for the master comminution testwork program. Therefore, a separate master composite was required for comminution testwork.

The remaining samples of the variability composites available were used for the compositing of the comminution master composite in accordance with the above criteria. The sample size exceeded the sample amount of 50 kg required for comminution master tests. Samples available for master comminution were not full HQ, full PQ or half PQ as required for crushing work index (CWi) testing. For this reason, the CWi test results have been regarded as referential.

11.2.4 VARIABILITY SAMPLE SELECTION

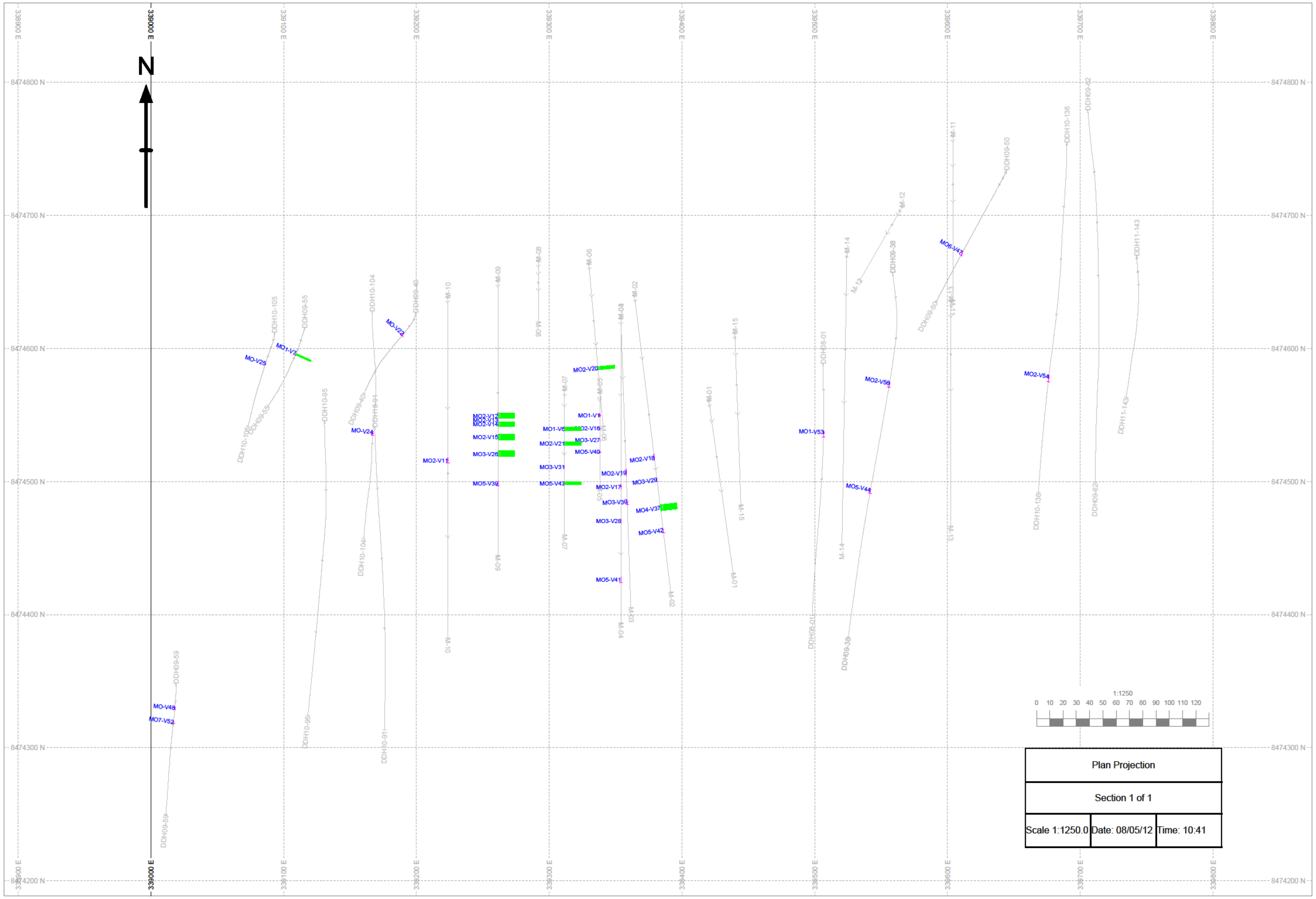
Variability composites were used in the FS metallurgical testwork program to investigate the effect of depth, ore grade, spatial distribution and ore zone on metallurgical performance.

Composites were prepared from intervals of in-fill drill cores and historical drill cores (stored in a freezer at MKK's core yard in Juliaca) located in the Minapampa Zone (any reference to Minapampa East in the PFS document is historical; all Minapampa East drill holes are now included within the overall, continuously mineralized, Minapampa Zone). Composites consisted of samples which were along the strike (down hole composites) and intersecting with an ore zone. Samples were selected based on the following criteria:

- The number of samples from each ore zone reflecting the percentage of each ore zone mined over the proposed 11 year mine life.
- Grade span: 1 g/t to > 12 g/t Au
- Samples spatially distributed across the ore body (location and depth).
- For leaching samples, the number of samples from each location (easting section) matching the frequency distribution curve of gold (Figure 11-1).

For the intervals selected for the recipe, 85% of each of the available samples was used for compositing and the remainder were saved as rejected intervals.

Figure 11-2 Spatial Distribution of Variability Samples



11.3 Density Determinations

The Ollachea database contains 777 bulk density determinations. A total of 103 of these determinations are within the mineralized zones (see Section 14.5).

All bulk density determinations were completed using the paraffin coated, water-immersion (Archimedean) technique on dried drill-core sample billets.

The sample billets used were approximately 10 cm long, half-core samples. The drill hole name and down hole distance were recorded for each sample, and were used to determine the spatial location.

The determinations were carried out by British Inspectorate Services Peru S.A.C.

11.4 Quality Assurance and Quality Control

QA/QC programs since the beginning of exploration work are listed in Table 10-1. The QA/QC measures employed by Peruvian Gold are unknown. No Peruvian Gold data have been used for the Minapampa Mineral Resource estimate or for any other aspect of the feasibility study.

All of the MKK samples in the Mineral Resource database have been submitted with standard reference materials to control assay accuracy and, depending on the program, have included field duplicate samples, coarse crush duplicates, pulp duplicates to control sampling, sub-sampling and analytical precision. Not all programs have included preparation duplicates. Pulp duplicates at the primary assay laboratory have not been included in the QA/QC program since the second half of 2009; however, pulp duplicates have been used for check assaying at a secondary laboratory for the entire QA/QC program.

A check assaying program has also been used to demonstrate the reproducibility of the assaying carried out in the primary laboratory, and to help establish assaying accuracy.

11.5 Databases

MKK has no formal database for the collation of data. A series of Excel spreadsheets are used to store the data. MKK has recently acquired the Maxwell GeoServices Datashed database management system for the storage of validated geological / analytical data.

In the meantime, Coffey Mining validated all of the data at Ollachea, against all original certificates (assay and survey), into a central database. This validated database will be merged into the MKK Datashed database when operational, by early 2013.

Coffey Mining considers the database to be globally robust and is appropriate for use in Mineral Resource estimation.

11.6 Sample Security

Coffey Mining has reviewed the entire sample chain of custody at Ollachea, from the drilling of the samples to the receiving of final analytical results, and is of the opinion that the in-house MKK Custody Control systems in place are of industry standard, and are adequate and appropriate for use in Mineral Resource and Reserve estimation.

12 DATA VERIFICATION

12.1 Independent Verification

In 2009 and 2011, Barry Smee conducted an independent audit of MKK sampling procedures and of the preparation and analytical laboratories used (Smee 2009, 2011). There were a number of issues that required addressing. These issues were addressed by MKK subsequently, and have been documented; refer Section 11.1.1.

During the site visit in 2010, Coffey Mining observed a hole being drilled (DDH10-110), and the recovered drill core material. On return to the Ollachea core storage / core cutting facility, Coffey Mining observed the logging, mark-up, sample cutting and bagging, and the sample dispatch and tracking procedure to the sample preparation laboratory at Juliaca. Coffey Mining did not visit the sample preparation laboratory, but did follow up on the recommendations by Smee (2009, 2011), which have since been rectified (confirmed by MKK and an email from the Juliaca laboratory manager). Whilst in Lima, Coffey Mining visited the analytical laboratory (Certimin – previously known as CIMM), and viewed their facilities and procedures.

It is the opinion of Coffey Mining that the sample preparation, sample security and analytical procedures associated with data generated to date are consistent with current industry practice and are considered entirely appropriate and acceptable for use in the mineral resource and reserve estimates.

The database currently held by MKK is a collection of Excel spreadsheets, which capture a whole variety of relevant data. As part of the mineral resource estimation, Coffey Mining has independently verified the entire database against the original assay and survey certificates for the entire project. The new database generated by Coffey Mining, has been compiled as new data is collected. MKK is in the process of implementing the Maxwell GeoServices Datashed database, which will use the database collated by Coffey Mining.

12.2 QA/QC

MKK's QA/QC program has varied during the history of exploration on the Property (Table 10-1), but generally has consisted of standards, blanks and pulp duplicates inserted with a frequency of approximately one in 20 (5%). Coffey Mining has assessed the QA/QC data and provides a summary in the following sub-sections.

12.2.1 MKK Standards and Blanks

MKK, using project specific mineralised rock material has had eight gold standards (8001 to 8009) of various grades prepared and assayed at independent laboratories. Coffey Mining (April 2010), identified issues with standards 8001 to 8004, which are no longer used.

Coffey Mining considers the current accuracy of the new standards 8006 to 8009 to be reasonable, but identified a number of poorly-monitored issues from the earlier standards. Results are summarized below:

- 8006 Over time shows a negative bias from the expected value (-2.14%). From 4 May 2010 to 5 October 2010 this bias is more pronounced, and could be attributed to a calibration error at the laboratory, as results return to expected values.

- 8007 Generally, the results are around the expected value, although there is a slight negative bias (-2.00%); this has been exaggerated by a possible misallocated standard submitted towards the end of January 2010.

- 8008 Similar to 8007, generally expected values are returned, a possible misallocated sample was included in November 2010 and early August 2011. Apart from these results, there is an almost negligible bias recorded from the expected value (-0.91%)

- 8009 A possible misallocated standard was included in late April 2011. Overall good accuracy with the expected value, with a very slight positive bias (+0.15%).

MKK used blanks to control sample contamination in the laboratory. Until August 2009, a pulverized blank (8005) was prepared from material obtained on the property that was defined as waste. However, analyses of blanks have shown some variability between detection limits (0.005 g/t Au) and approximately 0.02 g/t Au. A commercially-prepared blank was used after August 2009 and, in general, returned results below detection limits. From 21 February 2012, a new pulverised blank standard has been used (8017); to date; only 51 analyses have been returned and, in general, all results are below detection limits.MKK Duplicates

12.2.1.1. Field Duplicates

A field duplicate is collected after every 30 samples by MKK. Initially in the project, the field duplicates compared ½ core with ¼ core. Coffey Mining recommended that field duplicates be submitted based on a similar sample volume. That is, a ½ core sample (1 m interval) would have a ½ core field duplicate, a ¼ core sample (5 m interval) would have a ¼ core field duplicate.

Coffey Mining has compared the results of the ½ core versus ¼ core, ½ core versus ½ core and ¼ core versus ¼ core using the QC Assure software package. After examining the field duplicates, there does not appear to be much difference in the relative sample precision. For the ½ versus ¼ core samples (686 results) only 68% pass a 30% half absolute relative difference (HARD), whereas for the ½ versus ½ core samples (389 results) 70% pass a 30% HARD. The ¼ versus ¼ core samples (265 results) 71% pass a 30% HARD. In both cases, the precision levels are moderate, as is often encountered in nuggetty gold deposits.

The comparison of the ¼ core versus ½ core and the ½ core versus ½ core field duplicates, to date, shows there is a slight improvement when similar sample volumes are compared (½ core vs ½ core). There is a negative bias in the higher grade values (> 10 g/t Au), indicating the possible presence of coarse gold; in the higher grade samples.

The ¼ core versus ¼ core field duplicate is mainly restricted to the non-mineralized areas (5 m length). There are a few samples with grades greater than 1 g/t Au which are affecting the correlation.

Coffey Mining recommended that ½ core versus ¼ core duplicate be discontinued, in infill drill areas, as comparing different sample sizes does not produce conclusive results. During the last two drilling campaigns MKK has only collected ½ core field duplicates.

12.2.1.2. Preparation Duplicate Sample

After crushing the sample to a -2 mm size, the sample is split twice to 500 g with the second split representing the preparation duplicate. This occurred on samples up to and including DDH10-80 (last primary laboratory assay date – 18 January 2010).

Coffey Mining compared the preparation duplicate data (289 samples) using the QC Assure software. The results of these data show that the preparation duplicate has over 86% precision at 20% Rank HARD and 74% precision at 10% Rank HARD. This is a good result for this style of gold mineralization.

12.2.1.3. Pulp Duplicate

During the 2008 and 2009a drill programs, Certimin laboratories provided two pulps obtained from each sampled interval. MKK personnel recoded all the samples and regularly sent the second pulp of the same sample as pulp duplicate back to Certimin (i.e. a blind pulp duplicate). This occurred on samples up to and including DDH09-43 (with a last primary laboratory assay date of 17 June 2009). The 228 pulp duplicates submitted returned a poor precision of 58% at 10% Rank HARD with the mean grade of the duplicates being 8% higher than the mean grade of the original pulp samples (0.69 ppm Au versus 0.64 ppm Au).

The reasoning behind the poor precision levels seen in the pulp duplicates is unclear as the preparation laboratory duplicates returned an overall good precision. Smee (2009) suggested that the resubmitted pulps have been contaminated in some way, possibly due to humidity and or mixing of pulps. Poor homogenisation during pulverisation could also be an issue.

A total of 80 umpire pulp samples from the 2010 drilling campaign were sent to ALS Chemex laboratories in Santiago, Chile. The pulps were analysed using the same method as used by Certimin and showed high precision levels. The improved result from the umpire pulps indicates that oxidation of pulps may have an effect on the precision of the duplicate study. During the 2011 drilling campaign, a total of 86 umpire pulp samples were sent to ALS Chemex laboratories, Chile. The pulps showed a high precision level when compared to the original Certimin data, with a mean half relative difference of 0.75%, with over 66% of the data within a 10% precision level.

During the 2012 drilling campaign, a total of 513 umpire pulp samples were sent to Actlab, Chile. The pulps showed a high precision level when compared to the original Certimin data, with a mean half relative difference of 3.79%, with over 60% of the data within a 10% precision level.

12.3 Screen Fire Assay

In 2009, a screen fire assay (SFA) program consisting of 122 analyses was carried out, focusing on high-grade (> 10 g/t Au) samples. The results indicated that there is a nugget effect, and the nugget effect, as would be expected, is most pronounced in the high-grade samples.

As a follow up to the 2009 screen fire assay program, MKK submitted 221, one kilogram coarse reject samples from the 2009/2010 diamond drill program to conduct a screen fire assay evaluation at CIMM. The analysis compared the fine fraction (-150 mesh) with AAS and FA results, and the coarse fraction (+150 mesh) gravimetric with AAS finish and FA. The main findings were that there was no real nugget effect in the fine (-150 mesh) fraction but in the coarse fraction the nugget effect becomes an issue for values over about 6 g/t Au, where the FA shows a positive bias for the same AAS value.

Table 12-1 lists screen fire assay results for samples in six grade ranges. The quantity of samples in each of the grade ranges, the screen fire assay gold grade (SFA), the fine-fraction fire-assay gold grade (AAS (1)), and the original CIMM database fire assay gold grade (AAS (0)) are listed. Except for the 2 g/t Au to 5 g/t Au set, the SFA results are higher than the original assays. This is likely due to the larger support volume of the 1,000 g fire assays compared to the 500 g sub-samples for the standard assaying package. MKK note that the 2010 SFA campaign results indicated an average of 12.1% of gold in the +150 mesh fraction.

Table 12-1: Screen Fire Assay Results

Original Assay Au Grade (g/t)	Samples	Average Screen Fire Assay Au (g/t)	Fine Fraction Assay Au Grade AAS (0) (g/t)	Original Assay Au Grade AAS (1) (g/t)	Difference (AAS (1) -SFA)
> 10 g/t Au	3	21.8	13.71	18.32	81%
5 - 10 g/t Au	21	6.75	5.56	6.58	97%
2 - 5 g/t Au	57	3.15	2.73	3.2	100%
1 - 2 g/t Au	55	1.48	1.33	1.43	96%
0.5 - 1.0 g/t Au	42	0.81	0.75	0.74	91%
< 0.5 g/t Au	43	0.47	0.41	0.32	69%

12.4 Adequacy of Procedures

Procedures are in place to review assay results on a batch by batch basis. If any standards or blanks fail, the batch is immediately re-assayed, this is in line with industry standards. Analysis of all QA/QC results to date, have shown there to be no major discrepancies in the assay results received, and the results are considered to be adequate for use into the mineral resource estimate.

12.5 Comments on Section 12

Coffey Mining considers that the current drilling and sampling procedures undertaken by MKK are adequate for use in the mineral resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testwork

As part of the FS, metallurgical testwork was conducted on samples from the Ollachea deposit to investigate the ore's metallurgical response and to generate process design data. The FS testwork program was conducted between March and September 2012 at ALS Ammtec Limited (Ammtec).

13.1.1 Historic Testwork Program

The results of the historic testwork were reported in the previous NI 43-101 report submitted relating to the prefeasibility study. The results of previous testwork will be summarised below.

The first testwork program performed on Ollachea ore types was completed for the scoping study by Kappes Cassiday and Associates (KCA) between 2009 and early 2010. The testwork conducted at that stage focused on the initial flowsheet development:

- Scoping level tests for whole of ore direct leaching yielded low recoveries, which suggested preg-robbing or refractory components present in the ore.
- Gravity concentration, flotation and magnetic separation tests were conducted to investigate the effect of concentration prior to leaching. Gravity concentration and magnetic separation results indicated gold loss to tails to be significant and gravity/magnetic only flowsheets were not viable options.
- Flotation testwork was successful in achieving a concentrate; however, mass pulls to concentrate were high and significant gold losses to tails were prevalent.
- Leaching in the presence of activated carbon provided acceptable extractions, suggesting the CIL process to be a possible flowsheet.

Subsequent testwork programs were completed for the prefeasibility by Ammtec between 2010 and mid-2011. PFS testwork was conducted with the aim of further developing the flowsheet and investigating optimal conditions:

- Flotation testwork, which was conducted to investigate carbon pre-flotation and carbon suppression, was unsuccessful in the preferential separation of carbon species.
- Leaching extractions of flotation concentrates resulted in extractions which were lower than whole of ore leaching (CIL).
- Using whole of ore leaching, tests were conducted to investigate alternative methods of mitigating organic carbon influences including resins, use of kerosene "blanking", use of NaOH in place of hydrated lime for pH control in direct leaching, use of pre-oxygenation prior to direct leaching and use of low pulp density for direct leaching.

The results of this series of tests suggested use of pre-aeration and kerosene blanking may provide advantages combined with CIL leaching.

- Based on these results, the CIL flowsheet was taken forward as the basis of the Prefeasibility study and further testwork.
- In agreement with KCA testwork, initial variability testwork conducted at high carbon populations achieved acceptable extractions but at high cyanide consumption.
- Optimisation of the CIL flowsheet indicated leaching in the presence of a carbon population of 5 g/L, with the addition of 0.1 kg/t of kerosene, with a slurry pulp density of 40% w/w solids, using lime for pH adjustment, 0.05% w/v initial cyanide (maintained at >0.02% w/v) and ambient temperature to be acceptable.

Interpretation of metallurgical testwork results in the PFS suggested that crushing and grinding of ore to P_{80} of 75 µm with gravity concentration and carbon-in-leach (CIL) treatment of the whole of the ore stream (using PFS optimised conditions) could be used to achieve gold recovery of over 90% from the Ollachea mineralization.

Additional variability testwork was conducted during the PFS, but was not available for inclusion in the NI43-101 report. This series of tests was performed under conditions derived from the PFS optimisation testwork. The results of this series of tests indicated low and moderately variable overall gold extractions, which averaged 80%. Poor leaching kinetics were also observed for the composites tested, with gold dissolution still occurring after 48 h of leaching time.

With the aim of improving project economics, the FS testwork program investigated an alternate flowsheet. The alternate flowsheet addressed the high cyanide consumption, low recoveries and slow leaching kinetics of the standard CIL circuit in previous testwork and, subsequently, improvements to the project economics was forecasted.

13.1.2 Master and Variability Composites

Master composites were used in the FS testwork program to investigate the metallurgical response representative of the ore body. The master composites involved the combination of several drillcore intervals based on the following criteria: spatial distribution, the masses of the main three ore zones proportional to the mine plan, and a target head grade of 3.69 g/t Au. The assay head grade obtained for the master composite was 3.3 g/t.

Variability composites were used in the FS metallurgical testwork program to investigate the effect of depth, ore grade, spatial distribution and ore zone on metallurgical performance. Variability composites are intervals of samples from various drill holes, along different strikes, at various depths, gold grades and from different ore zones.

The composites formed, samples selected and methodology used is discussed in Section 11.2.

13.1.3 Master Composite Mineralogy

The mineralogy analysis on the master composite by Roger Townend and Associates (performed in 2012) has confirmed pyrrhotite to be the predominant sulphide mineral with graphitic gangue. The analysis showed occurrences of gold to be associated with arsenopyrite, pyrrhotite, pyrite and arsenopyrite/pyrrhotite. As a consequence, some gold losses due to sulphide locking would be expected. Both fine grained (-10 µm) and coarse liberated gold occurrences were observed. These gold occurrences would be expected to be leached with cyanide.

13.1.4 Optimisation Testwork

Optimisation testwork was conducted on the master composite to confirm the CIL conditions which were selected in the PFS. Base conditions for this series of tests were established by analysing previous testwork information, including the results from the scoping study and PFS testwork. The comparison of the results from the optimisation series to the test at base conditions was used to investigate the effect of changing individual conditions on the extraction kinetics, the final residue (extraction) and reagent consumption. Conditions which were explored included grind size, cyanide concentration, kerosene conditioning dosage, slurry pH, carbon population and temperature. The results of the optimisation series are summarised in Table 13-2. Optimal conditions which were derived from this testwork are presented in Table 13-1.

Table 13-1 FS Optimisation Testwork Optimal Conditions

Parameter	Units	Value
Grind Size P_{80}	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

With the exception of the grind size of P_{80} of 106 µm and the cyanide concentration of 0.1% w/w being maintained for 36 h, the results confirmed the leaching conditions used in PFS as optimal.

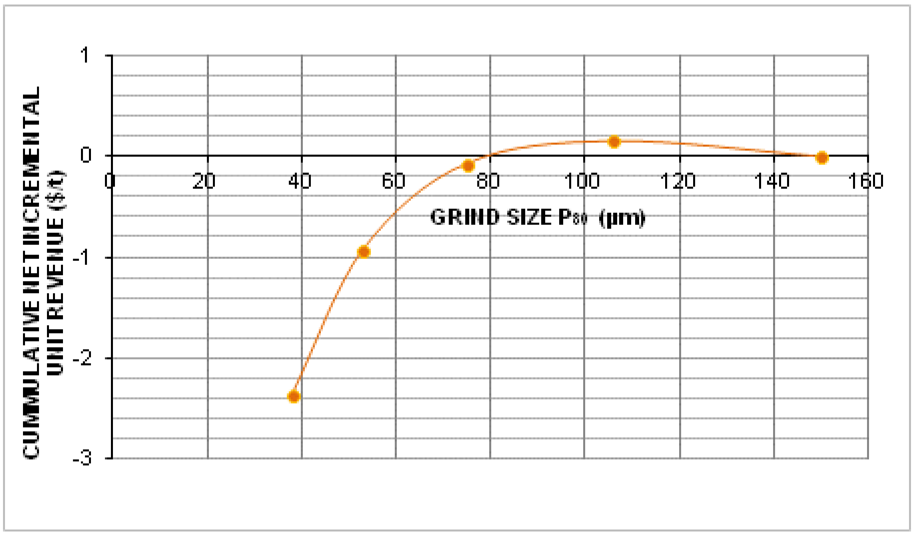
For the different grind sizes, results showed similar extraction kinetics with no obvious kinetic improvement with a finer grind. The results also indicated a marginal decrease in final residue grade, and, hence, small improvements to overall extraction at a finer grind size. However, elevated sodium cyanide consumption was observed for tests conducted at a finer grind size.

The selection of the optimal grind size took into consideration not only the gold extraction but also the associated operational and capital cost impacts. Incremental revenue gain against the associated cost of reagents in leaching, power consumption in grinding, and capital due to mill sizing for the different grind sizes was analysed. As an outcome of the analysis (Figure 13-1), a coarse grind P_{80} of 106 µm was chosen as the optimal grind size for FS testwork, as well as for design. Improvements to downstream processes (such as thickening, filtration and paste backfill cement addition) as a result of the coarse grind size were not quantified in the analysis.

Table 13-2 FS Optimisation Testwork Optimal Conditions

Test Description	Test Number	Calc Head Grade g/t Au	Residue Grade g/t Au	Gravity Ext % Au	Extraction at 48h % Au	Extraction at 72h % Au	NaCN consumption kg/t	CaO consumption kg/t	Carbon Population g/L	Temperature °C
Base Conditions, 75 µm	JS1277	2.43	0.37	40.4	83.6	84.8	1.6	0.8	10	Ambient
125 µm	JS1275	2.46	0.41	40.9	83.3	83.3	1.5	0.6	10	Ambient
106 µm	JS1276	2.37	0.39	45.4	82.2	83.5	1.5	1.0	10	Ambient
0.2% w/v NaCN	JS1278	2.43	0.39	44.2	81.8	84.0	2.9	0.9	10	Ambient
0.05% w/v NaCN	JS1279	2.33	0.47	45.4	78.7	79.9	1.1	1.0	10	Ambient
pH 11.5	JS1280	2.75	0.37	56.8	85.6	86.5	1.3	1.9	10	Ambient
pH 12.5	JS1281	3.20	0.39	51.1	87.0	87.8	1.1	9.3	10	Ambient
0.1 kg/t Kero	JS1282	2.42	0.35	42.3	82.1	85.6	1.5	2.5	10	Ambient
0.5 kg/t Kero	JS1283	2.04	0.35	29.7	82.8	82.8	1.5	0.7	10	Ambient
1 kg/t Kero	JS1284	3.29	0.33	37.0	90.0	90.0	1.5	0.7	10	Ambient
5 g/L Carbon	JS1285	2.66	0.40	53.0	83.0	85.0	1.5	0.9	5	Ambient
10 g/L Carbon	JS1286	3.19	0.27	44.9	90.4	91.5	2.0	1.0	10	Ambient
25 g/L Carbon	JS1287	2.28	0.20	48.3	87.3	91.2	3.6	1.3	25	Ambient
50 g/L Carbon	JS1288	2.91	0.22	28.5	86.4	92.4	4.7	1.5	50	Ambient
Temperature and Carbon Concentration Effects	JS1289	3.01	0.27	38.8	90.6	91.0	5.3	3.2	15	70
	JS1290	2.59	0.28	47.5	89.0	89.2	5.5	3.1	25	70
	JS1291	2.87	0.34	51.3	87.8	88.2	5.8	3.7	50	70
	JS1292	2.47	0.24	39.1	88.8	90.3	6.4	5.4	15	90
	JS1293	2.27	0.14	47.3	93.8	93.8	6.3	5.2	25	90
	JS1294	2.35	0.14	38.9	94.0	94.0	7.5	6.4	50	90

Figure 13-1 Grind Size Optimisation Analysis[2]



The benefits of these other conditions (carbon population, cyanide concentration and temperature) provide opportunities to increase the leach extraction depending on the prevailing costs and economic conditions.

Although the sodium cyanide consumption increased by approximately 0.5 kg/t, improvement to final residue gold grade by 0.1 g/t was demonstrated for the increase in cyanide concentration from 0.05% w/v to 0.10% w/v. The financial benefit of further gold extraction was found to outweigh the additional cost of cyanide. No further extraction was seen with the increase in concentration to 0.20% w/v. Consequently, a cyanide concentration of 0.10% w/v was selected as the optimal concentration.

Testwork results suggested extraction to be highly influenced by temperature and carbon population. Residue grades of below 0.2 g/t Au were achievable at elevated temperatures or elevated carbon populations. However, cyanide consumption to achieve these residue grades was very high, ranging from 3.6 kg/t to 7.5 kg/t. Based on the increase in cyanide consumption alone, elevated temperatures and high carbon populations were not considered viable.

[2] Analysis using 150 µm as Base Case, US$1250/oz gold price and US$3.6/kg sodium cyanide cost.

13.1.5 Initial Evaluation of Alternate Flowsheet

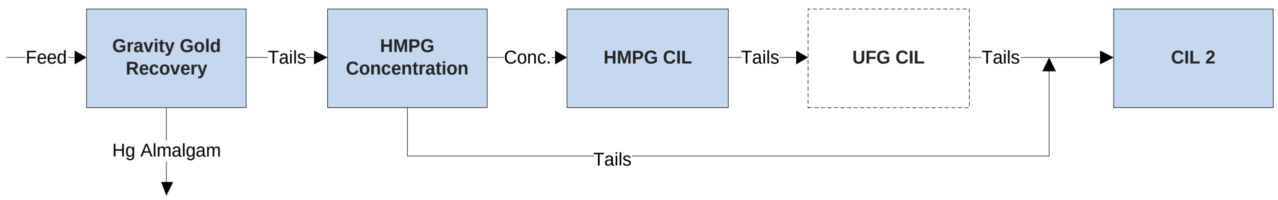
Several CIL tests at optimal conditions were performed on the master composite to provide material for subsequent stages of ancillary testwork. The results of these tests yielded lower than expected gold extractions (86.3% average) combined with high cyanide consumption.

With the aim of improving the extraction and cyanide consumption, an alternative to the standard CIL flowsheet was proposed. The main feature of the proposed flowsheet is the high mass pull gravity (HMPG) concentration step prior to CIL. The purpose of this concentration step is to separate higher grade gold bearing sulphides from preg-robbing carbonaceous material. Concentrate from the gravity step will then be treated in a separate leaching step (HMPG CIL) prior to recombining with gravity tails and undergoing the final CIL stage.

To investigate the alternate flowsheet, four (4) variability composites together with the master composite were subjected to initial alternate flowsheet evaluation tests. For the purposes of comparison, the same variability composites were also subjected to standard CIL tests.

Mineralogical analysis completed on the master composite sample identified -10 µm gold to be locked up in arsenopyrite. Consequently, the effect of ultrafine grinding of the HMPG concentrate was investigated. In the initial evaluation tests, ultrafine grinding (UFG) to -10 µm was performed on the product of HMPG CIL prior to recombining with gravity tails for the final CIL step (CIL 2). The flowsheet used for the initial evaluation tests is illustrated by Figure 13-2 .

Figure 13-2 Alternate Flowsheet – Initial Evaluation Testwork



The results of the initial evaluation tests are summarised in Table 13-3. The alternate flowsheet testwork results demonstrated decrease in residue grades but increased cyanide consumption compared to standard CIL results. An analysis of these two factors indicated revenue increase was achievable for all composites (with the exception of Variability Composite #21) with gains ranging from US$2/t to US$9/t for the alternate flowsheet. From this analysis, the leaching variability testwork proceeded using the alternate flowsheet.

Ultra-fine grinding of the HMPG tails contributed an additional extraction of some 2% of the new feed gold content. At this extraction, ultra-fine grinding is deemed uneconomic and was not included in the alternate flowsheet variability testwork or in the design.

Table 13-3 Initial Evaluation Tests – Standard CIL and Alternate Flowsheet Results Summary

	Parameter	Units	Master Composite	Variability Composite #12	Variability Composite #16	Variability Composite #21	Variability Composite #33
Standard CIL	Sample ID	-		MO2-V18	MO2-V22	MO3-28	MO6-V48
	Head Assay	g/t Au	3.29 / 3.31	7.68 / 7.44	1.58 / 1.39	0.77 / 1.12	1.24 / 0.81
	Calc Head Assay	g/t Au	2.43[3]	5.64	2.27	1.55	0.94
	Residue Grade[4]	g/t Au	0.37[3]	0.50	0.27	0.29	0.51
	Overall Extraction[4]	% Au	85.1[3]	91.2	88.1	81.3	45.7
	Cyanide Consumption[4]	kg/t	2.0[3]	1.8	1.1	1.4	1.2
Alternate Flowsheet	Calc Head Assay	g/t Au	2.59	7.66	2.00	1.95	1.06
	Residue Grade[4]	g/t Au	0.19	0.35	0.19	0.30	0.25
	Overall Extraction[4]	% Au	92.7	95.4	90.5	84.4	76.4
	Cyanide Consumption[4]	kg/t	3.2	3.0	1.5	4.3	1.6
Financial Analysis	Increase in gold	US$/t	7.23	5.70	3.22	-0.58	10.45
	Difference in cyanide	US$/t	-3.86	-3.77	-1.34	-9.98	-1.43
	Incremental increase in Revenue	US$/t	2.62	1.94	1.87	-10.56	9.02

[3] Average of 7 tests using master composite and optimal conditions

[4] Final residue grade, gold extraction and cyanide consumption at 36 h for alternate flowsheet and 72 h for standard

13.1.6 Alternate Flowsheet Variability Testwork

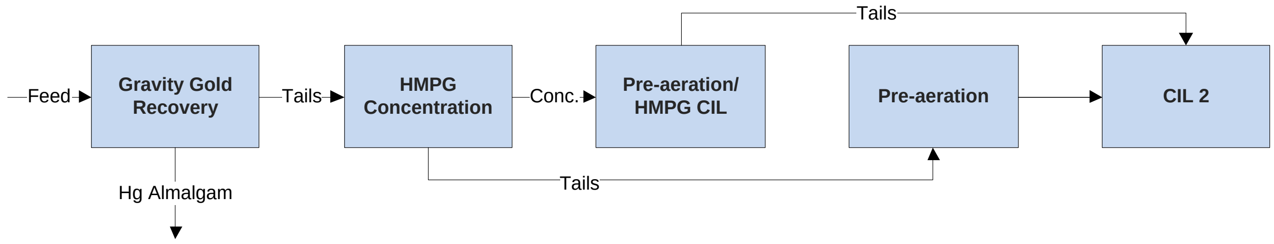
13.1.6.1. Pre-aeration

During initial detoxification testwork, high soluble iron levels in the CIL circuit were found to be the main contributor to high total cyanide (CN_{Total}) concentrations experienced in the detoxification product. In the remainder of the leaching testwork, pre-aeration was incorporated as an additional step prior to the CIL to gauge the effectiveness of lowering these iron concentrations and reducing both the total cyanide concentration of the CIL discharge and subsequent cyanide detoxification process demands.

13.1.6.2. Variability Testwork Flowsheet

The remainder of the variability leaching program on the twenty seven (27) variability composites progressed using the alternate flowsheet (without ultrafine grinding) as illustrated in Figure 13-3.

Figure 13-3 Alternate Flowsheet – Variability Testwork



13.1.6.3. Variability Testwork Results

Alternate flowsheet variability testwork results are summarised in Table 13-4. In agreement with initial evaluation tests, low residue grades and, consequently, high gold extractions of up 90% and above can be achieved for many of the samples. However, variability was evident for the samples tested, with residue grades and cyanide consumption varying between samples of different head grades and from different ore zones. Gold extractions as low as 63% were also obtained (Composite #34 - with a low gold head grade of 0.9 g/t and originating from Ore Zone 7). The distribution of overall gold extractions as a function of average head assays and calculated head grades is illustrated by Figure 13-4 and Figure 13-5 respectively.

Table 13-4 Variability Leaching Testwork Results Summary

Comp ID	Comp ID	Geo Log Grade	Assay Head	Calc. Head	Residue[5]	Overall Extraction Calc Head[5]	Overall Extraction Average Assay Head[5]	Cyanide Cons.[5]
Ammtec	Amec	g/t Au	g/t Au	g/t Au	g/t Au	%	%	kg/t
VC #01	MO2-V55	1.6	0.84 / 0.54	0.4	0.10	77.5	85.5	1.4
VC #05	MO2-V11	1.3	0.55 / 0.89	1.0	0.19	80.5	73.6	1.3
VC #07	MO2-V13	1.5	1.94 / 1.52	1.9	0.16	91.7	90.8	1.3
VC #08	MO2-V14	4.3	3.59 / 4.27	2.2	0.25	88.7	93.6	1.2
VC #10	MO2-V16	4.7	4.07 / 4.58	4.5	0.38	91.6	91.2	1.2
VC #11	MO2-V17	5.0	3.42 / 2.37	2.0	0.13	93.5	95.5	1.7
VC #13	MO2-V19	4.5	6.77 / 6.86	4.6	0.33	92.9	95.2	1.0
VC #14	MO2-V20	1.2	11.5 / 11.8	8.2	0.39	95.2	96.7	1.8
VC #15	MO2-V21	4.2	5.15 / 5.36	3.9	0.17	95.6	96.8	1.7
VC #17	MO2-V24	1.7	1.79 / 1.7	1.4	0.08	94.2	95.4	1.2
VC #18	MO2-V25	2.7	0.77 / 1.12	1.3	0.18	85.8	81.0	0.8
VC #37	MO2-V54	3.6	2.92 / 2.93	3.3	0.31	90.6	90.6	1.5
Ore Zone 2 Average				2.9	0.21	89.8	90.5	1.2
VC #19	MO3-V26	12.2	3.13 / 3.31	3.6	0.53	85.0	83.4	1.2
VC #20	MO3-V27	4.7	4.78 / 4.23	2.6	0.40	84.5	91.2	1.7
VC #22	MO3-V29	2.6	3.46 / 5.41	4.4	0.22	95.0	95.0	1.1
VC #23	MO3-V30	2.4	2.49 / 2.67	2.7	0.36	86.7	86.0	1.0
VC #24	MO3-V31	2.1	1.48 / 1.65	1.5	0.22	85.4	85.9	0.8
Ore Zone 3 Average				2.9	0.35	86.9	88.3	1.2
VC #26	MO5-V39	2.3	2.26 / 2.44	2.3	0.37	83.9	84.1	1.4
VC #27	MO5-V40	4.6	14.2 / 11.6	10.7	0.59	94.5	95.4	1.5
VC #28	MO5-V41	3.6	3.72 / 4.15	4.7	0.74	84.1	81.1	1.7
VC #29	MO5-V42	1.9	1.39 / 1.44	1.4	0.43	68.7	69.9	0.8
VC #30	MO5-V43	2.2	2.8 / 4.24	2.4	0.62	74.0	82.2	1.1
VC #31	MO5-V44	2.8	7.71 / 6.72	6.8	0.44	93.5	93.9	1.1
Ore Zone 5 Average				4.7	0.53	83.1	84.4	1.3
VC #04	MO1-V7	1.6	1.4 / 1.47	1.6	0.16	90.2	88.9	1.3
VC #25	MO4-V37	2.4	3.06 / 2.11	1.9	0.26	86.1	89.9	1.8
VC #32	MO6-V47	1.2	0.99 / 1.25	1.2	0.36	69.2	68.0	1.2
VC #34	MO7-V52	1.6	0.63 / 1.4	0.9	0.34	63.2	66.5	1.0
Overall Minimum				0.4	0.08	63.2	66.5	0.8
Overall Maximum				9.8	0.74	95.6	96.8	1.8
Overall Average				2.9	0.32	85.4	86.9	1.3
Overall Standard Deviation				2.2	0.17	8.8	8.9	0.3
Overall 80th Percentile				4.2	0.42	93.5	95.1	1.6

[5] Residues, extractions and cyanide consumptions are given for 36 h leach time.

Figure 13-4 Alternate Flowsheet Variability Testwork – Overall Extraction (Average Head Assay)[6]

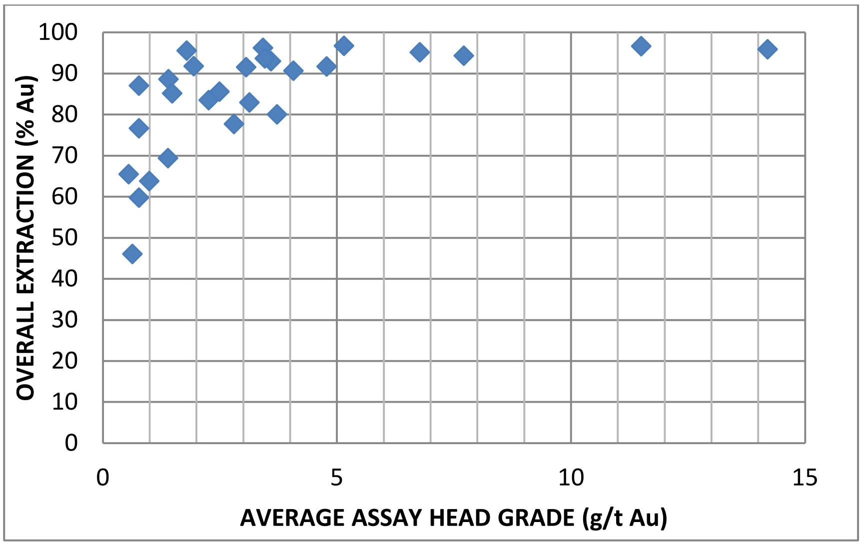
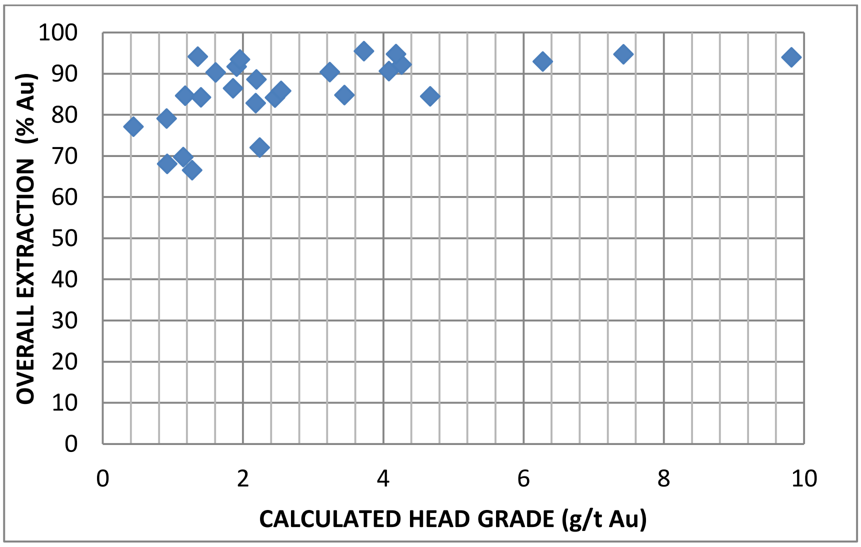


Figure 13-5 Alternate Flowsheet Variability Testwork – Overall Extraction (Calculated Head Grade)[7]



[6] Extractions based on residue gold grades and for 36 h leach time and average head assay.

[7] Extractions based on residue gold grades at 36 h leach time and calculated head grades.

13.1.6.4. Variability Testwork Extractions

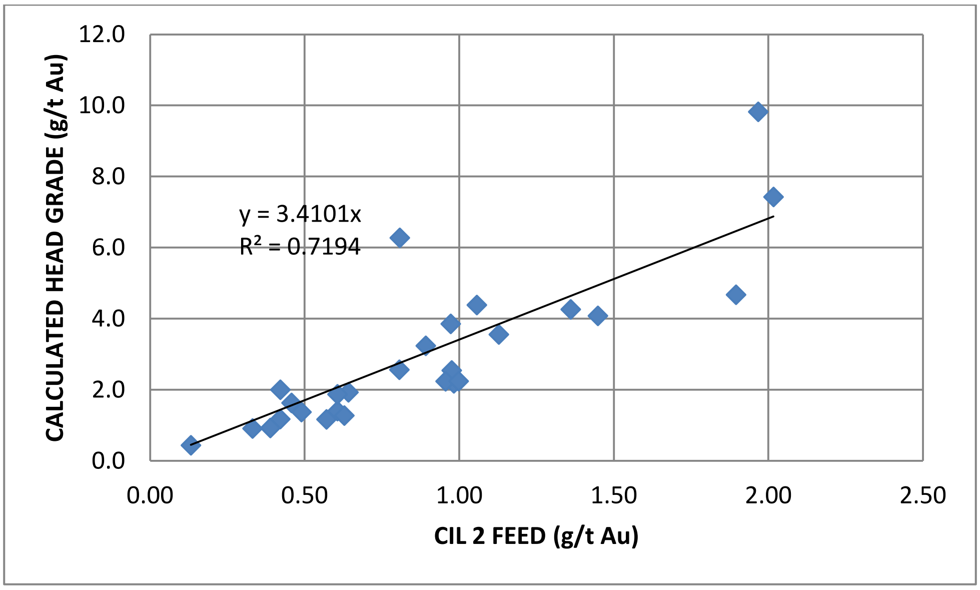
The variability testwork yielded consistent gravity gold recovery to mercury amalgam, sulphide associated gold to HMPG concentrate and consistent gold dissolution within the HMPG CIL circuit (summarised in Table 13-5). CIL 2 exhibited less consistent gold dissolution results. The consistent recoveries and dissolution prior to CIL 2 lead to a feed to CIL 2 containing approximately 29% of the total gold as seen in

Figure **13-6**.

Table 13-5 Alternate Flowsheet Distribution of Gold

Parameter	Units	Value
Gravity Recoverable Gold and HMPG circuit recovery	% Total Gold	71
Feed to CIL 2	**% Total Gold**	**29**

Figure 13-6 Alternate Variability Tests – CIL 2 Feed (Calculated Head Grade



Results from the CIL 2 tests show leaching performance to be dependent on the ore zone from which the sample originated. Ore Zone 1 and 2 showed better overall extractions; whilst moderate overall extractions for Ore Zone 3 and 4 were observed. Ore Zones 5, 6 and 7 gave poor extractions relative to other ore zones.

The correlation between final residue grade and CIL 2 feed grade for Ore Zone 2, 3 and 5 is shown in Figure 13-7, Figure 13-8 and Figure 13-9 respectively.

Figure 13-7 Alternate Variability Tests – Ore Zone 2

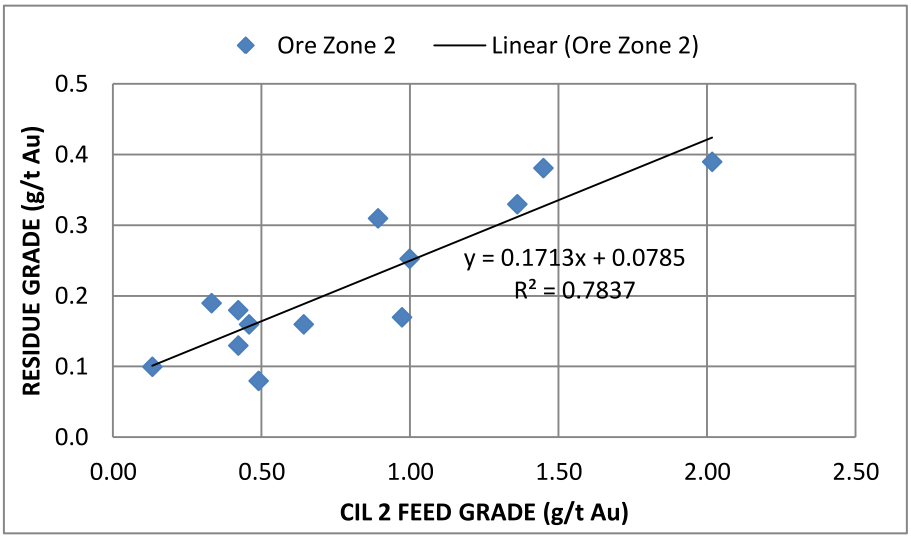


Figure 13-8 Alternate Variability Tests – Ore Zone 3

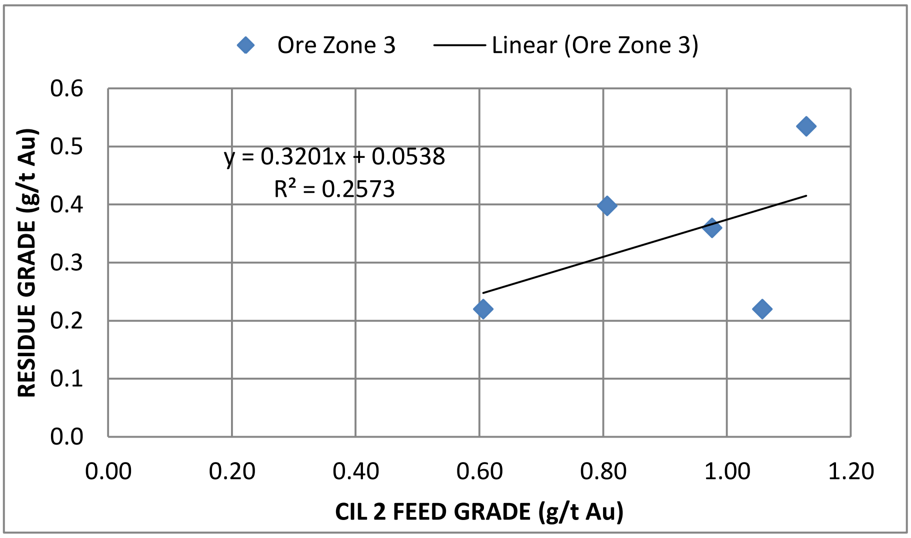
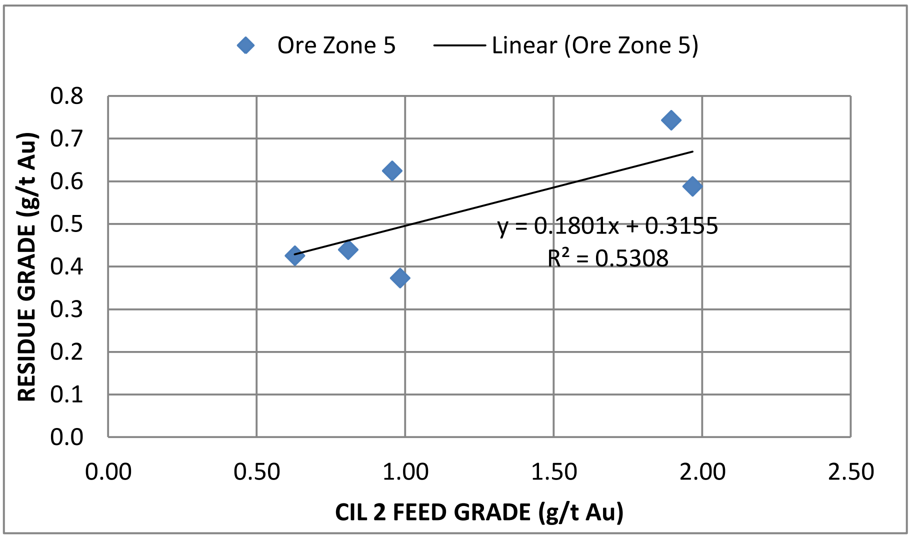


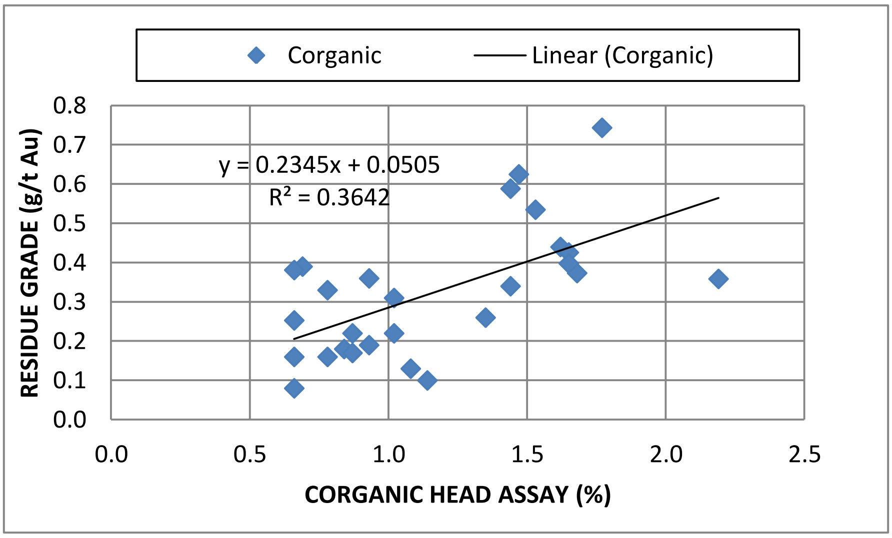
Figure 13-9 Alternate Variability Tests – Ore Zone 5



Analysis of the relationship between the final residue grade and organic carbon content (Figure 13-10) indicated that final residue grade is influenced by the organic carbon content of the sample, with final residue grades increasing (linearly) with organic carbon head assays. Samples from Ore Zone 5 tested in the variability testwork displayed high organic carbon assays ranging from 1.44% to 1.77%. The high organic carbon assays of these samples are likely the contributing factor to the poor extractions obtained, due to pre-robbing of gold in solution.

In previous testwork (mainly PFS variability testwork), samples from Ore Zone 5 did not display the degree of organic carbon concentrations seen with the FS variability samples. Initial examination of organic carbon assays ($C_{organic}$) of several in-fill drill cores were in agreement with the FS samples with $C_{organic}$ assays averaging 1.6%.

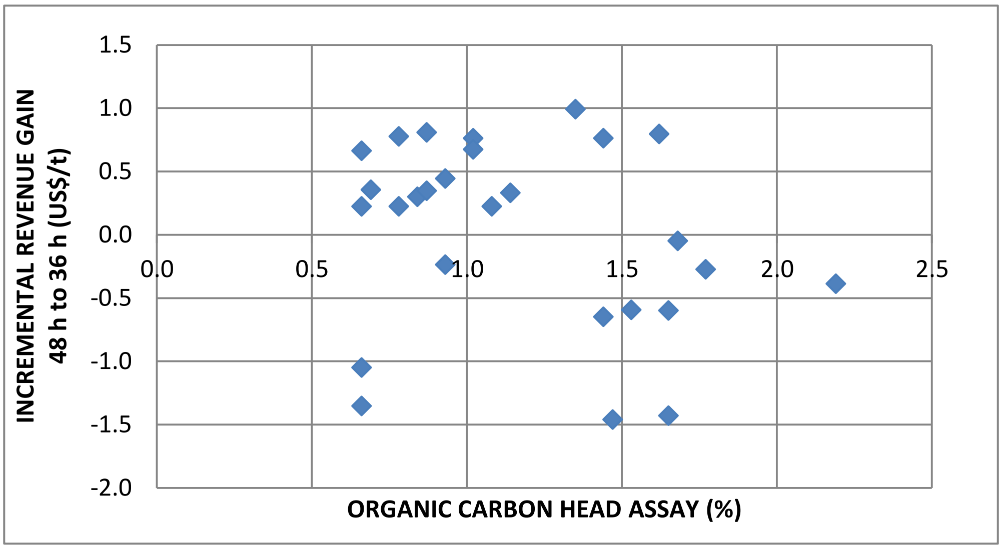
Figure 13-10 Alternate Variability Tests – Organic Carbon Concentration



13.1.6.5. Leaching Kinetics

Similar to the results of the initial alternate flowsheet evaluation, variability testwork displayed improved leaching kinetics for the final stage of CIL (CIL 2), which forms the basis for the main CIL circuit in the design. The comparison of the incremental revenue gain based on residue gold grade (gold extraction) and cyanide consumption, with the decrease in leach time from 48 h to 36 h is presented in Figure 13-11. For the majority of samples, the data indicated a positive revenue gain was achievable with the decrease in residence time.

Figure 13-11 Alternate Variability Tests – Leaching Time from 48 h to 36 h[8]



The same analysis was performed for the decrease of residence time from 36 h to 24 h; however, results indicated that revenue losses would be incurred and suggested that the 36 hour residence time should be retained as a basis of design.

This analysis did not take into consideration the operating and capital costs associated with decreased residence time and consequently decrease in tank volumes. Further evaluation is required at the detailed design stage.

Cyanide Consumption

Total consumption of cyanide for the alternate flowsheet (36 h leach in CIL 2) in the variability testwork averaged approximately 1.3 kg/t. These results indicated lower cyanide consumption than was observed in the initial alternate flowsheet evaluations. The decrease in cyanide consumption is attributed to the inclusion of pre-aeration.

[8] Using US$1250/oz gold price and US$3.6/kg sodium cyanide cost

13.1.7 Further Evaluation Tests of Alternate Flowsheet

The remainder of the alternate flowsheet evaluation tests were run in parallel with the leaching variability testwork. An additional twelve (12) variability composites were also subjected the standard CIL tests (including pre-aeration) to provide comparative data regarding the benefits of the alternate flowsheet.

Similar to the initial evaluation tests, the incremental revenue gain of alternate flowsheet compared to the standard CIL flowsheet, was analysed with respect to the additional gold extracted (residue grade) and the associated cyanide consumption increase. This analysis is summarised in Table 13-6.

The analysis[9] indicated a benefit in net revenue for all twelve composites, ranging from US$3/t to US$50/t. The reduced cyanide consumption identified by the variability testing increases the benefit attributed to the alternate flowsheet compared to the initial evaluation tests.

Table 13-6 Further Evaluation Test – Alternate Flowsheet Financial Comparison

Parameter	Units	Average	Average[10]	Variability Composite #19[11]
Difference in Gold Extracted	US$/t	18.37	14.23	13.07
Difference in NaCN Consumption	US$/t	-1.61	-1.90	-0.82
Increase in Revenue	US$/t	16.89	12.33	12.25

13.1.7.1. Head Grades

Head grades for the variability samples were developed from the following sources:

- Head assay grades – two or more assays for each sample
- Calculated head grades – based on summation of assays of residue, carbon and solution of each test for the 31 variability composites. For the 12 composites which were subjected to the three different tests (alternate tests and standard CIL), three head grades can be calculated.
- Geological log head grades – calculated based on the weighted assays of individual intervals used in the composite.

Analysis of these various head grades for the samples show discrepancies. With head grades and corresponding final residue grades forming the basis for extraction, conflicting head grades consequently leads to discrepancies in calculated extractions.

[9] Using US$1250/oz gold price and US$3.6/kg sodium cyanide cost

[10] Outliers Removed

[11] Composite #19 has a gold head grade of 3.5 g/t, similar to the LOM head grade.

Calculated head grades for all samples subjected to the alternate flowsheet (in the variability testwork) were approximately 27% lower than the average head assay, as indicated in Figure 13-12. The calculated head grade for the twelve samples subjected to standard CIL tests is compared to the average head assay in Figure 13-13. The comparison shows calculated head grades using standard CIL test results to be similar (approximately 97%) to the average head assay. Comparisons of the average assay head grade and the calculated head grade with geological log head grade are presented in Figure 13-14 and Figure 13-15, respectively. These comparisons suggest that the average assay grade is closer to that of the grade obtained from the geological log.

The calculation for head grades for the alternate flowsheet is based on the summation of more residue, carbon and solution assays than the calculation for head grades for the standard flowsheet. Consequently, the resulting cumulative sum of errors is greater.

In the calculation for head grade, the proportion of gold considered "extracted" is the most significant proportion and also the proportion with the least accurate method of determination. Gold extraction is mainly based on gold loading on activated carbon, therefore, the detection limit of gold on activated carbon and the difficulty in repeatable assaying due to the issues associated with representative sampling of carbon can have a considerable impact on calculated head accuracy. Often the loaded carbon gold assay is of the same magnitude as the detection limit or accuracy of the carbon gold assay procedure. At concentrations below the detection limit, results are reported as half of the lowest value detectable. It appears that, for the samples under consideration, this may have led to a systematic under-reporting of gold loading on activated carbon. Similarly, carbon assays reported for carbon which is loaded to high concentrations of gold, may also be under-reported, due to the detection limit. Consequently, the average assay head grade was used for the purpose of analysing testwork recoveries and projecting design recoveries.

Figure 13-12 Calculated and Assay Head Grades – Alternate Flowsheet

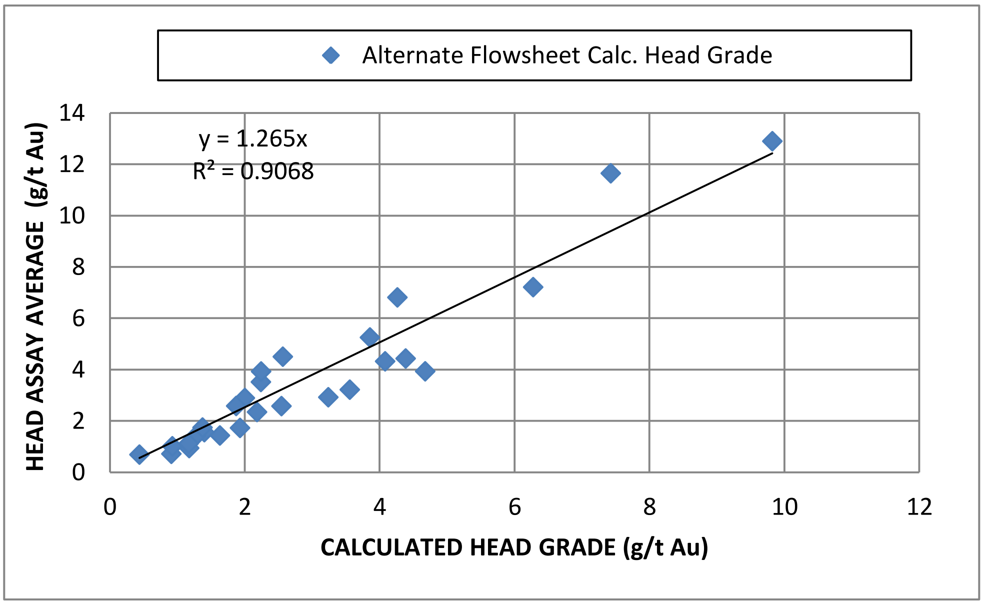


Figure 13-13 Calculated and Assay Head Grades – Standard CIL

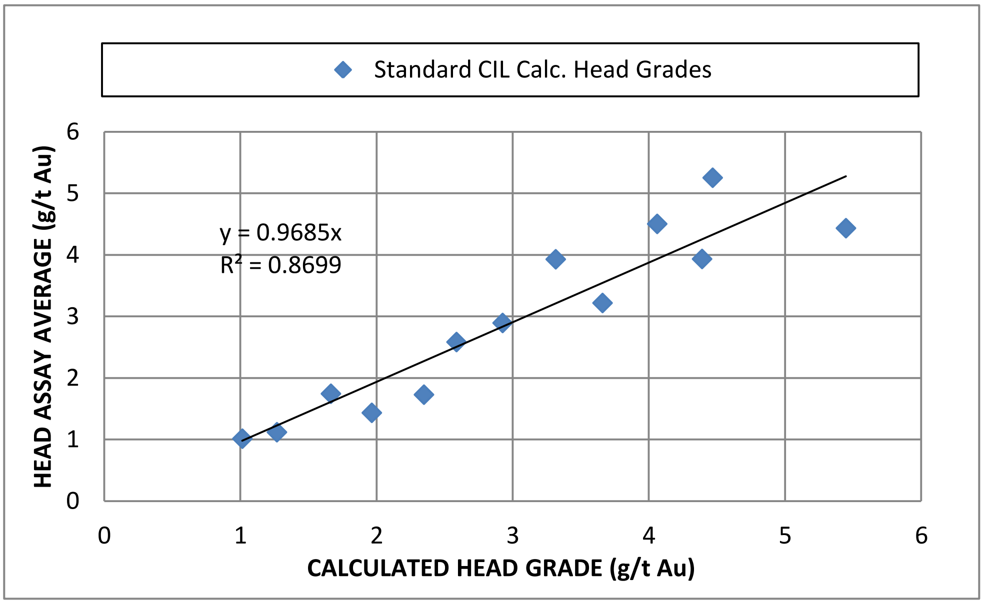


Figure 13-14 Geological Log and Assay Head Grades

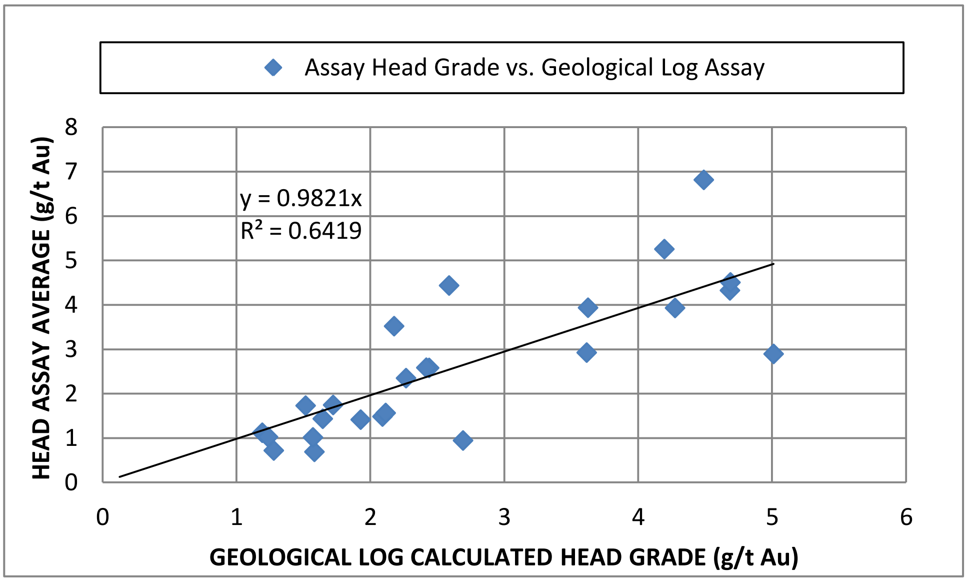
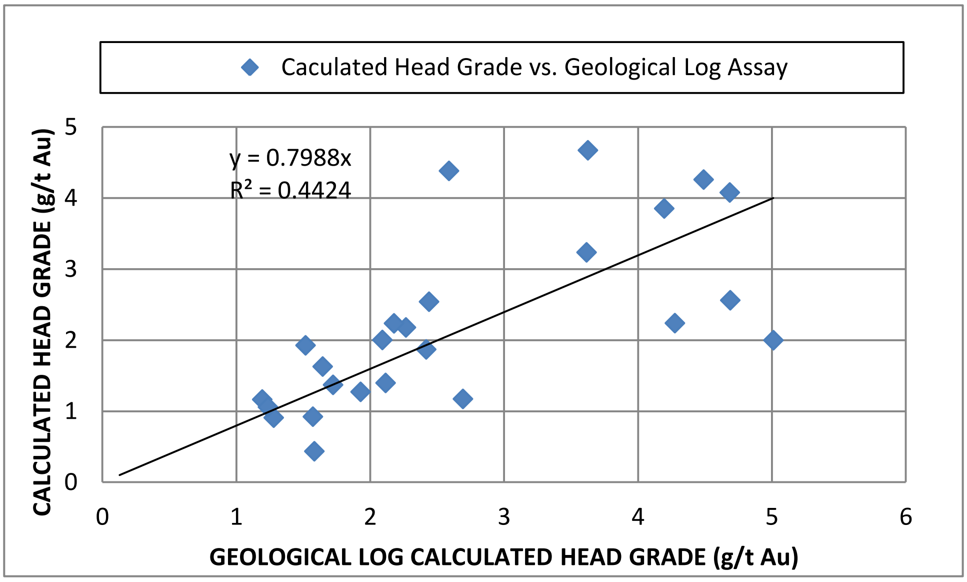


Figure 13-15 Geological Log and Calculated Head Grades (Alternate Flowsheet)



13.1.8 Cyanide management

With the aim of reducing cyanide levels in the tailings to meet discharge limits, testwork using the SO_2/Air oxidation process was conducted on CIL tailings obtained from the leaching of the master composite at FS optimal conditions. The result of the preliminary detoxification testwork (Table 13-7) shows elevated iron and, hence, elevated calculated total cyanide concentrations.

Applicable Peruvian regulations state that a water discharge from a mine site must contain an average of less than 0.8 mg/L of total cyanide with a short term peak value of less than 1 mg/L. For the process plant, in the event that a positive water balance is achieved, then a detoxified solution recovered from subsequent thickening stages may have to be discharged from site. Also, seepage and surface run-off associated with detoxified filtered tailings moisture is also governed by this legislation.

Preliminary detoxification testwork resulted in total cyanide levels which were much higher than the discharge limits, suggesting that further treatment would be required if the process water were to comply.

Table 13-7 Preliminary Detoxification Results Summary

Cu mg/L	Fe mg/L	Hg mg/L	Ni mg/L	Zn mg/L	CN_{WAD} mg/L	Calculated CN total mg/L
0.84	105	<0.02	0.03	0.01	5.50	299

Further testwork was conducted to investigate the combination of detoxification with a subsequent stage of iron precipitation using zinc sulphate (as $ZnSO_4.7H_2O$) to reduce total cyanide levels. The testwork results (Table 13-9) indicated detoxification followed by iron precipitation was able to reduce the cyanide levels (CN_{total}) to meet discharge limits. Reduced CN_{total} content in iron precipitation was achieved at almost a stoichiometric ratio of zinc sulphate.

Table 13-8 Detoxification/Iron Precipitation Results Summary

	Concentration in Product (After $ZnSO_4$)					Calculated	Reagent Consumption		
$NaCN_{FREE}$	Cu	Fe	Ni	Zn	CN_{WAD}	CN total	SMBS	$CuSO_4$	$ZnSO_4$
mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L	kg/t	kg/t	kg/t
N/A	0.54	0.1	0.03	2.32	0.13	0.41	3.7	0.2	1.3

Detoxification/iron precipitation testwork was carried out on samples which were not subjected to pre-aeration. Pre-aeration prior to CIL was included as part of the variability testwork flowsheet to reduce iron levels in CIL and, hence, CN_{Total} concentrations in the feed to detoxification. With the inclusion of pre-aeration, iron levels in CIL tailings decreased to approximately 30% of the iron encountered in the feed to detoxification/iron precipitation testwork. Stoichiometrically, this would result in a decrease in $ZnSO_4$ consumption to approximately 0.4 kg/t.

13.1.9 Comminution Testwork

Bond Ball Work Index (BWi) tests, Bond Rod Work Index (RWi) tests, and Bond Abrasion (Ai) tests were conducted on the comminution master composite and several variabilty composites. The master composite (BWi, RWi and Ai of 18.6 kWh/t, 22.1 kWh/t and 0.018 g/rev respectively) displayed similar comminution characteristics to PFS samples indicating a hard and non-abrasive ore.

Ai and RWi results for the variabilty composites were similar to results seen during the PFS. The BWi results for the variability composites indicated some variabilty with values ranging from 18.8 kWh/t to 22.7 kWh/t. The analysis of comminution results conducted for the FS compared with the historic results from previous studies are summarised in Table 13-9.

Table 13-9 Comminution Summary

	FS			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	-
80^{th} Percentile	20.6	23.7		19.5	24.1	0.012

These results show consistent behaviour and little variation.

13.1.10 Thickening, Filtration and Rheology Testwork

Thickening testwork was performed at Outotec on the tailings of the master composite (from leaching at FS optimal CIL conditions). The results of the testwork indicated thickening with a flux of 0.75 t/h/m^2 and flocculant dosage of 15 g/t can achieve an underflow solids concentration of 58% w/w. The increase in flux rate achievable compared to PFS testwork, which achieved 57% w/w solids with a flux of 0.25 t/h/m^2, can be explained as the result of the coarser FS grind size.

FS thickening testwork was performed using M10 as flocculant based on results of the PFS testwork. Clarity of the thickener overflow was found to be poor for tests undertaken at pH values that would be present under full-scale operating conditions. The testwork investigated the benefit of elevating pH with hydrated lime. At a pH of over 11.1, the overflow clarity was greatly improved. Based on this testwork, Outotec recommended thickening to be performed at pH 11.1. However, the design adopted in the FS does not allow for thickening at elevated pH.The testwork observation indicates the high concentration of divalent calcium cations has neutralised negative mineral repulsive charges. Furthermore, it suggests that a different flocculant with a higher cationic charge than M10 and/or the use of coagulants would be able to achieve the results seen at elevated pH. Current thickening testwork information was deemed adequate, however, for the purpose of optimising settling and subsequent filtration performance, further work is recommended.

Outotec performed several filtration tests, including disc filter and filter press tests using the same sample which was used for thickening testwork. Similar to the thickening testwork, the coarser grind has yielded improved filtration rates. Additionally, due to the removal of the majority of cyanide components during detoxification/iron precipitation testwork, the washing cycle for filtration was not required. The results show filtration rate of 228 kg/h/m^2 for the filter press was able to obtain a cake moisture content of 14.8% w/w.

Rheology testwork was conducted on the product from the slurry obtained from CIL leaching testwork and detox testwork (both performed on the master composite at optimal CIL conditions). The notable difference between the two slurries was the slurry pH (10.5 and 9 respectively) and the solution chemistry. Slurries were tested at solids concentrations of 40%, 50% and 60% w/w. Reduction in viscosity was observed with the reduction in pH. For both slurries tested, results indicate that slurry rheology should not pose constraints on pumping, agitation and screening. For both pH values tested, slurries at 40% w/w did not exceed 60 cPS in the tests. The shear stresses did not exceed 50 Pa at any stage of the testwork.

Anecdotal comment from the metallurgical testing laboratory stated that due to high viscosity of the HMPG slurry, the pulp density of the leach testing was reduced to some 30% w/w solids. A 30% w/w solids slurry pulp density has been used as the basis of design and no issues with viscosity are anticipated as a result.

13.2 Recovery Estimates

Analysis of the testwork results in Section 13.1 indicates that residue gold grade, and consequently overall gold extraction is influenced by:

- The gold feed grade to CIL 2 which varied with head grade.
- The extraction achieved in CIL 2 which varied for different ore zones.
- The association between organic carbon content and residue grade.

Algorithms which incorporate these factors were derived and integrated into the mine plan, for a LOM production of 8,730,300 t of high grade ore and 590,300 t of low grade development ore, to yield a life of mine residue of 0.30 g/t Au and a calculated life of mine gold recovery of 91.0% from a mine plan gold head grade of 3.38 g/t (3.5 g/t Au high grade ore, and 1.5 g/t Au low grade development ore).

13.2.1 Mine Plan and Mill Feed

Ollachea ore will be mined from Ore Zones 1, 2, 3, 4, 5 and 6. The process plant will treat Ollachea ore (high grade), as well as low grade development ore from these zones. Due to the variable mine production rate, stockpiling of material during months of peak mine production is required, as well as the reclaim of stockpiled material during months of low mine production. Two stockpiles will be required, a low grade low grade stockpile and a high grade ore stockpile. As a result the process plant will see a variety of feed materials.

A mill feed and stockpiling model was generated to understand the characteristics of plant feed and estimate the required capacities for stockpiles. The model has a resolution of one month and is in accordance with the FS mine plan (dated 2012-10-19). Within the model, feed to the plant is taken from various sources, in the following order of preference:

- High Grade Ore from mine in the following order: Ore Zones 1, 2, 3, 4, 5 then 6
- High Grade Stockpile (Blend)
- Low Grade Development Ore from mine in the following order: Ore Zones 1, 2, 3, 4, 5 then 6
- Low Grade Stockpile (Blend)

Testwork has indicated that different recoveries are obtained from the different ore zones, with Ore Zone 1 and 2 showing better overall extractions, whilst moderate overall extractions for Ore Zone 3 and 4 were observed. Ore Zones 5 and 6 gave poor extractions relative to other ore zones. Hence, the model allows for the preferential feed of material from ore zones which achieve better extractions.

Testwork has also indicated low extractions for lower grade ore (Figure 13-4). Higher grade ore will also be treated preferentially to low grade development ore to improve the economics of the project.

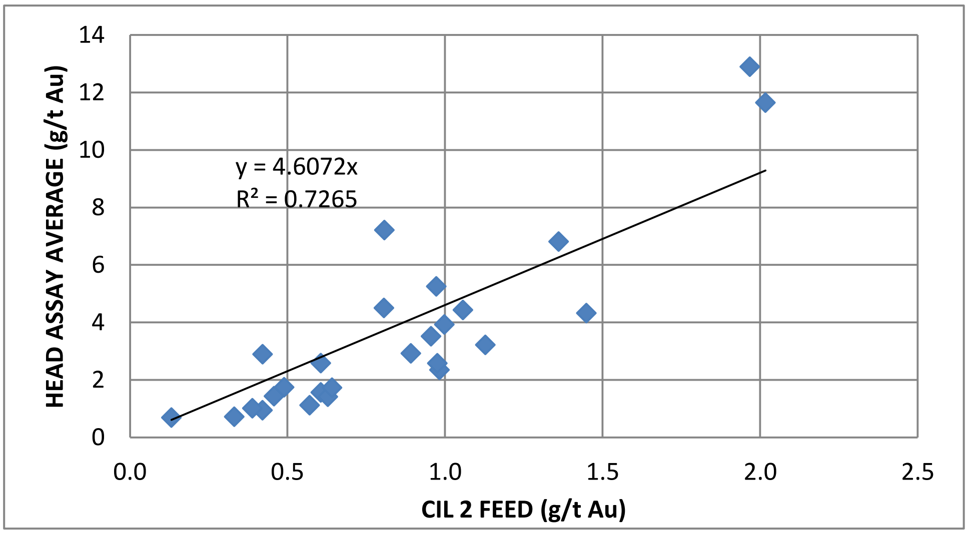
13.2.2 Feed to CIL 2

Analysis of the testwork results concluded that the use of average assay head grade for the purpose of estimating recoveries (as opposed to the more typical use of calculated head assay grades) is acceptable given the fit with geological log head grade.

Figure 13-16 presents the relationship between CIL 2 feed grade and the average assay head grades, this plot shows feed to CIL 2 can be calculated using the following equation:

CIL 2 feed grade = Head grade/4.6072

Figure 13-16 Alternate Variability Tests – Final CIL Feed Grade (Assay Head)



*One outlier removed

13.2.3 Final Residue Grade and Ore Zones

The correlation between the residue grade and the CIL 2 feed grade for Ore Zones 2, 3 and 5 derived from testwork results is summarised in Table 13-10.

Table 13-10 Ore Zone Correlations – CIL 2 Feed Grade

Ore Zone	Relationship
2	Residue = 0.1713 x (CIL 2 Feed Grade) + 0.0785
3	Residue = 0.3201 x (CIL 2 Feed Grade) + 0.0524
5	Residue = 0.1801 x (CIL 2 Feed Grade) + 0.3155

Testwork has also indicated that the organic carbon content, within the feed, negatively impacts residue grade. Analysis of organic carbon content of samples from different zones shows that the poorly performing ore zones are associated with high organic carbon assays.

For Ore Zones 1, 4 and 6, the numbers of samples tested were not sufficient to derive residue correlations. Samples from Ore Zone 1 indicated similar extractions and organic carbon assays to that of Ore Zone 2; consequently, the residue correlation for Ore Zone 2 was used for Ore Zone 1. Similarly, the correlation for Ore Zone 3 was applied to Ore Zone 4 and the correlation for Ore Zone 5 was applied to Ore Zone 6.

The relationship between the CIL 2 feed grade and the head grade was incorporated into the ore zone correlations, and an allowance of 0.0015 g/t Au was included for solution gold loss, to derive the equations presented in Table 13-11.

Table 13-11 Ore Zone Correlations – Head Grade

Ore Zone	Correlation
1 and 2	Residue = 0.1713 x (head grade/4.6072) + 0.0785 + 0.0015
3 and 4	Residue = 0.3201 x (head grade/4.6072) + 0.0524 + 0.0015
5, and 6	Residue = 0.1801 x (head grade/4.6072) + 0.3155 + 0.0015

13.2.4 Recovery

For the life of mine (LOM) recovery, a mathematical method is required to estimate the recovery for a given set of conditions. Such conditions include the flowsheet being modelled, the grade of the ore, the influence of other aspects such as sulphur and organic carbon levels as well as other parameters and conditions. Typically the residue grade of a block or parcel of ore is estimated and the leach recovery ascertained to provide production data for the block or parcel of ore.

Summation of these estimates provides the LOM recovery as well as providing information as to the economics of the project as may be determined by the associated financial modelling.

The mill feed and stockpiling model provides an estimation of monthly head grade and the tonnage of ore and low grade development ore from the different ore zones from the mine, as well as the stockpile grade and tonnage feeding the process plant.

The estimate of the monthly residue grade, which would be achieved based on the given feed, was based on the following approach:

Ore and low grade development ore from the mine: for the different ore zones, the monthly feed head grade was used in the associated correlations (Table 13-11) to determine the monthly residue grades

Material in high grade and low grade stockpiles: the ratios of ore zones in the stockpile were calculated, based on the material which is added to the stockpile and assuming that the material taken out of the stockpile is perfectly mixed. The stockpile combined grade, the ore zone ratios, and the different Ore Zone correlations (Table 13-11) were used to give a residue grade for the material in the stockpile at the end of the month.

Loss of gold, due mainly to GRG, is not expected to be significant for high grade stockpiled material, as the material is not preferably fed to the mill. Although low grade development ore will be stockpiled for extended durations, oxidation will not affect recovery due to the selected CIL process.

Calculated monthly residues are weighted, based on the ore production tonnage, to estimate residues for the calendar year. Table 13-13 shows the mining schedule, head grades and residue grades according to calendar year.

Yearly mining tonnage, head grade and residue grade for the various sources of mill feed is presented in Table 13-13. A life of mine recovery of 91.0% was found to be achievable for a LOM feed gold grade of 3.38 g/t.

Mining tonnage, head grade and residue grade for high grade ore from the different ore zones (excluding stockpiled material, accounts for 92.5% of feed material), weighted across the mine life is summarised in Table 13-12.

Table 13-12 Summary of Overall Extraction - High Grade Ore Exc. Stockpiled Material

Ore Zone	Fraction of Mined Ore % Total	Head Grade g/t Au	Residue Grade g/t Au	Extraction % Au
1	3.8	2.5	0.2	93
2	42.3	4.1	0.2	94
3	19.9	3.4	0.3	91
4	1.7	2.6	0.2	91
5	30.2	2.9	0.4	85
6	2.2	3.7	0.5	88
LOM		3.50	0.31	91.3

*Calculated based on weighted head grade and weighted residue grade

Table 13-13 Summary of Yearly Extractions

	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	LOM
HG Tonnage	571,134	802,983	1,089,432	1,087,515	1,088,856	1,099,181	1,092,996	862,304	720,833	203,175	8,618,408
HG Grade	3.43	3.07	3.48	3.67	3.64	3.82	3.59	3.25	3.51	2.91	3.50
HG Residue	0.32	0.33	0.33	0.32	0.32	0.30	0.30	0.27	0.28	0.20	0.31
HG Recovery	90.8	89.3	90.4	91.3	91.2	92.1	91.7	91.7	92.0	93.1	91.3
HG Stockpile Tonnage	35,308	21,570	7,650	7,042	8,191	8,870	5,990	0	17,271	0	111,893
HG Stockpile Grade	3.15	4.33	2.97	2.85	2.79	3.10	2.93	0.00	3.16	0.00	3.31
HG Stockpile Residue	0.37	0.48	0.43	0.43	0.42	0.44	0.43	0.00	0.44	0.00	0.42
HG Stockpile Recovery	88.3	88.8	85.5	85.0	84.8	85.9	85.3	0.0	86.1	0.0	87.2
LG Tonnage	44,886	48,139	12,356	27,218	17,084	20,519	3,824	0	0	0	174,026
LG Grade	1.49	1.56	1.48	1.37	1.38	1.46	1.68	0.00	0.00	0.00	1.48
LG Residue	0.29	0.29	0.27	0.17	0.13	0.13	0.20	0.00	0.00	0.00	0.23
LG Recovery	80.6	81.5	81.9	87.9	90.5	90.9	88.1	0.0	0.0	0.0	84.3
LG Stockpile Tonnage	29,014	65,667	7,894	4,891	6,121	3,392	34,019	265,272	0	0	416,271
LG Stockpile Grade	1.57	1.52	1.48	1.47	1.48	1.49	1.49	1.49	0.00	0.00	1.50
LG Stockpile Residue	0.23	0.26	0.27	0.25	0.21	0.19	0.20	0.20	0.00	0.00	0.21
LG Stockpile Recovery	85.4	83.0	81.6	82.8	86.0	87.2	86.8	86.8	0.0	0.0	85.9
Total Tonnage	**680,342**	**938,359**	**1,117,333**	**1,126,666**	**1,120,252**	**1,131,962**	**1,136,829**	**1,127,576**	**738,104**	**203,175**	**9,320,599**
Total Grade	**3.21**	**2.91**	**3.45**	**3.60**	**3.59**	**3.76**	**3.51**	**2.83**	**3.50**	**2.91**	**3.38**
Total Residue	**0.31**	**0.32**	**0.33**	**0.32**	**0.32**	**0.30**	**0.30**	**0.25**	**0.29**	**0.20**	**0.30**
Total Recovery	**90.2**	**88.9**	**90.3**	**91.2**	**91.2**	**92.0**	**91.6**	**91.1**	**91.8**	**93.1**	**91.0**

13.3 Comments on Section 13

An extensive testwork program was conducted at ALS Ammtec (Perth) on Ollachea samples for the Definitive Feasibility Study. The results obtained from this program are considered sufficient for FS level testwork with adequate data generated to understand the ore's metallurgical characteristics, be able to derive parameters required for design and to support the operating and capital estimates and financial analysis.

Testwork has indicated that the metallurgical response of the Ollachea ore zones will be characterized by:

- A significant component of gravity recoverable gold (GRG)
- Partial preg-robbing given the presence of carbonaceous material and
- Moderate double refractory component, with some gold locked in silicates and sulphides (minor arsenopyrite and dominant pyrrhotite)

Use of CIL and blanking reagents are techniques that have been tested, and will be employed to reduce the influences of preg-robbing minerals in the ore, such that almost all of the leachable gold will be recovered by the activated carbon in the CIL.

Iron precipitation (by $ZnSO_4$ addition) subsequent to detoxification forms the basis for meeting cyanide total discharge limits. For this process a single test was conducted which yielded results within the legislated limits. Further testwork should be conducted to confirm the outcome of this test, under varied ore zones and process conditions to ensure discharge integrity con be maintained.

14 MINERAL RESOURCE ESTIMATE

14.1 Key Assumptions/Basis of Estimate

Coffey Mining has estimated an Indicated and Inferred Mineral Resource for the Minapampa Zone of the Project as at 6th July 2012. All grade estimation was completed using Ordinary Kriging (OK) for gold. This estimation approach was considered appropriate based on a review of a number of factors, including the quantity and spacing of available data, the interpreted controls on mineralization, and the style of mineralization. The estimation was constrained within mineralised geological-grade interpretations that were created with the assistance of MKK geologists.

14.2 Geological Models

Seven high-grade domains have been interpreted using north-south-oriented vertical sections based on grade information and geological observations from Coffey Mining and MKK's geologist, consistent with the previous interpretation.

Interpretation of the Ollachea geological sections has been based upon information obtained from drill hole core-logging which compiles the different lithological, mineralogical, structural and alteration characteristics in the Minapampa area.

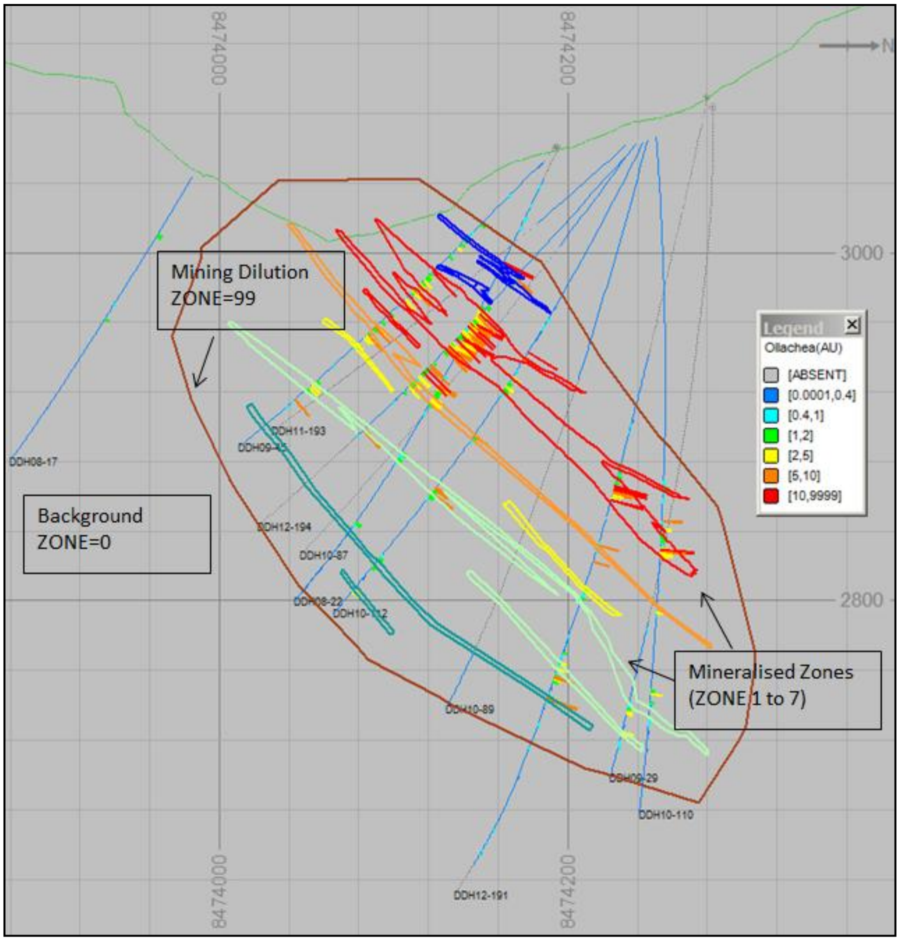
The topographic surface is based on a total station survey provided to Coffey Mining by MKK

14.2.1 Mineralized Zones

For the purpose of Mineral Resource estimation, seven main high-grade mineralized domains were interpreted and modelled on a lower threshold of 1.0 g/t Au corresponding to the lower limit of mineralization having reasonable spatial continuity.

The Ollachea interpretation was restricted to the high-grade, relatively continuous zones (ZONE 1 to 7), within each ZONE, there are sub-domains of continuous mineralization called LODE's. A low-grade envelope (Zone 99) was also modelled around the main mineralized zones to account for mining dilution. Background mineralization (Zone 0) was also modelled. The modelled domains are shown in Figure 14-1.

Figure 14-1: Cross Section of Geological Model – 339,160 mE



Interpretation and digitizing of all constraining boundaries was undertaken on cross sections orthogonal to the drill line orientation. The generated wireframes were all snapped to the available drill core data.

The resultant digitized boundaries have been used to construct wireframes defining the three-dimensional geometry of each interpreted feature. The interpretation and wireframe models were developed using the commercially-available CAE Datamine (Studio 3) mining software package.

14.2.2 Oxidation Divisions

No oxidation delineation was made. Due to the minor effect of weathering and oxidation in the project area, all material is treated as fresh.

14.2.3 Sample Flagging

The wireframes generated were used to flag various constraints in the drilling by ZONE, LODE or sub-zone.

14.2.4 Treatment of Missing / Absent Samples

Un-sampled intervals less than 5 m are treated as missing (i.e. grade=absent). This was the maximum sample interval sampled, in areas adjacent to the mineralized zones, and missing intervals less than 5 m are assumed to be due to core recovery issues.

Un-sampled intervals greater than 5 m and the first un-sampled interval in every drill hole, usually colluvial cover material, are treated as barren (i.e. grade=0.0025 g/t Au).

14.3 Exploratory Data Analysis

Descriptive and distribution statistics were compiled based upon the 2 m composite gold (Au g/t) data and the raw (un-composited) data. The interpreted data relevant to resource estimation studies was coded to the composite data.

14.3.1 Summary Statistics – Raw Data

Table 14-1 presents the summary table of the raw statistics, grouped by mineralized zone.

Table 14-1 Summary Statistics of Raw Gold Assays by Zone (g/t Au)

Zone	Description	Count	Min	Max	Mean	Std. Dev.	Variance	CV
0	Background	11811	0.003	82.54	0.136	1.384	1.917	10.183
99	Dilution Zone	15899	0.003	137.64	0.227	1.522	2.318	6.697
1	Min. Lens 1	226	0.019	42.55	3.092	3.965	15.719	1.282
2	Min. Lens 2	915	0.01	153	5.071	10.183	103.699	2.008
3	Min. Lens 3	418	0.012	118	3.981	7.828	61.282	1.966
4	Min. Lens 4	80	0.116	23.84	3.121	3.428	11.749	1.098
5	Min. Lens 5	587	0.003	121.45	3.469	7.233	52.309	2.085
6	Min. Lens 6	172	0.013	51.29	3.348	6.997	48.955	2.09
7	Min. Lens 7	57	0.063	17.04	2.596	2.539	6.448	0.978

14.3.2 Summary Statistics – Composite Data

Two metre composite statistics based on the mineralized zones are listed in Table 14-2.

Table 14-2 Summary Statistics of 2 m Gold Assay Composites by Zone (g/t Au)

Zone	Description	Count	Min	Max	Mean	Std. Dev.	Variance	CV
0	Background	16,521	0.003	82.54	**0.08**	0.85	0.72	11.19
99	Dilution Zone	12,156	0.003	36.05	**0.20**	0.72	0.52	3.66
1	Min. Lens 1	178	0.019	42.55	**3.12**	3.94	15.49	1.26
2	Min. Lens 2	633	0.038	153.00	**5.07**	9.03	81.60	1.78
3	Min. Lens 3	304	0.012	61.61	**3.87**	5.58	31.12	1.44
4	Min. Lens 4	63	0.116	23.84	**3.10**	3.46	11.98	1.12
5	Min. Lens 5	410	0.003	66.43	**3.26**	4.92	24.23	1.51
6	Min. Lens 6	142	0.014	51.29	**3.52**	7.51	56.40	2.14
7	Min. Lens 7	57	0.201	17.04	**2.67**	2.51	6.32	0.94
Total 1-7	MINZONE=1	1,787	0.003	153.00	**3.98**	6.86	47.02	1.72

14.4 Density Assignment

The Ollachea database contains 777 bulk density measurements. Table 14-3 summarises bulk density determinations by ZONE.

Table 14-3 Summary Statistics of Density Determinations by Zone (g/cm³)

Zone	Count	Min	Max	Mean	Median	Std. Dev.	Variance	CV
0	376	2.626	3.12	**2.818**	2.82	0.059	0.003	0.021
99	298	2.595	2.988	**2.794**	2.805	0.069	0.005	0.025
Total 0,99	674	2.595	3.12	**2.808**	2.815	0.065	0.004	0.023
1	10	2.71	2.887	**2.823**	2.832	0.052	0.003	0.018
2	31	2.605	2.92	**2.814**	2.821	0.081	0.007	0.029
3	22	2.719	3.11	**2.836**	2.821	0.079	0.006	0.028
4	2	2.663	2.831	**2.747**	2.663	0.118	0.014	0.043
5	28	2.747	2.96	**2.858**	2.87	0.052	0.003	0.018
6	5	2.662	2.856	**2.761**	2.733	0.085	0.007	0.031
7	5	2.656	2.868	**2.745**	2.679	0.102	0.01	0.037
Total 1-7	103	2.605	3.11	**2.824**	2.834	0.077	0.006	0.027

A bulk density of 2.80 g/cm³ has been assigned to all waste block (MINZONE=0) and a bulk density of 2.83 g/cm³ has been assigned to all mineralized blocks (MINZONE=1) within the current model below the topographic surface.

14.5 Grade Capping/Outlier Restrictions

High-grade capping (cutting) was determined for each zone. The composite data for each of the mineralized zones generally had a positively skewed grade distribution characterised by differences between mean and median grades, and moderate to high coefficients of variation (CV, standard deviation/mean). The CV is a relative measure of skewness and values greater than one can often indicate distortion of the mean by outlier data.

The requirement for high-grade caps was assessed via a number of steps to ascertain the reliability and spatial clustering of the high grade composites. The steps completed as part of the high-grade cap assessment included:

- A review of the composite data to identify any data that deviate from the general data distribution. This was completed by examining the cumulative distribution function
- A review of summary statistics comparing the percentage of metal and change in CV caused by the high-grade cuts
- A visual 3D review to assess the clustering of the higher-grade composite data.

Based on the review, appropriate high-grade caps were selected for each zone. The application of high-grade caps resulted in relatively few data being capped. The capping has resulted in minor reduction in mean grade except for Zone 6, where the capping of six outlier values resulted in a 19% reduction in mean grade.

A cap of 0.9 g/t Au was applied to Zones 0 and 99, due to the presence of highly variable, higher grades within the dominantly lower-grade zones. The capping was required to reduce the amount of metal which would be artificially added during the estimation process in these zones.

The summary statistics for the 2 m composite data, calculated for uncut and cut values, are presented in Table 14-4.

Table 14-4 Cut and Un-cut Composite Statistics

ZONE	Element	Uncut				Cut					% Change in
		Number Data	Mean	Std. Dev.	CV	Upper Cut	Mean	Std. Dev.	CV	Number Data Cut	Mean
1		178	3.12	3.92	1.26	20	3.00	2.87	0.96	1	-4.1
2		633	5.06	9.03	1.78	40	4.80	6.24	1.30	3	-5.3
3		304	3.87	5.57	1.44	22	3.62	3.78	1.04	4	-6.5
4		63	3.10	3.43	1.11	18	3.00	2.91	0.97	1	-3.0
5	Au(g/t)	410	3.26	4.92	1.51	25	3.12	3.66	1.17	3	-4.3
6		142	3.52	7.48	2.13	20	2.85	3.92	1.38	6	-19.0
7		57	2.67	2.49	0.93	NC	2.67	2.49	0.93	0	0.0
99		12156	0.20	0.72	3.66	0.9	0.16	0.21	1.34	321	-19.8
0		16521	0.08	0.85	11.19	0.9	0.05	0.11	2.29	136	-35.6

14.6 Composites

The drill hole database was composited to a 2 m down hole composite interval within each of the zones. The composite datasets were completed using Datamine mining software package and its COMPDH function using a residual retention routine, where residuals are added back to the adjacent interval. The majority of composite lengths are 2 m, with a small amount of composite lengths ranging from 1 to 3 m and mean lengths equal to 2 m. The global effect of the compositing produces negligible effect to the total length and mean grade. A decrease in the sample variance is noted as a natural effect of compositing. The 2 m composite files were used for all statistical, geostatistical and grade estimation studies. The decision to use 2 m composites was based on the targeted underground mining method which will have a relatively high level of mining selectivity. The majority of the sampling has been collected using 1 m sample intervals.

14.7 Variography

Experimental correlograms were calculated and modelled using the Isatis geostatistical package. General aspects of the variography are:

- Experimental correlograms were calculated from capped 2 m composite data. Downhole and directional correlograms were generated. Variogram orientations reflected obvious trends for strike, dip and thickness in the data.
- The variogram for the combined mineralized zones was based on the dataset for Zones 1 to 7. The variography for Zones 2, 0 and 99 was based on the respective data subsets.
- Variograms were modelled with a nugget effect and two nested spherical structures.

- Within the mineralized zones, the total range in the major direction varied from 90 m for Zone 2 to 120 m for the combined variogram model. Ranges are greater than the average drill hole spacing which is a nominal 30 m x 30 m grid. For the low-grade zones, the total range in the major direction varied from 160 m for Zone 99 to 220 m for Zone 0.

- The relative nugget effect or short-scale variability in the mineralized zones was 45%, displaying a moderate degree of short-spaced variability. For the lower-grade zones the nugget effect ranges between 50% for Zone 0 and 55% for Zone 99

14.8 Estimation/Interpolation Methods

A three dimensional block model was generated to enable grade estimation and mine planning and mine design. A parent block size of 20 mE x 20 mN x 4 mRL was selected with sub-blocking to a 2 mE x 2 mN x 0.4 mRL cell size to improve volume representation of the interpreted wireframe models.

The sample search strategy was based upon analysis of the variogram model anisotropy, mineralization geometry and data distribution, a summary of the interpolation parameters is given in Table 14-5.

Table 14-5 Interpolation Parameters

Zone	0	99	1	2	3	4	5	6	7
Major Range (m)	150	150	100	100	100	100	100	100	100
Semi-major Range (m)	150	125	100	100	100	100	100	100	100
Minor Range (m)	150	56	25	25	25	25	25	25	25
Major Dip (°)	0	0	9.9	9.9	9.9	9.9	9.9	9.9	9.9
Major Az (°)	0	90	80	80	80	80	80	80	80
Semi-major Dip (°)	0	45	43.3	43.3	43.3	43.3	43.3	43.3	43.3
Semi-major Az (°)	90	360	341	341	341	341	341	341	341
Minor Dip (°)	90	45	45	45	45	45	45	45	45
Minor Az (°)	0	180	180	180	180	180	180	180	180
Minimum Composites	8	8	8	8	8	8	8	8	8
Maximum Composites	16	16	16	16	16	16	16	16	16
Search Volume Factor	2	2	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Minimum Composites	4	4	4	4	4	4	4	4	4
Maximum Composites	16	16	16	16	16	16	16	16	16
Search Volume Factor	-	-	-	-	-	-	-	-	-
Minimum Composites	-	-	-	-	-	-	-	-	-
Maximum Composites	-	-	-	-	-	-	-	-	-
Maximum Comps/DH	4	4	4	4	4	4	4	4	4

Grade estimates were interpolated into parent cells and all sub-cells were assigned the parent cell grades.

During estimation runs, the block model was coded with the number of composites selected, the average distance of composites, Slope of Regression, Kriging Variance, Block Variance, and Kriging Efficiency %, which were later used in the determination of the resource classification.

14.8.1 Depletion for Underground Workings

A large majority of historical underground workings and those recently developed by artisanal mining have been surveyed. Coffey Mining reviewed the data and determined that the majority of artisanal workings are within 10 m of the natural surface, although individual workings do go deeper. In order to account for some depletion in the resource model, all blocks within 10 m of the surface were flagged as depleted cells and mineral resources are reported only for non-depleted blocks.

14.9 Block Model Validation

14.9.1 Volumetric Validation

A comparison between the measured volumes of the solids generated during the geological modelling and the volume of mineralization in the block model was carried out and indicated that the volume of mineralized blocks in the block model corresponds well to the volume of the mineralized wireframes.

14.9.2 Block Model Comparison against Drill Data

A detailed validation of the OK estimate was completed for each zone and included both an interactive 3D and statistical review. The validation included a visual comparison of the input data against the block model's grade in plan and cross section. It also included review of the distribution of recorded estimation controls including search pass, average sample distance, number of contributing samples and drill holes.

A spatial comparison of the mean grade of the input composites against the block model's grade was also made. The models were divided into slices by directions (Easting and RL) and average grades calculated for the various domains. Similarly, the composite averages and de-clustered composite averages were also computed. Examination of these plots indicated that the models were appropriately honouring the input data and trends.

14.10 Classification of Mineral Resources

The Mineral Resource estimates for the Ollachea Project (Minapampa zone) conform to the requirements of CIM Definition Standards (2010) and Australasian Code for Reporting of Identified Mineral Resources and Ore Reserves, published by the Joint Ore Reserves Committee (JORC) of the Australasian Institute of Mining and Metallurgy, the Australian Institute of Geoscientists, and Minerals Council of Australia, 2004. The criteria used to categorise the Mineral Resources include the robustness of the input data, the confidence in the geological interpretation including the continuity of both structures and grades within the mineralized zones, the distance from data, and amount of data available for block estimates within the respective mineralized zones.

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 (2010) categorization guidelines. A summary of the criteria considered and confidence level of the QP is listed in Table 14-6.

Table 14-6: Mineral Resource Confidence Criteria and Assessment

Items	Discussion	Confidence
Drilling Techniques	Diamond drilling is Industry standard approach.	High
Logging	Standard nomenclature and apparent high quality.	High
Drill Sample Recovery	Good recovery recorded except in shear/fault zones.	High
Sub-sampling Techniques & Sample Preparation	A 1 m sampling method has been implemented, with only minor areas within the mineralization still sampled at 2m intervals.	Moderate
Quality of Assay Data	Available field duplicate data shows a moderate precision.	Moderate
Verification of Sampling and Assaying	Umpire samples have shown good precision	Moderate-High
Location of Sampling Points	Survey of all collars with down hole survey completed for most holes.	Moderate to High
Data Density and Distribution	Approximately 30 m x 30 m spaced drilling in the Minapampa zone has provided adequate data for an inferred / Indicated resource. Infill to 20 m x 20 m or better will be required to increase the confidence of the current interpretation.	Moderate
Audits or Reviews	Audits have been routinely completed, last one by Smee (2011), on laboratory and QA/QC procedures. All issues identified have been rectified.	High
Database Integrity	Assay hard copy sheets were checked against the digital database with no errors identified	High
Geological Interpretation	The current 7 high grade zones are preliminary but relatively robust. Mineralization appears parallel to the dominant foliation, and has been confirmed by orientated core measurements	Moderate
Estimation and Modelling Techniques	Ordinary Kriging has been used to obtain estimates of Au g/t grade. Coffey Mining used a two pass estimation method for all blocks.	High
Cut-off Grades	A threshold of 1 g/t Au was used to define the high grade envelopes. Estimated results are reported above a 2.0 g/t Au cut-off.	Moderate-High
Mining Factors or Assumptions	None.	N/A

An Inferred Mineral Resource confidence category was assigned for blocks:

- Having an estimated Au grade
- Within the mineralized zones (Zone 1 to 7).

The Indicated Mineral Resource confidence category was assigned to blocks:

- Located in a portion of the deposit with a density of drilling of approximately 40 m x 40 m or better, and an estimated grade greater than 2 g/t Au.
- With a slope of regression for the Au OK estimate that is greater than 0.47
- Where the distance to the nearest sample used in the Au OK block estimate is within 0.3 (30%) of the first pass search ellipse radius.

A Datamine string file produced in section, and checked in plan, was used to define the final Inferred and Indicated zones. The resulting wireframes were used to code the resource confidence categories to the block model.

14.11 Reasonable Prospects of Economic Extraction

Mineral Resources are reported above a cut-off grade of 2.0 g/t Au and within three-dimensional geological wireframes constructed to constrain the gold mineralization in the Mineral Resource estimate to zones defined by mineralized diamond drill core intersections. Mineral Resources above a 2.0 g/t Au cut-off grade have reasonable prospects for economic extraction, based on mineralization continuity, shape and distribution and as demonstrated in Section 15.

14.12 Mineral Resource Statement

Mineral Resources for the Ollachea property (Minapampa) above a 2.0 g/t Au cut off consist of 10.6 Mt of Indicated Mineral Resources with an average grade of 4.0 g/t Au and 3.3 Mt of Inferred Mineral Resources with an average grade of 3.3 g/t Au. Mineral Resources were estimated by Doug Corley, MAIG, R.P. Geo, of Coffey Mining Perth, a Qualified Person under National Instrument 43-101, and have an effective date of 6 July, 2012 (Table 14-7).

The Mineral Resources replace those declared in the September 2011 Technical Report (AMEC, 2011) and previous estimates declared for the Property. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Mineral Resources are reported inclusive of Mineral Reserves that are discussed in Section 15.

Table 14-7 Mineral Resources for the Ollachea (Minapampa) Project

Mineral Resources above a 2.0 g/t Au Cut-off Grade	Tonnage (Mt)	Au Grade (g/t)	Contained Au (Moz)
Minapampa			
Indicated	10.6	4.0	1.4
Inferred	3.3	3.3	0.3

Note:
Mineral Resources are inclusive of Mineral Reserves.
Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
Mineral Resources are reported above a cut-off grade of 2.0 g/t Au and within three-dimensional geological wireframes constructed to constrain the gold mineralization in the Mineral Resource estimate to zones defined by mineralized diamond drill core intersections. Tonnages are metric tonnes and ounces of contained gold are troy ounces. Mineral Resources above a 2.0 g/t Au cut-off grade have reasonable prospects for economic extraction, based on mineralization continuity, shape and distribution and as demonstrated in this study. Mineral Resources are estimated by Doug Corley, MAIG, R.P. Geo, QP, of Coffey Mining and have an effective date of 6 July, 2012.

14.13 Factors That May Affect the Mineral Resource Estimate

At the current level of drill sample density, there are no known factors which may affect the Mineral Resource estimate.

However increased drilling density and bulk density sampling may identify unknown structures and effect the interpretation and reporting tonnages of future Mineral Resource estimations.

15 MINERAL RESERVE ESTIMATES

The Mineral Reserve estimates are based on the Indicated Mineral Resources as declared in Section 14. No Measured Mineral Resources have been declared.

15.1 Geological Design Influences

The *in situ* ounces of gold per vertical and lateral step (15 m) using a 2.0 g/t Au COG for the Indicated Mineral Resource are shown in Figure 15-1 and Figure 15-2. Total ounces of gold are split into geologically interpreted mineralised zones.

Points related to Figure 15-1 are:

- The exploration incline is located around 2775 mRL. Approximately 85% of the *in situ* ounces are located above this RL.
- Zone 2 contains approximately 46% of the *in situ* ounces over approximately 315 vertical metres. This zone contains the mineralized lodes that have the greatest width.
- Zone 5 contains approximately 24% of the *in situ* ounces over approximately 450 vertical metres. This zone has mineralized lodes that are of significantly lesser width than the mineralized lodes in Zone 2.
- Zone 3 contains approximately 18% of the *in situ* ounces over a similar vertical distance to that of Zone 2. Zone 3 mineralized lodes are of a similar width to those of Zone 5.
- The remaining 12% of *in situ* ounces are contained in Zone 1, Zone 4 and Zone 6.

Figure 15-1: Ounces per 15m Vertical Step (Indicated Category Only)

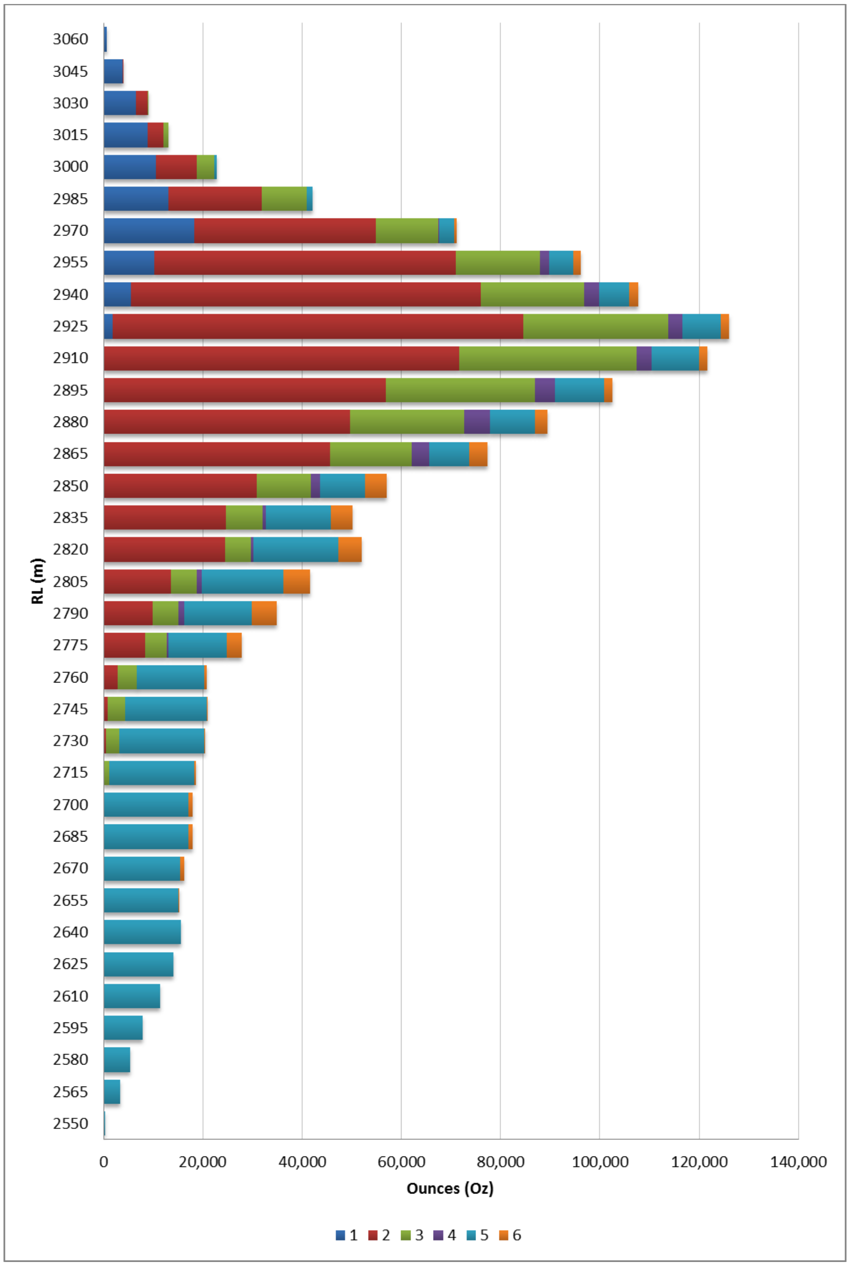
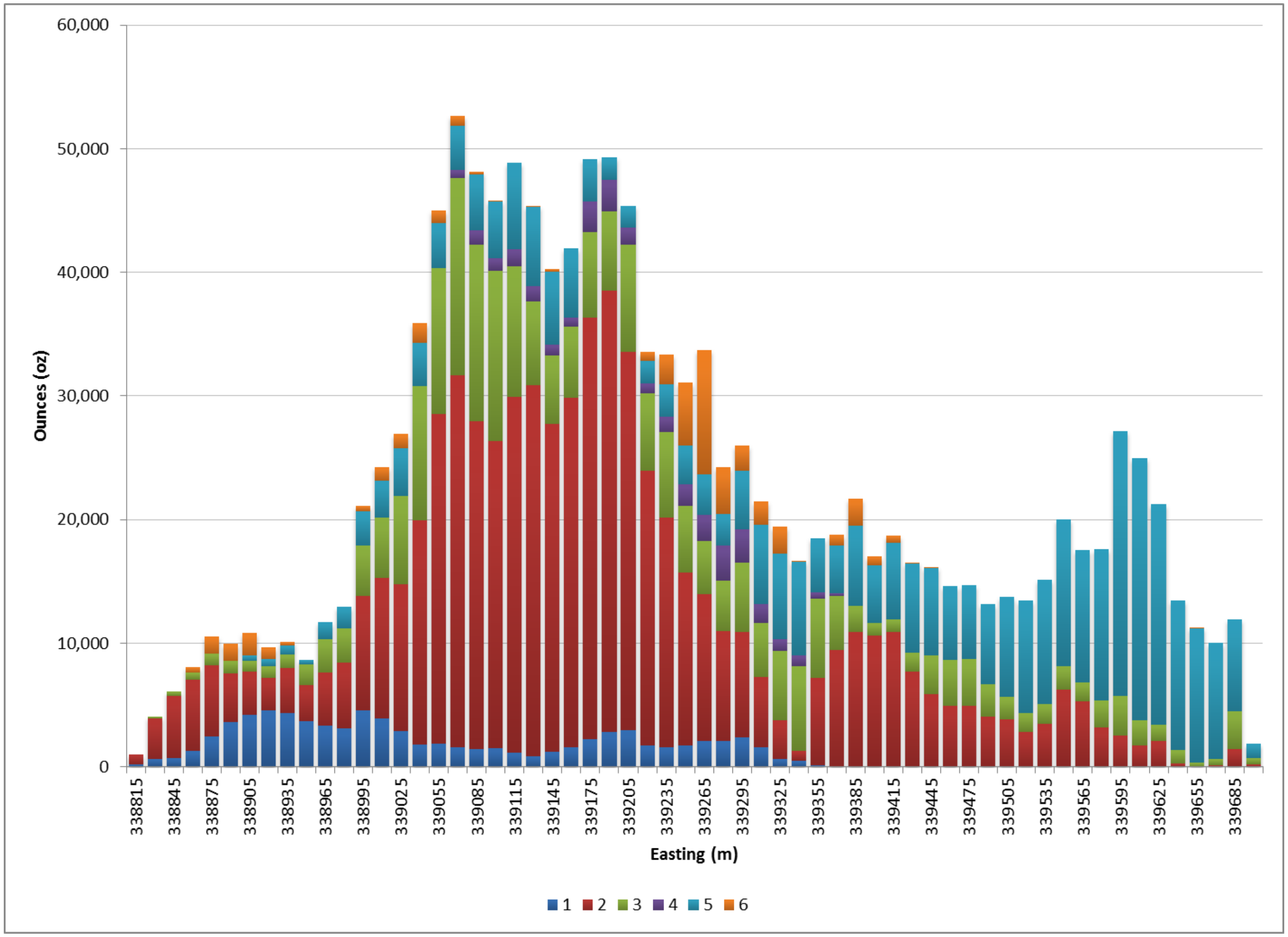


Figure 15-2: Ounces per 15 m Easting Step (Indicated Category Only)

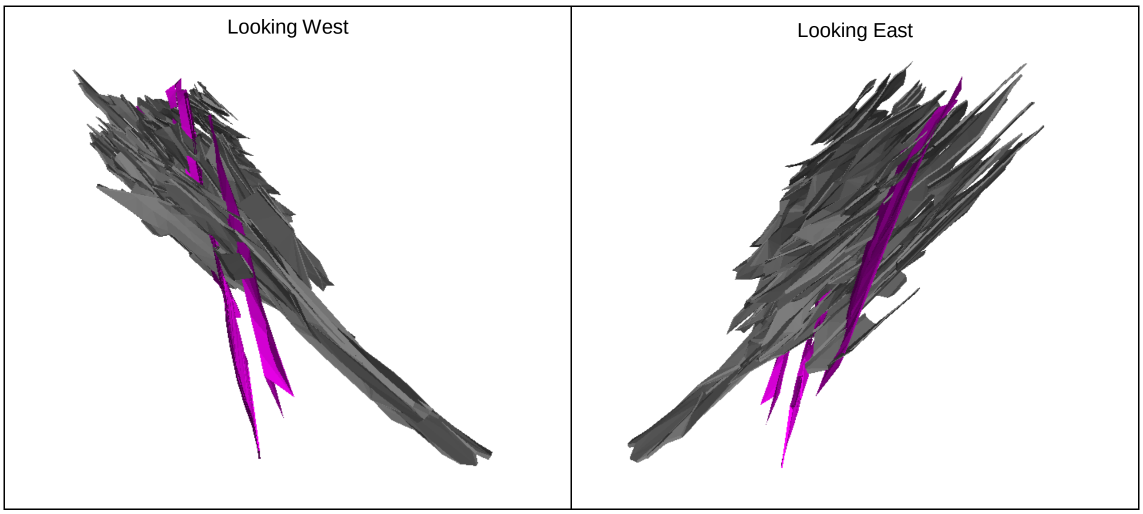


Points related to Figure 15-2 are:

- The majority of the *in situ* ounces are located towards the west where multiple stacked zones are located. Approximately 74% of the *in situ* ounces are located west of Easting 339370m.

A three-dimensional structural model was created as part of the FS. Included in this model is an interpreted shear zone that cuts through the mineralised zones at an oblique angle. An area of very poor rock quality was identified within this zone. The mine design process took cognisance of this area. The location of this shear zone in relation to the interpreted mineralised zones is shown in Figure 15-3.

Figure 15-3: FS 3D Structural Model Shear Zone



15.2 Geotechnical Design Influences

Geotechnical mine design constraints and considerations are outlined in Section 16.5. These include:

- A minimum 30 m surface crown pillar thickness.
- Supported stable stope strike length ranging from 13 m to 23 m dependent on stope width and location relative to surface.
- Footwall to hangingwall extraction sequence.
- Recommended minimum stand-off distance of 30 m for capital infrastructure.
- Recommended minimum stand-off distance of 15 m for level access drives.

15.3 Cut-off Grade

No project COG optimisation was completed during the FS. The selected COG is based on the application of a single COG for the life of mine. The basis of the calculation for a FS mine design COG was the costs and recoveries estimated at the end of the PFS. Table 15-1 shows the calculated Project *in situ* Au break even grade based on operating costs and recoveries estimated for a range of gold prices.

Table 15-1: Break Even Grade Estimate

Parameter	PFS Cost and Recoveries						FS
Gold Price (US$/oz)	850	1,000	1,150	1,300[12]	1,450	1,600	1,300
Mill Recovery (%)	91.33	91.33	91.33	91.33	91.33	91.33	91.04
Mining Recovery & Dilution (%)	85	85	85	85	85	85	81
Mining Cost (US$/t ore)	18.48	18.48	18.48	18.48	18.48	18.48	23.40
Mill Cost (US$/t ore)	24.26	24.26	24.26	24.26	24.26	24.26	21.50
G&A Cost (US$/t ore)	3.87	3.87	3.87	3.87	3.87	3.87	4.30
Realisation (US$/t ore)	0.53	0.53	0.53	0.53	0.53	0.53	0.35
Royalty (US$/t ore)	3.04	3.04	3.04	3.04	3.04	3.04	2.89
Total Cost (US$/t)	50.18	50.18	50.18	50.18	50.18	50.18	52.44
***In situ* break even grade Au g/t**	2.37	2.01	1.75	1.55	1.39	1.26	1.70[13]

A mine design cut-off grade of 2.0 g/t Au was used for the FS. The financial outcome of the PFS and company strategic objectives were key considerations in the selection process. The Project Mineral Resource is also reported above a 2.0 g/t Au COG.

Table 15-1 includes the calculated break-even grade based on the final FS costs and recoveries at the base case gold price. This aligns with and supports the COG grade selected as the basis of the FS mine design, and does not affect the outcome of the project.

15.4 Mining Limits

Determination of the FS underground mining limits was completed by taking cognisance of the economic, geological and geotechnical constraints, and considerations and using three primary software processes:

- CAE Mining (formerly Datamine) Mineable Shape Optimiser (MSO): creation of stope shapes.
- CAE Mining Studio 5DP (formerly Mine2-4D): mine design, sequence and geological resource model evaluation.

[12] DFS Base Case gold price

[13] Break even grade based on DFS costs, recoveries and base case gold price

- CAE Mining Enhanced Production Scheduler (EPS): mine development and production schedule and final design element reserve classification.

The second and third parts of the process were iterative.

No modifications or additions were made to the FS geological resource model for use in the creation of the mine design and schedule.

15.4.1.1. MSO Process

The initial mineable limits of the study were identified by using the MSO process. This uses various user defined variables such as cut-off grade, stope dimensions, minimum mining width, minimum waste pillar width and dilution factors to generate three-dimensional stope shapes based on the defined criteria. As part of the process, the stope shapes were evaluated against the user-selected geological resource model and each stope's physicals were exported to a results file that could be scrutinised.

For the FS, multiple runs were completed to determine input variable sensitivity and the practicality of the shapes generated. Final variable selection was based on the sensitivity analysis.

Key final criteria or data used in the MSO process were:

- FS geological resource wireframes that represent the location of the lode mineralization;
- FS resource block model;
- dilution: 0.3 m on each wall for a total of 0.6 m;
- minimum mining width: 2.0 m (2.6 m with dilution);
- minimum waste pillar width: 7.5 m lode true width;
- minimum mining unit dimensions of 15 m high by 15 m along strike by lode thickness (a 7.5 m half stope option for the strike direction was used in MSO to account for lode pinch out or changes in mineralisation grade).

The selection of a 15 m by 15 m mining unit dimension limit is based on Project geotechnical recommendation, mining practicalities (drilling and support installation), lode geometry and the nature of the MSO process and economic considerations.

Stope shape COG sensitivity analysis was also completed as part of the process. This showed in lower-grade areas that small increases in the stope shape COG above 2.0 g/t Au impacted on the continuity of adjacent stopes located in the same lode.

Essentially, as the stope shape COG is increased, less ore is defined and more isolated stopes or groups of isolated stopes are created. This outcome would result in the cost effectiveness of stope access development being reduced.

15.4.1.2. Studio 5DP Process

The process steps completed in Studio 5DP were:

- Import stope shape strings created by using the MSO process and re-wireframe and re-evaluate against the geological block model.
- Flag stopes that meet the FS ore criteria.
- Complete mine development and infrastructure design based on flagged stopes.
- Evaluate combined mine development and stope designs.
- Create mine design activity dependencies based on selected mining sequence.
- Export mine design activities, including mine physicals data, and dependencies to EPS.

15.4.1.3. EPS Process

The process steps completed in EPS were:

- Determine final material classification by cut-off grade (>=2.0 g/t Au) and mine design shape resource classification (Indicated only) and flag activities as ore.
- Apply practical resource constraints e.g. development jumbos, stope production rates, to allow a practical development and production schedule to be determined.
- Export final mine design and schedule data for use in the mining cost model.

15.5 Low Grade Development Ore

Economic value has been identified in processing low grade development ore below the project COG at different stages of the Project life. The material to be processed is sourced from stope access development that traverses through Indicated Mineral Resources but has been diluted below the Project COG of 2.0 g/t Au. A mill cut-off grade has been applied to this material as this material must be mined as part of the mine plan to provide access to the designed stopes.

Three stages of processing and stockpiling low grade development ore have been identified for the Project:

Stage 1 – Mill Ramp-up Period

Low grade development ore will supplement mill ore feed.

Stage 2 – Mill Steady State Period

Low grade development ore will be stockpiled and used as mill feed make up if underground production issues are encountered.

Stage 3 – Mill Ramp Down Period

Low grade development ore will supplement mill ore feed.

Table 15-2 shows the calculated Au mill break even grade based on FS operating costs and recoveries for a range of gold prices. To estimate a mill break even grade, mining costs are excluded, as these would already have been expensed, and are replaced with a material rehandle charge for stockpile reclaim.

Table 15-2: Mill Break Even Grade Estimate

Parameter	FS Costs and Recoveries					
Gold Price (US$/oz)	850	1000	1,150	1,300[14]	1,450	1,600
Mill Recovery (%)	91.04	91.04	91.04	91.04	91.04	91.04
Rehandle Cost (US$/t ore)	2.00	2.00	2.00	2.00	2.00	2.00
Mill Cost (US$/t ore)	21.50	21.50	21.50	21.50	21.50	21.50
G&A Cost (US$/t)	4.30	4.30	4.30	4.30	4.30	4.30
Realisation (US$/t)	0.35	0.35	0.35	0.35	0.35	0.35
Royalty (US$/t)	2.89	2.89	2.89	2.89	2.89	2.89
Total Cost (US$/t)	31.04	31.04	31.04	31.04	31.04	31.04
Mill Feed break even grade Au g/t	1.25	1.06	0.92	0.82	0.73	0.66

A mill cut-off grade of 1.0 g/t Au was used for the low grade development ore in the FS.

15.6 Financial Model

The final step in the Mineral Reserve estimation process is to combine all estimated project capital and operating costs and summarise for input into the Project financial model.

The financial model was used to determine economic viability and sensitivity to changes in key project variables.

[14] DFS Base Case gold price

15.7 Mineral Reserve

From the outcome of the financial modelling step, Table 15-3 shows the Mineral Reserve estimate for the Project, based on a COG of 2.0 g/t Au.

Table 15-3: Mineral Reserve Estimate (November 29, 2012)

Classification	Tonnes (Mt)	Au Grade (g/t)	Contained Gold (koz)
Ore (+ 2 g/t Au)	8.7	3.5	983
Low Grade Development Ore (+1 g/t to 2 g/t Au)	0.6	1.5	28
Probable Mineral Reserves	**9.3**	**3.4**	**1,011**

Notes:
Probable Mineral Reserves are included within Indicated Minerals Resources and are declared inclusive of mining dilution with an effective date of 29 November, 2012.
Tonnages are metric tonnes and ounces of contained gold are troy ounces.
Probable Mineral Reserves are declared based on a base case gold price of US$1300 / oz, a project COG of 2.0g/t, LOM project operating costs of US$49.2 /t ore and a mill recovery of 91.04%.
Low Grade Development Ore is sourced from development drives that traverse through Indicated Mineral Resources but has been diluted below the project COG of 2.0 g/t Au. As the mining cost for this material will have already been expensed, it is economic to treat through the plant. A mill cut-off grade of 1.0 g/t Au has been applied to this material.
Mineral Reserves were estimated under the supervision of John Hearne, BEng(Mining), MBA, FAusIMM, CP(Mining), of Coffey Mining, and who is recognized as a Qualified Person for the purposes of National Instrument 43-101.

The Mineral Reserves are included within the declared Indicated Mineral Resource and is declared inclusive of mining dilution. The low grade development ore is sourced from development drives that traverse through Indicated Mineral Resources but has been diluted below the Project COG of 2.0 g/t Au. As the mining cost for this material will have already been expensed, it is economic to treat through the plant. A mill cut-off grade of 1.0 g/t Au has been applied to this material.

The Mineral Reserve estimate has been determined and reported in accordance with the CIM Definition Standards (2010).

The reported Mineral Reserve has been compiled under the supervision of John Hearne, FAusIMM (CP), and an employee of Coffey Mining, and who is recognized as a Qualified Person for the purposes of National Instrument 43-101.

A summary of the main parameters used in estimating the Mineral Reserve are shown in Table 15-4.

Table 15-4: Main Parameters used for the Mineral Reserve Estimate (November 29, 2012)

Description	Units	Value
Gold Price	US$/oz	1,300
Mine Design Au Cut-off Grade	g/t	2.0
Mill Au Cut-off Grade	g/t	1.0
Mining Method		LHOS
Minimum Mining Width (excluding dilution)	m	2.0
Annual Production Rate	Mt /a	1.1
Mining Operating Cost	US$/ t ore	23.4
Milling Operating Cost	US$/ t ore	21.5
G&A Operating Cost	US$/ t ore	4.3
Mining Dilution - Development.	%	21

Mining Dilution - Stopes.	%	19
Mining Recovery (within mine design shape)	%	100
Mill Recovery	%	91.04
Project Capital Cost	US$M	177.5
Sustaining Capital Cost	US$M	41.5
Closure Cost	US$M	4.2
Royalties	%	3.3
Special Mining Tax (SMT) or Especial de Mineria (IEM)	%	3.1
Workers Profit Share	%	7.3
Corporate Income Tax	%	25.1

The data that supports the Mineral Reserve estimate is discussed in other sections of this technical report and was obtained from the sources listed in Table 15-5.

Table 15-5: Mineral Reserve Estimate Sources of Supporting Information (November 29, 2012)

Modifying Factor	Source
Mineral Resources	Coffey Mining
UG Geotechnical Engineering	Coffey Mining
UG Mine Design and Scheduling	Coffey Mining
UG Mine Cost Estimation	Coffey Mining
Hydrology and Hydrogeology	AMEC
Metallurgical Test-work and Process Design	AMEC
Process Plant, Backfill Plant and Infrastructure Design	AMEC
Process Plant, Backfill Plant and Infrastructure Cost Estimation	AMEC
Environmental	AMEC
Marketing	MKK
Financial Modelling	MKK
Property and Land Tenure	Tong (2010b), Arevalo (2011) & MKK
Social	MKK

The underground mine design includes a 30 m crown pillar as recommended by geotechnical analysis to maintain the stability of the upper valley floor and retain watercourse integrity. Economic stopes located in this area were excluded from the Mineral Reserve estimate. Further geotechnical work is recommended to optimise the size of this pillar.

Artisanal mining activities in the Project area have been taken into account in the Mineral Resource estimate process (Section 14). Analysis has indicated these activities as captured in the geological block model were confined to the exclusion area defined by the 30 m crown pillar.

The Mineral Reserve estimate excludes economic stopes that are located in close vicinity to the widest part of the shear zone that is discussed in Section 15.1. The viability of recovering economic material contained within these stopes requires future evaluation once new geological and geotechnical data becomes available.

Surface raisebore holes that form the basis of the primary ventilation circuit have been located based on surface topographic constraints and with the intent to minimise underground level access development. This has resulted in the holes passing through or close to economic stopes. These stopes have been excluded from the Mineral Reserve estimate. The viability of recovering economic material contained within these stopes towards the end of the mine life requires further consideration.

To minimise the production tail created by adopting the selected FS stope extraction sequence for the western part of the mine a sill pillar level has been planned. This pillar splits this part of the mine into two mining areas. Adopting this strategy has resulted in the sterilisation of economic material that is located directly below the sill pillar level and contained in the wider mineralised lodes. An alternative stope extraction sequence has been identified that may negate this sill pillar requirement. This should be evaluated and compared to the current stope extraction sequence to determine if project financial benefits exist.

A reconciliation of Indicated Mineral Resources contained within the Probable Mineral Reserve is shown in Table 15-6.

Table 15-6: Indicated Mineral Resources to Probable Mineral Reserves Ounces Reconciliation (November 29, 2012)

Description	Contained Gold (Moz)
Indicated Mineral Resources	**1.4**
MSO Stope Design Process Exclusions	0.28
Crown Pillar Exclusions	0.05
Shear Zone Exclusions	0.02
Raise bore Exclusions	0.01
Western Sill Pillar Exclusions	0.03
Production Tail Exclusions	0.002
Indicated Mineral Resources contained within Probable Mineral Reserves	**0.99**

Totals may not sum due to rounding

There is a potential for mining losses to be incurred at paste stope interface, especially in the wider stopes. Good operational management and control combined with appropriate technical application will mitigate this potential. The Mineral Reserve estimate does not account for any mining losses related to this issue.

Project risks and opportunities have been captured qualitatively and are outlined in Section 25.2.

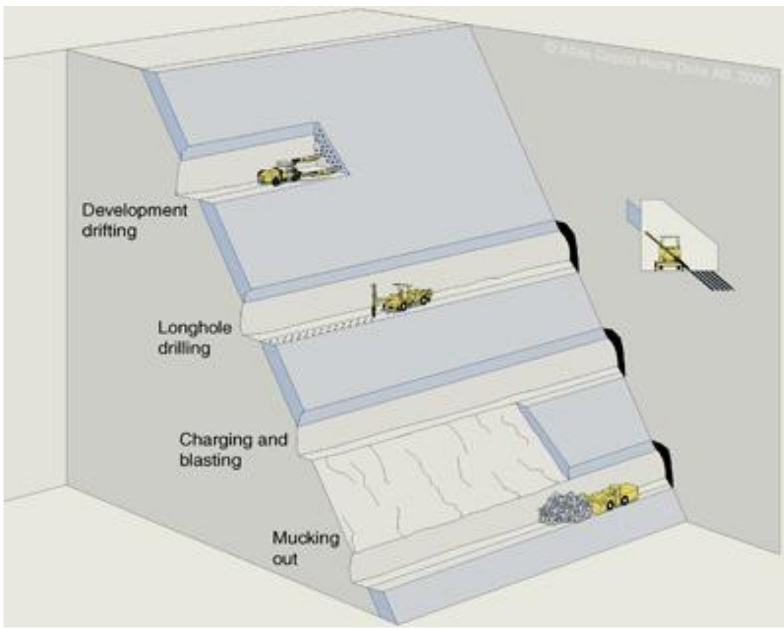
Project financial sensitivity analysis has been completed and is outlined in Section 22.5. This shows the project to be economically viable within the range of variation of the parameters considered.

16 MINING METHODS

The mining method selected for the FS was long hole open stoping (LHOS) with paste backfill, which can also be referred to as bench stoping with paste backfill. Extraction occurs along the orebody strike direction on a retreat basis.

Stopes will be accessed longitudinally (along strike) on each level by, one, two or three strike ore drives dependent on lode thickness. Figure 15-3 shows a generic interpretation of the main components of the LHOS mining method excluding the paste backfilling, which occurs after mucking out. Open stope strike length is dictated by geotechnical considerations and varies with lode width

Figure 16-1: Typical View of the Selected Longitudinal Mining Method



(Sourced from Atlas Copco)

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

16.1 Long Hole Open Stoping with Paste Fill

Key mining method selection parameters and objectives are:

- Stacked multiple lode orebody. Seven zones interpreted with multiple lodes per zone.

- Non-visual orebody (no defined contact and no visually distinct variation between mineralized and non-mineralized material)
- Predominant dip of lodes between 40° and 60°. (see Section 7.2)
- Orthogonal lode thickness between 0.5 m to 48 m.
- Deposit grade distribution requires production volume to provide acceptable mine economics.
- Maximize economic extraction of the delineated Mineral Resource.
- Maximize underground disposal of waste rock and process plant tailings.
- Minimize surface disturbance.

Key selection characteristics of the LHOS with paste backfill mining method are:

- Mining is fully mechanized.
- Provides a suitable mix of productivity, low cost and selectivity.
- High ore recovery and controlled dilution is possible.
- Can be used longitudinally or transversally.

16.2 Mine Design Process

Key mining method selection parameters and objectives are:

- Stacked multiple lode orebody. Seven zones interpreted with multiple lodes per zone.
- Non-visual orebody (no defined contact and no visually distinct variation between mineralized and non-mineralized material)
- Predominant dip of lodes between 40° and 60°. (see Section 7.2)
- Orthogonal lode thickness between 0.5 m to 48 m.
- Deposit grade distribution requires production volume to provide acceptable mine economics.
- Maximize economic extraction of the delineated Mineral Resource.

- Maximize underground disposal of waste rock and process plant tailings.
- Minimize surface disturbance.

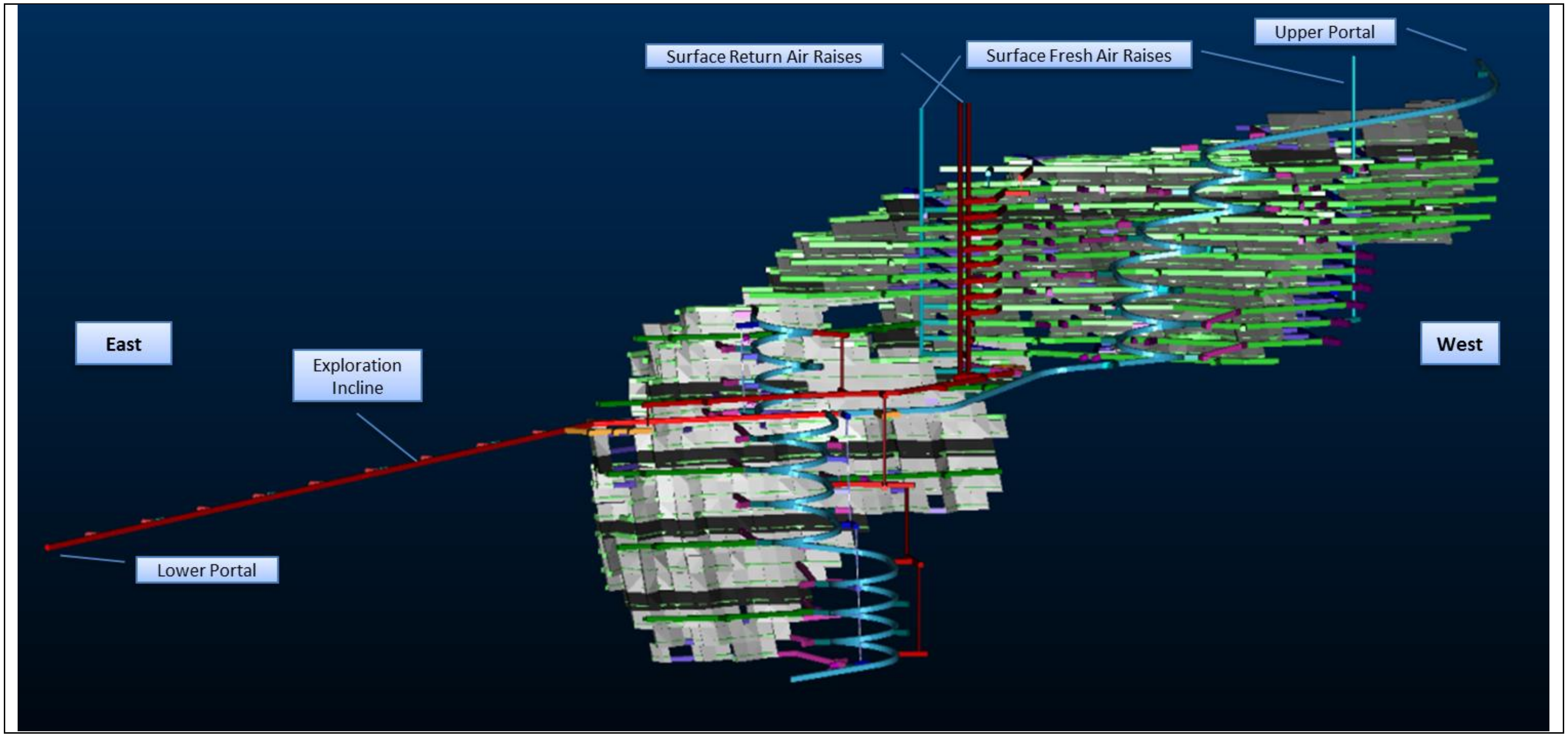
Key selection characteristics of the LHOS with paste backfill mining method are:

- Mining is fully mechanized.
- Provides a suitable mix of productivity, low cost and selectivity.
- High ore recovery and controlled dilution is possible.
- Can be used longitudinally or transversally.

16.3 Final Mine Layout

The final FS mine layout is shown in Figure 16-2.

Figure 16-2: Isometric Mine Layout Looking South



The main access to the mineralisation will be via a 1.2 km-long exploration access incline (1.5%) which has its portal in a valley on the north-eastern side of Cerro Joropiña and the Oscco Cachi River valley. The drive is currently being excavated and will be used for exploration drilling as well as providing the primary access and haulage drive for the planned mine. This portal (lower) will be the main mine access portal and is located above the process plant area at 2765 mRL.

An incline drive and a decline drive will be excavated at a grade of one in seven from the main exploration incline, located at approximately 2782 mRL, to access the eastern part of the mine. The decline drive will extend to 2550 mRL to service the deepest planned mining level at 2565 mRL. The incline drive will extend to a mining level at 2865 mRL.

The main exploration incline will be extended as an incline drive at a grade of one in seven and will be developed to meet a decline drive that will be developed simultaneously from a second (upper) portal at 3060 mRL. These drives when connected will provide a second means of egress, access to all the mineralisation in the western part of the mine, and early establishment of the primary ventilation system.

The mine is split into two main production areas, east and west, with the western part of the mine providing approximately 71% of the life of mine production tonnage. All mining is completed using a bottom up mining direction.

To maximise mine extraction, the eastern part of the mine will be split into multiple mining panels consisting of four levels that can be mined simultaneously. The lowest level of each of these mining panels requires an artificial sill pillar to be created using high strength paste backfill to allow the mineralisation located directly below to be completely extracted. The western part of the mine has also been split to minimise the impact from the life of mine production tail. A sill pillar level has been located on 2940 mRL.

Due to the non-visual nature of the orebody, grade control diamond drilling is planned on a minimum of a 15 m by 15 m grid. In the eastern part of the mine, this will be completed from dedicated hanging wall drives that will provide coverage for four production levels. The western part of the mine will be grade control drilled on each level from the main hanging wall access drive. Mineralised zones will be re-interpreted from the grade control program; ore drives will then be driven primarily on survey control and backed by face and wall channel sampling. An onsite laboratory is planned and is this has been designed to provide a 24 hour turnaround of samples.

Production from the eastern part of the mine will start on 2790 mRL and 2805 mRL for the western part of the mine. The primary ventilation system will be fully established prior to the start of stope production.

Stope size will be controlled by the nature of the lodes (dip and width variability) and interpreted geotechnical conditions. Stope sublevel spacing will be 15 m vertically floor to floor. Planned stope strike length is based on geotechnical interpretation and varies between 13 m and 23 m dependent on lode width. To control the stability of the longitudinal stopes and minimise dilution, the length of open voids can be altered based on local ground conditions.

Production drilling will be medium diameter (76 mm or 89 mm) down holes with some requirement for up-holes when mining below an artificial sill pillar. Up-holes will also be used where lodes pinch out and there is no requirement for development above. Stope blast initiation (void) will be via the use of drop raise slots as the distance from the floor of the top cut to the back of the bottom cut will be approximately 10 m vertically or 14 m on dip.

To minimise dilution, maintain stability and maximise open stope strike length, cable bolts will be installed in the hanging wall of the stopes. A dedicated cable-bolter (drill and install) is planned to complete this activity. Development ground support installation will be completed by development jumbos.

Stopes will be backfilled using paste derived from mine tailings to maximise the resource extraction, provide long term mine stability and reduce the surface area required for waste and tailings disposal. Small quantities of waste rock will be used as a capping for tramming purposes on all paste filled stopes.

The primary ventilation system consists of the exploration incline, other incline and decline drives, four surface raises (two return air raises and two fresh air raises), and an internal return air system and connecting drive that services the eastern part of the mine. Primary fans will be located on the two surface return air raises.

16.4 Hydrogeology

AMEC completed a hydrogeological characterization of the Ollachea project site. The Ollachea project infrastructure is planned to be distributed along the lowermost elevation area of three sub-basins, near the Ollachea River: the Oscco Cachi, Challuno, and Cuncurchaca sub-basins. The three sub-basins are characterized by seasonal precipitation patterns, with a rainy season from December to March, a dry season from May to August and the remaining months of April, September, October, and November as transition months.

The geomorphology surrounding the Ollachea project site consists of moderate to steep slope mountains and ranges, with occasional valley areas, several of which will be occupied by the project components. As described in detail in Section 7, the geology of the Oscco Cachi river area consists of phyllites of the Devonian Sandia

Formation, and variably bedded graphitic slates and shales of the Siluro-Devonian Ananea Formation. South of these formations andesitic volcanic rocks outcrop. Both sedimentary and volcanic rocks are intruded by nepheline syenite to the south and granodiorite to the north. The surficial geology of the Cuncurchaca and Choyouno sub-basins areas is mainly comprised of sandstone rocks, with some granodiorite intrusions in the south side. Unconsolidated materials overlying the bedrock can be found along the active stream channels of the sub-basins (refer to Figure 7-1).

The rock sequence is highly affected by tectonic events, which have deformed and fractured the rock units. Evidence of these tectonic events is the two regional-scale thrust faults that cross the area: the Ollachea and the Paquillosi faults (refer to Figure 7-1).

AMEC has defined two main hydrogeological units: an unconsolidated deposit unit and a consolidated deposit unit.

- The unconsolidated unit comprises Quaternary deposits of alluvial, colluvial, and fluvial origin. These deposits are located at the sides of the streams and rivers; most of them appear to have been disturbed. The hydraulic conductivity of these deposits ranges in the order of 10^{-1} to 10^{-4} cm/s, according to field tests performed by AMEC during the feasibility study.

- The consolidated deposits are represented by the Ananea Formation rock units, which have hydraulic conductivities ranging from 10^{-6} to 10^{-8} cm/s, based on field work and drilling evidence. Also, these deposits are highly affected by fractures, which increase the secondary porosity and hydraulic conductivity.

AMEC developed a hydrogeological numerical model to understand the behavior of the groundwater system in the Ollachea project area. The model was based on and calibrated using the hydrogeological information obtained during the completion of the PFS and FS. The main goal of the model was to estimate the groundwater exfiltration into to the underground mine, and to determine the quantity of water that had to be extracted from the mine via dewatering. According to the pre-development modeling results (Figure 16-3), the groundwater flow is generally towards the Ollachea river, roughly parallel to the Oscco Cachi and Cuncurchaca streams.

It is estimated that the water flow rate from the underground mine will be up to 80 m^3/h during the exploration tunnel excavation, and will reach a flow rate of approximately 120 m^3/h during the production period (Figure 16-4). This numerical model is updated frequently based on the on-going water monitoring taking place in the exploration tunnel and recalibrated as more data become available.

Figure 16-3: Steady-State (pre-development) hydraulic heads of Ollachea project site, from calibrated numerical model

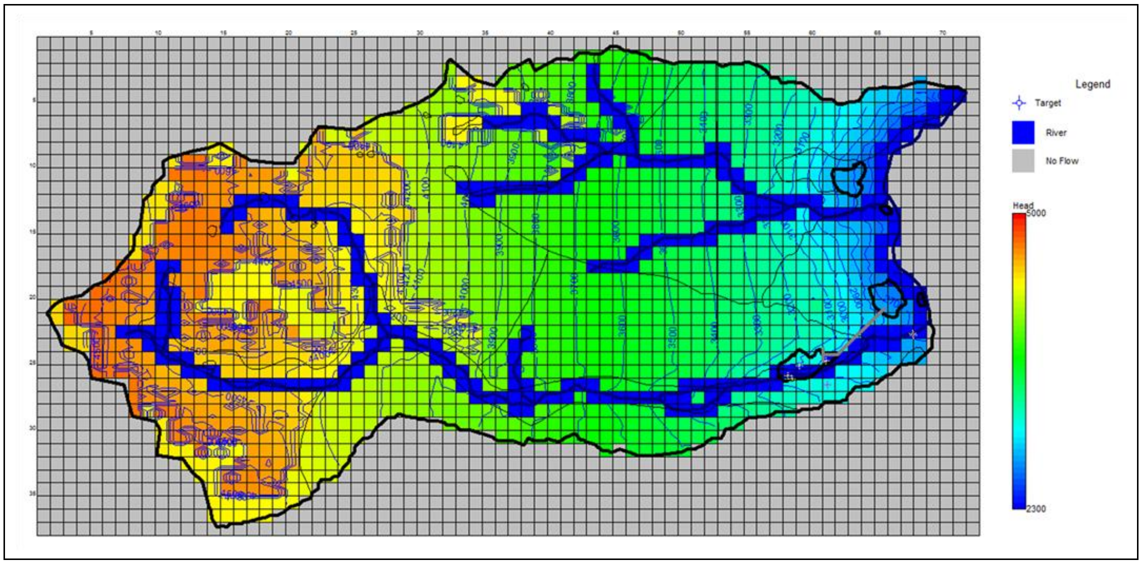
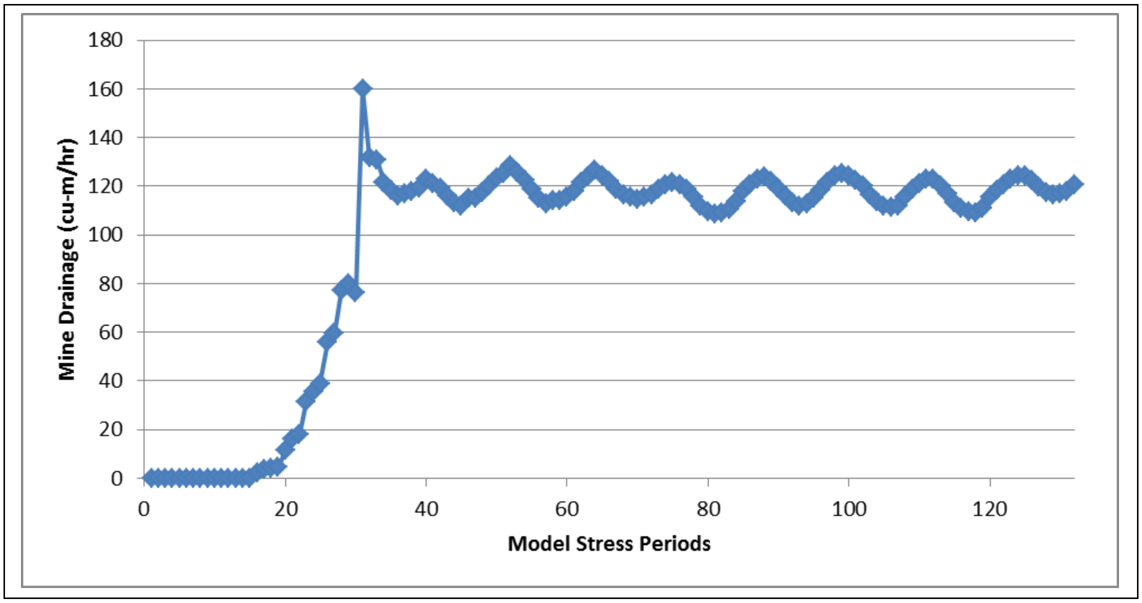


Figure 16-4: Monthly time series for mine drainage beginning in Jan-2011 to Dec-2020 (120 monthly stress periods)



Due to the nature of the planned mine development mine dewatering will be predominately gravity assisted. The water volumes estimated are not considered sufficiently large to present mine dewatering problems.

16.5 Geotechnical

The FS geotechnical assessment completed by Coffey Mining is based on core photographs and geotechnical core logging, structural studies (Telluris Consulting (2009) and Guzman and Aquino (2012)), rock test results (point load data, uniaxial compressive strength, triaxial & direct shear strength), and a site visit completed during January 2012.

Face mapping information from the current development of the exploration incline, which is now traversing the orebody host rock, has not be considered in the FS geotechnical assessment due to timing of data availability. It is recommended that this data be assessed on an ongoing basis and used to update the geotechnical analysis completed as part of the FS. This should form an integral part of the proposed project development

16.5.1 Rock Mass Conditions

The rock mass conditions assessment was based on the core log data, core photographs and intact rock strength properties. A total of 44 diamond drill holes were logged using the Rock Quality Index Q System.

16.5.2 Stress State

The Deformation Rate Analysis (DRA) technique was used to determine the local *in situ* stress tensor as summarised in Table 16-2.

Table 16-1: *In Situ* Stress State

Principal Stress	Trend
Major Principal Stress (σ_1)	y=0.0188x + 5.5749
Intermediate Principal Stress (σ_2)	y=0.0249x + 3.1312
Minor Principal Stress (σ_3)	y=0.0227x + 0.9247

16.5.3 Intact Rock Properties

The uniaxial compressive strength values of slate (the dominant rock type) range from 35 MPa to 69 MPa, with an average of 50 MPa, thus classifying the rock as of ‘medium strength’. Triaxial results range from 35 MPa (σ1) under 2 MPa (σ3) of confinement, to 67 MPa (σ1) under 10 MPa (σ3) of confinement. A mean friction angle of 33° and a mean cohesion of 0.1 MPa were determined from the nine direct shear tests. These properties were used as part of the basis to estimate rock mass strength and also as input variables to the excavation stability analysis and ground support selection.

16.5.4 Rock Mass Properties

The presence of discontinuities and other inherent defects in the rock mass downgrades intact rock strength. RocData software was used to determine the rock mass parameters which are presented in Table 16-2. The rock mass strength has been estimated to be 7.7 MPa. The data shown in Table 16-2 was used as input parameters to the numerical model.

Table 16-2: Rock Mass Properties

Hoek-Brown		Mohr-Coulomb	
mb	1.33623		
s	0.0026	Cohesion (C)	1.98983 MPa
a	0.505734	Friction angle (Phi)	30.4359°
sigt	-0.09728 MPa	sigcm	7.73277 MPa
sigc	2.4638 MPa		

16.5.5 Major Structures

Oriented core logs indicate that the foliation is the major structural feature followed by the sub-vertical faults striking parallel to the orebody. A bias in structural information is present in the data set due to the dominant drilling angle of the geotechnically logged holes.

Reports provided by MIRL identify three fault set groups and a shear zone. Of the identified major structures, the shear zone, which cuts obliquely across the orebody, is considered to present the greatest risk to the project. The shear zone material has been classified as “extremely poor”, however, its thickness and persistence across the orebody has not been accurately defined. To mitigate risk, stopes immediately adjacent to the thickest part of the interpreted shear zone have been excluded from the Mineral Reserves estimate. As new geological and geotechnical data become available the structural model should be updated.

16.5.6 Numerical Modelling

Non-linear finite element numerical modelling was undertaken using the ABAQUS/FEA program. The modelling results indicate that zones of relaxation and tensile failure, in both footwall and hangingwall regions, will result in void instability unless adequately supported. Rock mass damage due to accumulated plastic strain is also seen to be problematic when remaining waste pillar dimensions are insufficient. Restrictions on stope excavation sequencing (footwall to hanging wall only), long anchor cablebolt support and the provision of critical pillar dimensions (minimum pillar ratio of 1.5 times the drive width) will lessen these impacts.

16.5.7 Rock Mass Characterisation

The Modified Rock Mass Quality Index (Q') was utilised to characterise the in situ rock mass at Ollachea. Based on the information available, Q' is calculated to be from 1.8 to 4.0, which is rated as 'Very Poor' to 'Poor'. In the absence of any notable difference in rock mass conditions in terms of lithology, quality index and structure, geotechnical domains were based upon geometry of the orebody.

16.5.8 Mine Development

16.5.8.1. Ground Support Requirements

Ground support recommendations for the capital and ore drive development are based on the Q index. The analysis indicates that 2.4 m long rock bolts installed on a systematic pattern with nominal spacing of 1.3 m to 1.7 m, depending on the type of surface support, will provide safe ground conditions. In capital development, fibre reinforced shotcrete (FRS) is recommended as surface support at a nominal thickness of 50 mm. Galvanised weld mesh is recommended in the ore drives due to the short life and the importance of having the orebody visible for geological interpretation.

16.5.8.2. Large Excavation Support Requirements

Fully encapsulated twin strand cable bolts of up to 8 m long at 2 m spacing are to be installed in the backs of excavations with a span greater than 6 m, such as intersections and passing bays.

16.5.9 Stable Stope Analysis

16.5.9.1. Stable Span Methodology

The stability graph method, after Potvin (1988) and Nickson (1992), was used to assess the maximum stable spans for the stoping geometry at Ollachea. Under this method, the stable stope dimensions expressed in the form of the hydraulic radius (HR) are determined from the modified stability number N', which is a function of the rock mass quality index Q'.

16.5.9.2. Maximum Supported Hydraulic Radius for Stopes

Based on the calculated hydraulic radii of the different surfaces, the dimensions for stable stopes are given in Table 16-3. Fully grouted long anchor cable bolts will be used to provide reinforcement to the exposed stope walls and backs. The required cable bolt density was determined as 0.25 to 0.23 bolts per m^2. A minimum cable bolt length of 6 m was determined from the numerical modelling results and empirical charts.

Table 16-3: Stable Stope Dimensions

Stope Face	HR		Stope Dimension (m)									
Back	5.7	Length (L)	12.00	**15.00**	20.00	25.00	30.00	35.00	40.00	45.00	50.00	100.00
		Width (W)	247.00	48.26	26.75	21.10	18.50	17.00	16.03	15.35	14.48	12.92
Hangingwall	5.5	Inclined Height (H)	21.2									
		Length(L)	**23.0**									
Footwall	6.4	Inclined Height (H)	21.2									
		Length(L)	32.0									
End	7.1	Inclined Height (H)	21.2									
		Width (W)	334.0									
Back (Crown pillar)	5.0	Length (L)	12.00	15.00	20.00	25.00	30.00	35.00	40.00	45.00	50.00	100.00
		Width (W)	60.48	30.12	20.05	16.70	15.03	14.03	13.36	12.88	12.52	11.13

16.5.9.3. Recommended Stope Dimensions

The main controlling factor on the maximum recommended stope dimensions is the width of the stope backs, which, in turn, is controlled by the orebody width. For the maximum ore width of 45 m, the stable support strike length of the stope is calculated to be 15 m.

Where orebody widths are ≤ 21 m, the strike length can be increased to 23 m, which is calculated as the maximum stable supported stope strike length.

Stope strike lengths are further reduced where they intersect the poorer ground conditions found in the crown pillar domain in the uppermost stopes. Here stable stope strike lengths are limited to 13 m for the maximum ore body width of 45 m. Calculated stable strike lengths of 25 m are possible when stope widths are restricted to ≤16 m.

16.5.10 Infrastructure Stand-Off

Based on results of the numerical modelling analysis, infrastructure stand-off distances from the planned productions stopes have been determined. Table 16-4 provides a summary of the minimum stand-off distance used as design criteria for the FS mine design.

Table 16-4: Infrastructure Stand-off Distances

Development Category	Design Life span	Acceptable level of Strain damage (ε^p_{eq}) or Relaxation (Sigma 3)	Recommended Stand-off distances	Additional Restrictions / controls
Capital Development and infrastructure	Life of Mine	$\leq 0.5\%$ (ε^p_{eq}) and / or No relaxation or tensile failure	$\geq$ 30m	None
Exploration and Access drives	Medium term	$\geq 0.5\% \leq 1\%$ (ε^p_{eq}) and / or minor relaxation or tensile failure	$\geq$ 15m	Some rehabilitation Subjected to No-Entry exclusions when adjacent footwall voids are present
Ore drive development	Short to medium term	$\geq 1\%$ (ε^p_{eq}) and / or significant relaxation or tensile failure	0 – 15m	Rehabilitation required. Subjected to No-Entry exclusions when adjacent footwall voids are present

16.5.11 Crown Pillar Stability Assessment

Empirical analysis of the crown pillar, based on the work of Carter & Miller (1992) and Carter (2000) indicates a minimum crown pillar thickness of 30 m, based on a restriction of the adjacent stopes to 7 m wide and 15 m long. This equates to a Factor of Safety (FOS) of 1.2. This is a 50% increase in crown pillar thickness from recommendations made in the PFS. Additional work is necessary to further investigate and define controlling variables.

16.5.12 Shaft Stability Assessment

Two possible locations for vertical surface ventilation shafts were subject to a shaft stability exercise to provide an indication of raise boring risk and provide LOM support recommendations.

Drilling and geotechnical logging of dedicated diamond drill holes in close proximity to the final FS designed ventilation shaft locations is highly recommended. Rock mass reinforcement and surface support is anticipated to be necessary for any LOM shaft with a diameter greater than 3 m.

16.6 Mine Development Design

The location of the main mine accesses are in the orebody hanging wall. This was selected based on the location of the exploration incline, which is currently being developed, and planned process plant, the recommended stope extraction sequence of footwall to hanging wall, and because there is no discernible difference in the rock mass between hanging wall and footwall.

Orebody access development stand-off distances are based on geotechnical recommendations.

Key design parameters for lateral development are shown in Table 16-5. Other parameters are:

- maximum grade for access development is 1 in 7.
- minimum grade is zero (flat).
- minimum turning radius is 30 m.
- lateral development overbreak is 15%.

All production levels have been designed with no grade (flat), including level access crosscut development. This is required due to the complexity of the orebody, the strike extent of the orebody, small inter-level spacing and bottom-up mining direction. Drain holes for water will be drilled as required to remove water to lower, mined-out levels. Water from mining areas located above the main access incline will gravity drain. Water from mining areas located below the main access incline will gravity drain before being pumped to the main access level

Table 16-5: Lateral Development Design Parameters

Description	Design Criteria	Units	Value
DD_HW_Drive_Load Level_Access_Load_SP	Height Width Profile type (Radius) Profile area	m m m m²	6.0 5.0 Arch (4.0) 28.56
Incl_Decl Incl_Decl_SP Level_HW_Drive	Height Width Profile type (Radius) Profile area	m m m m²	5.5 5.0 Arch (4.0) 26.06
Infra_Crib Infra_Magazine Infra_Maint_Fuel Level_Access_Load_XC Level_Access_XC Level_Main_XC Infra_Crib	Height Width Profile type (Radius) Profile area	m m m m²	5.3 5.0 Arch (4.0) 25.06
Clean_Water_Sump DD_HW_Drive DD_HW_Drive_SP Dirty_Water_Sump Escape_Access Escape_Main_Access FAW_Drive Level_HW_Drive_SP Level_HW_Load_SP Level_ToOre_Drive Level_ToOre_Drive_LE Level_ToOre_Drive_LW Ore_Drive_LE Ore_Drive_LW RAW_Drive RAW_Drive_SP Stope_Slot_Drive	Height Width Profile type (Radius) Profile area	m m m m²	5.0 5.0 Arch (4.0) 23.56
DD_Cuddies	Height Width Profile type (Radius) Profile area	m m m m²	4.0 4.0 Square 16.00
Incl_Decl_Pass_Bay (stripping)	Height Width Profile type (Radius) Profile area	m m m m²	5.5 3.0 Arch (4.0) 16.20

Key design parameters for vertical development are shown in Table 16-6. The profile for vertical development is circular. All vertical development will be excavated by raise boring machines.

Table 16-6: Vertical Development Design Parameters

Description	Design Criteria	Units	Value
	Height	m	
Fresh Air Raise Primary	Width (Diameter)	m	4.0
	Profile type	m	Circle
	Profile area	m²	12.57
	Height	m	
Return Air Raise Primary	Width (Diameter)	m	4.5
	Profile type	m	Circle
	Profile area	m²	15.9
	Height	m	
Return Air Raise Secondary	Width (Diameter)	m	3.0
	Profile type	m	Circle
	Profile area	m²	7.07
Fresh Air Raise Secondary	Height	m	
Return Air Raise Tertiary	Width (Diameter)	m	1.5
Escape Raise	Profile type	m	Circle
	Profile area	m²	1.77

Infrastructure development includes lunch room/refuge, explosives and detonator magazines, refuelling and minor service bay.

ANFO type explosives and LP detonators will be used for all development with ANFO loaded using a specific charging vehicle. These are the same products that are being used in the ongoing development of the exploration incline.

16.7 Mine Production Design

The main elements of the stope design for longitudinal stopes are shown in Figure 15-3. Each level of development is separated vertically by 15 m (floor-to-floor). The top level drive is a drill drive for the bottom stope and becomes an extraction drive for the stope above. The stopes will be drilled using down holes, except for stopes located at the top of a lode or where artificial sill pillars are encountered. These will use up-holes to eliminate the requirement for specific drill drive development or because of extraction sequence practicalities.

Due to the variable lode width and geotechnical recommendation, three main stope configurations are required to employ longitudinal extraction. Stopes will be accessed longitudinally (along strike) on each level by, one, two or three strike ore drives dependent on lode thickness. Orebody lode thickness varies orthogonally between 2.0 m (minimum mining width) to 48.0 m. In general, one ore drive is planned when lode thickness is less than 18.6 m. Two ore drives are planned when lode thickness is between 18.6 m and 33.6 m, and three ore drives are planned when lode thickness is greater than 33.6 m. Ore drive spacing is based on a 15 m square grid.

A summary of three stope design configurations are shown in Table 16-7.

Table 16-7: Stope Design Sizes

Stope Configuration	Designation	Criteria
Two ore drives (2x top and 2x bottom)	S1S2	15m H x 23m L (strike) x 2.0m to 18.6m W
Four ore drives (4x top and 4x bottom)	S3S4	15m H x 15m L (strike) x 18.6m to 33.6m W
Six ore drives (6x top and 6x bottom)	S5S6	15m H x 13m L (strike) x >33.6m W

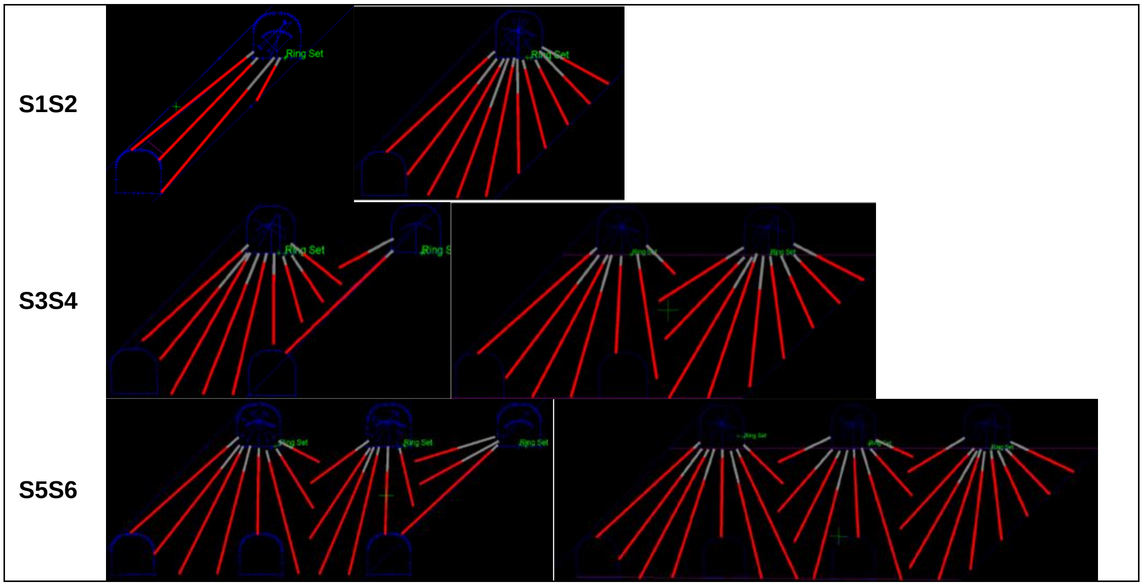
All stope slot raises will be drilled and blasted using a drop raise technique. This requires holes to be drilled in a similar pattern to a development drive drill pattern. A stope slot drive is required on the lower level to provide void for opening the stope slot. This is developed post-filling, with length varying with stope width.

Stope drill and blast parameters for the FS are:

- Recommended drill hole size is 76 mm for narrower stopes and 89 mm for wider stopes.
- Drill factor for narrower stopes is approximately 10 tonnes per drill metre, including slot raise metres; for wider stopes it is approximately 13 tonnes per drill metre, including slot raise metres.
- To assist in the control of dilution and minimize the number of stope blasts, Ammonium Nitrate Emulsion (ANE) type explosives and electronic detonators are recommended for all stopes, with ANE loaded using a specific charging vehicle.
- The average overall stope powder factor, inclusive of the slots and slot raises, for all stope configurations is approximately 0.44 kg/t.

Schematics of typical drill patterns for various stope configurations are shown in Figure 16-5.

Figure 16-5: Schematic of Stope Production Ring Drill Layout

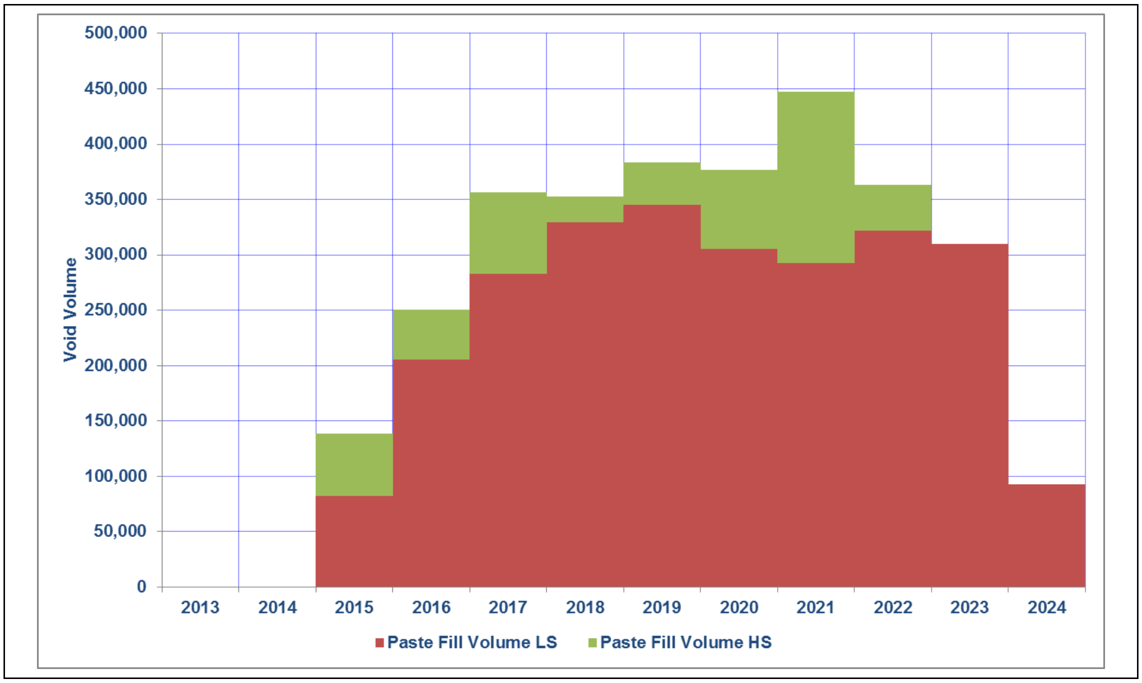


16.8 Backfill

The LHOS mining method and extraction sequence adopted for the Project is reliant on the use of paste fill. Process plant total tailings will be used to produce the paste fill. Approximately 42% 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.

Figure 16-6 shows the annual underground backfill volume requirement profile for the Project.

Figure 16-6: Backfill Volume Requirement per Year[15]



The overall backfill volume requirement, split between low strength and high strength paste fill, is 84% and 16% respectively.

16.8.1 Required Strength

The selected mining method requires stopes to be extracted on a continuous retreat basis. Cemented backfill, placed in retreat stopes, will be exposed by the mining of adjacent stopes, so it must be strong enough to remain stable when the confining rock is removed. The planned turnaround is 14 days; hence backfill should reach the design strength at this curing age.

The design strength of backfill required for vertical wall stability was estimated under the Limit Equilibrium Wedge approach described in Mitchell, Olsen and Smith (1982). The width of the exposed vertical paste fill walls will range from 4 m to 47 m at the nominal height of 15 m. The majority of the stopes (91%) will be exposed on average at 7 m width. Recommended paste fill strengths for the range of stope widths are presented in Table 16-8.

Over 200 paste UCS tests were conducted to investigate the impact of cement type and cement and tailings content on the UCS over a testing period of seven to 182

[15] Years discussed in this section are for illustrative purposes only, as any decision to proceed with mine construction will require regulatory and MKK management approvals.

days. Representative testing was conducted using locally sourced cements, ore tailings and tailings processing water produced during FS process testwork.

Table 16-8: Vertical Exposure Backfill Strength Requirements

Exposure width, m	Bottom 5m lift		Middle 7m lift		Top 2.7m lift	
	UCS, kPa	Cement%	UCS, kPa	Cement%	UCS, kPa	Cement%
4-10	160	2.6	140	2.4	350	3.9
11-20	230	3.2	180	2.8	350	3.9
21-35	280	3.6	210	3.0	350	3.9
36-47	310	3.8	220	3.2	350	3.9

Some backfilled stopes are required to be undercut by the mining of underlying stopes. The expected dimensions of the undercut are 25 m along strike and 8 m (average) to 10 m (maximum) width (span hanging wall to footwall). Stability of backfill sill pillars was assessed based on the analysis of five potential failure mechanisms as described by Mitchell and Roettger (1989). Recommended uniaxial compressive strength (UCS) values of backfill to be undercut are presented in Table 16-9.

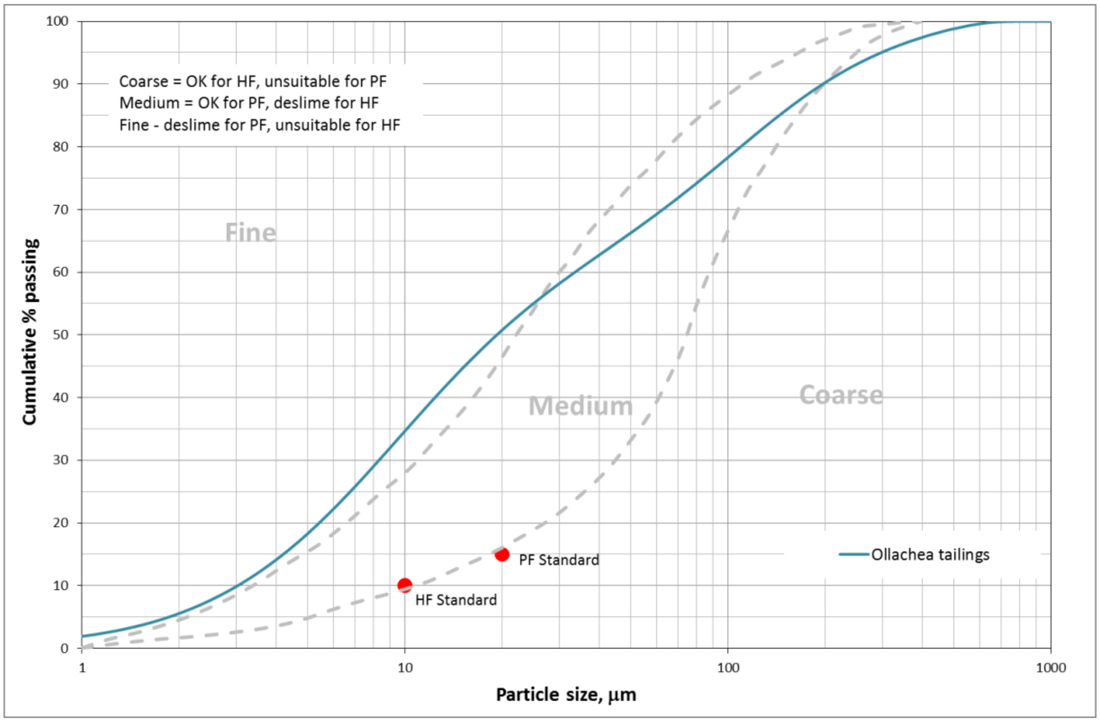
Table 16-9: Undercut Backfill Strength Requirements

Strike Length of Undercut Sill	Width of Undercut Sill	Required UCS, kPa
23m	3m to 12m	400-1550

16.8.2 Backfill Type

Suitability of process plant tailings for production of backfill was determined from the analysis of particle size distribution, mineralogy and rheological properties. It was found that the tailings are rather fine and suitable for paste fill (Figure 16-7).

Figure 16-7: Tailings Particle Size Distribution



Yield stress values of paste fill, determined by the vane method, are presented in Table 16-10.

Table 16-10: Yield Stress

Cement dosage, % Cw of total mix	Tailings, % Cw of total mix	Total solids, % Cw	Yield Stress, kPa
4	66	70	294
10	60	70	462

16.8.3 Mix Design

Paste fill mix designs that satisfy the backfill strength requirements are presented in Table 16-11. UCS requirements and cement dosage rates vary with wall and sill pillar exposure dimensions.

Table 16-11: Paste Fill Mix Design

Exposure	UCS required, kPa	Cement dosage, % Cw of total mix	Cement dosage, % Cw of total solids	Tailings, % Cw	Total solids, % Cw
Vertical wall	160 - 350	2.6 – 3.9	3.7 – 5.6	66.1 – 67.4	70
Undercut	400 - 1550	4.2 – 9.1	6.0 – 13.0	65.8 – 60.9	70

Physical properties of paste fill are presented in Table 16-12.

Table 16-12: Paste Physical Properties

Cement dosage, % Cw of total mix	Total solids, % Cw	Slurry density, kg/m³	Dry density, kg/m³	Cohesion, kPa	Friction Angle, deg.	Young's Modulus, MPa
4	70	1820	1270	80	21	35
10	70	1840				

16.8.4 Paste Fill Reticulation

A reticulation design was determined taking into consideration the geometry of the mine, location of the backfill plant, backfill demand and production rates and rheological characteristics. The outcome of the design is the geometry and specification of the backfill piping requirements and specification of pumping requirements.

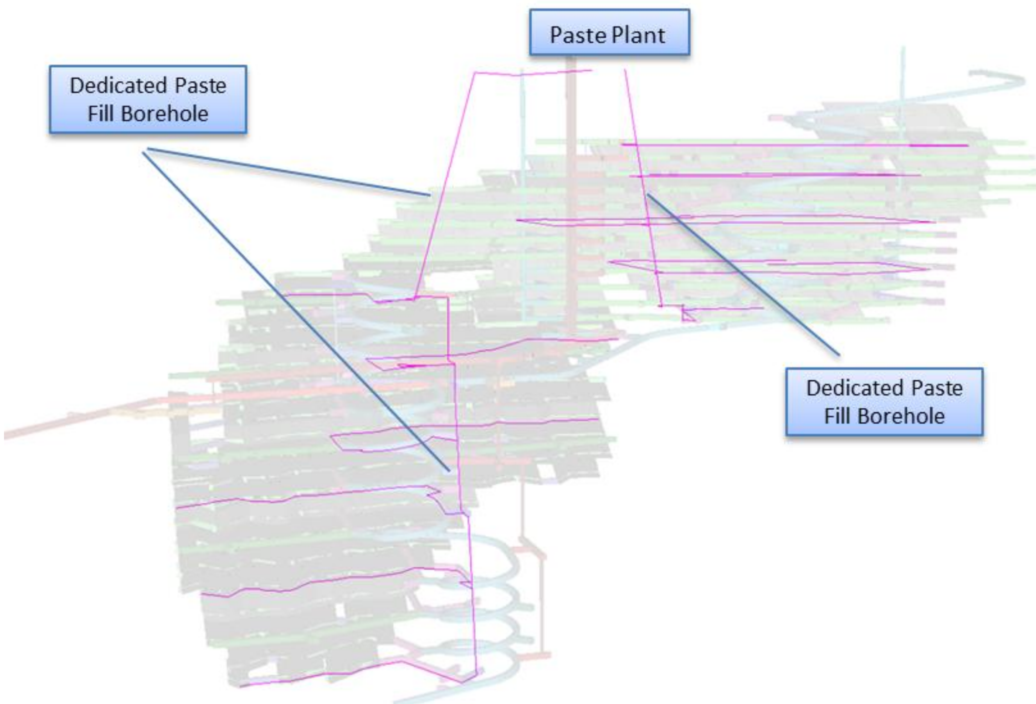
The assessment of rheological properties was undertaken using established laboratory scale tests including a Haake viscotester and a dynamic stress rheometer to determine the yield stress and viscosity of the design mix. Pipe line friction loss was calculated using established empirically derived formulae. Pipe loop testing was not undertaken at this stage due to the limited availability of tailings.

The paste fill plant will be located on the surface at Minapampa above the planned underground workings. Paste will be delivered to the two main production areas via two short horizontal surface runs linking to two sub vertical delivery boreholes. A network of internal boreholes and horizontal pipework will deliver the paste to the required locations within the two main production areas. Gravity alone will not be sufficient to overcome the required yield stress due to extensive horizontal runs along the orebody. A positive displacement pump of 4-2.6 MPa nominal capacity is required to ensure backfill is delivered to the most remote stopes. Reticulation specifications are presented in Table 16-13 and a simple schematic of the paste fill reticulation network is shown in Figure 16-8.

Table 16-13: Paste Fill Reticulation Specifications

Friction Losses	Pump Maximum Rated Pressure	Maximum Operating Pressure in Pipes	Pipe Types	Location
4.65 to 5.20 kPa/m	4.0 MPa	6,500 kPa	150mm Steel Sched 80	High Pressure areas (delivery boreholes, decline and level accesses)
			150mm HDPE PN16	Low pressure areas (level ore drives)

Figure 16-8: Schematic of Paste Fill Reticulation



16.8.5 Fill Retention

Fill retention barricades should have sufficient strength to withstand both earth and pore water pressures, which develop during the placement of paste fill slurry. A staged filling approach (maximum 2 m) of the initial paste plug (6 m) allows the use of a low capacity sling type barricade, which is low cost and easily installed from readily available materials. This method has been adopted for the FS.

16.9 Materials Handling

The strategy adopted for the FS is for all ore and waste material to be loaded using 14 t-capacity load-haul-dumps (LHDs) and transported to dumping areas located outside the two mine portals or internally as waste rock capping for paste filled stopes by dedicated 26.4 t-capacity on-highway tipper trucks.

For both development and stoping, LHDs will load trucks at the nearest stockpile/cuddy. Backs will be stripped as required to allow efficient truck loading. The stockpile/cuddy will be used as a temporary storage area when no trucks are available for direct loading.

Declared Mineral Reserves contain two ore categories (Section 15), ore (+2 g/t Au) and low grade development ore (+1 g/t to 2 g/t). Ore will be transported directly to the process plant ROM pad. Low grade development ore will be transported directly to the process plant ROM pad during the production ramp up period. During full production the majority of the low grade development ore will be transported to a stockpile located at Minapampa. Small quantities of this material will be processed with the ore during full production to ensure stockpile capacity at Minapampa is not exceeded. When ore production begins to decline stockpiled low grade development ore will used as a supplementary feed. This material will be back-loaded to ROM pad using trucks that transport tailings from the process plant to the paste plant that is also located at Minapampa.

Filter cake (tailings) from the process plant will be transported to the paste plant located at Minapampa via the underground mine using dedicated 34.2 t capacity on-highway tipper trucks.

16.10 Ventilation

16.10.1 Primary Ventilation

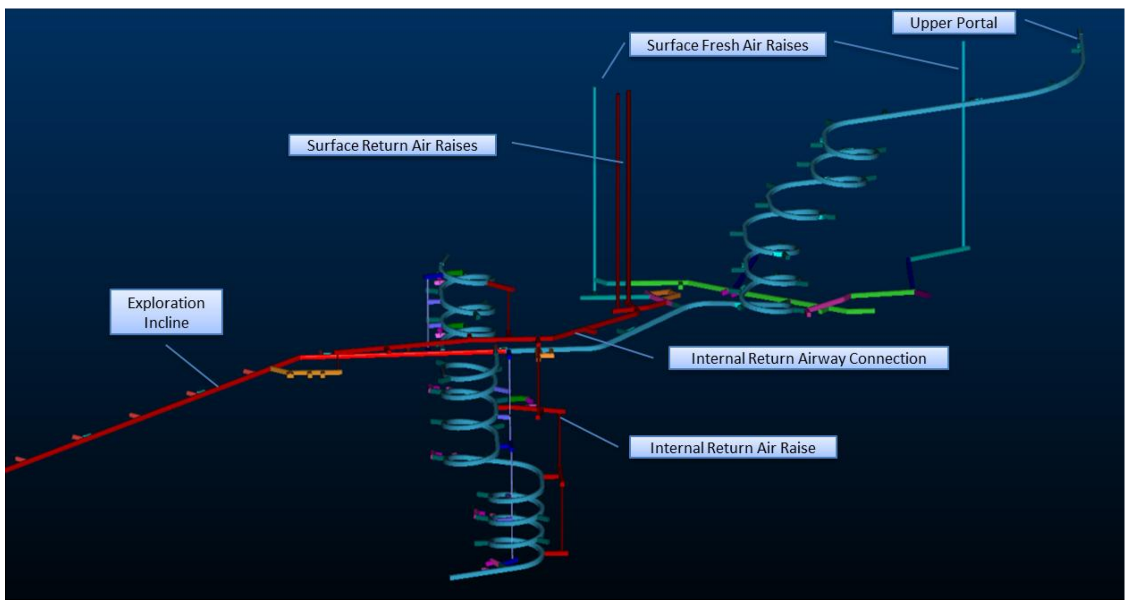
The planned primary ventilation system consists of:

- Two surface intake shafts.
- Two surface return air shafts that will have a single primary fan with a duty of 350 m^3/s.
- Two intake ramps and connected internal ramps.
- An internal return air way system connected to the surface return airway system.

The expected peak flow at full production will be 700 m³/s at a prevailing air density of 0.8kg/m³ (equivalent of 470 m³/s at 1.2kg / m³).

Figure 16-9 shows an isometric view of the major components of the Project primary ventilation system

Figure 16-9: Primary Ventilation System



Ventilation milestone analysis was used to determine the staged primary ventilation requirements for the Project. Maximum ventilation demand for each milestone was estimated by analysing the mine development and production schedule to determine the number of active stopes and development headings in each month. Each milestone was modelled using a mine ventilation simulation software package named VentSim Visual™. Table 16-14 briefly describes each milestone that was identified.

Table 16-14: Primary Ventilation Milestones

Milestone Date	Description of Milestone
April 2014	Maximum development before first surface FAR operational. Lower ramp Q_{TOTAL}=54m^3/s. Upper ramp Q_{TOTAL}=54m^3/s.
July 2014	First surface FAR commissioned. Lower ramp Q_{TOTAL}=100 to 180m^3/s. Upper ramp Q_{TOTAL}=54m^3/s.
October 2014	Both surface RAR's commissioned and main access ramps joined up. Q_{TOTAL}=250m^3/s.
March 2016	Eastern lower exhaust connected up; Second surface FAR operational (for some time); first internal RAR leg above and below ramp in the eastern part of the mine operational. Eastern lower ramp in progress. Q_{TOTAL}=700m^3/s.
May 2020	Mining is at its extremities and at maximum production rate Q_{TOTAL}=700m^3/s.

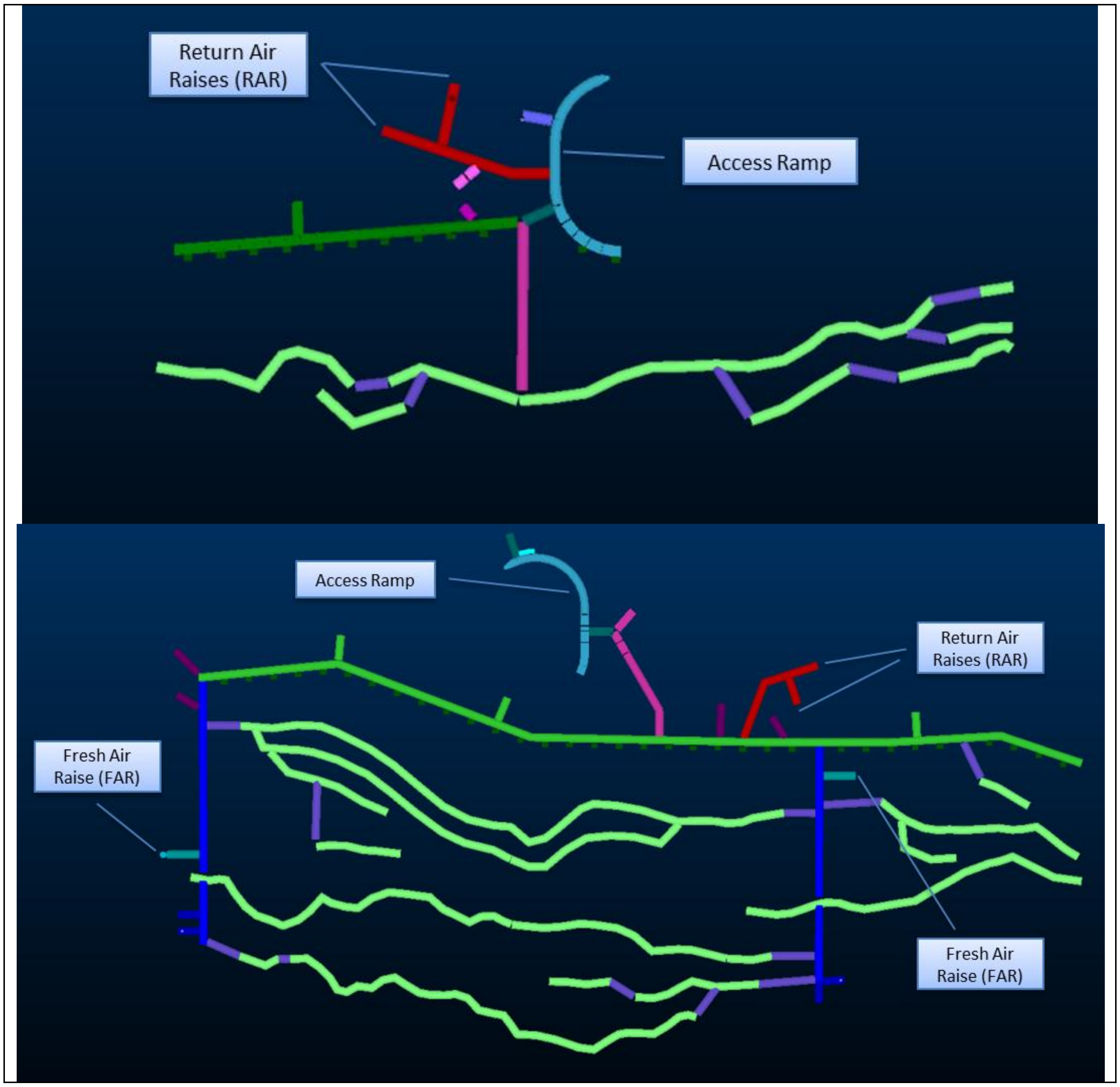
16.10.2 Secondary Ventilation

The mine has three general layouts for secondary ventilation circuits during planned operations:

- A long-range configuration for development designed to establish or extend the primary ventilation circuit.
- The levels of the eastern part of the mine where the secondary fan is located in the fresh air decline and ducting is run into the level with branches to each heading or stoping area.
- The levels of the western part of the mine where secondary fans are located in walls in drives that connect directly to the two primary surface fresh air raises. Ducting is run from these fans branching off where required into drives and stoping areas.

Typical second and third layouts are shown schematically in Figure 16-10.

Figure 16-10: Secondary Ventilation System



16.10.2.1. Emergency Egress and Entrapment

Access to the mine will be via two portals. The two portals will be connected via a single primary incline/decline. This will form the main egress system. The lower portal is located close to the processing plant and administration buildings and will be the main access to and from the planned underground mine. The upper portal will be used to provide access to the paste plant and shotcrete batch plant located at Minapampa.

The eastern part of the mine will be serviced by dedicated escape raises located off the main incline and decline. The majority of the western part of the mine will also be serviced by dedicated escape raises. These are located on each level at the extremities of each of the stope access crosscuts (two per level). These will join as the mine is developed to form two independent escape routes down the footwall of the western part of the mine.

In addition, self-contained refuge chambers of suitable size will be used and placed in locations where a second means of egress has not been established or where a second means of egress is available but not supplied with fresh (safe) air. This will ensure no person working underground will be at risk from rock fall entrapment or fire.

16.11 Sequence and Schedule

16.11.1 Mine Development

The mine development strategy employed for the FS is as follows:

- Contract to complete the exploration incline is extended for approximately ten months. The strategy assumes development is continuous and the necessary permits are granted in a timely manner.

- Expedite the development of the primary mine accesses, grade control diamond drilling platforms and primary ventilation system to minimize the production ramp up period and provide a second means of egress.

- Production will start on 2775 mRL in the eastern part of the mine based on the location of diamond drilling platforms. In the western part of the mine, production will start on 2805 mRL to establish the bottom-up mining method and maximize ore extraction from the area.

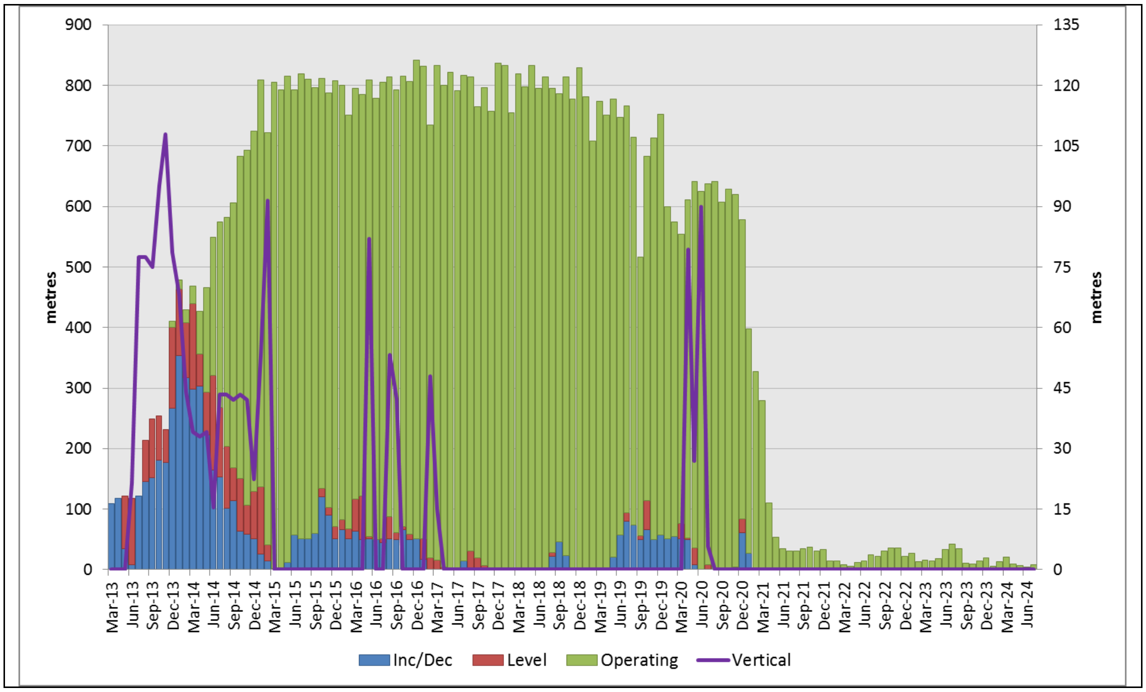
Table 16-15 shows the base jumbo development rates that were used to schedule lateral development activities.

Table 16-15: Jumbo Development Advance Rates

Jumbo Development	Scheduled Work Rate (m/month)
Single heading maximum per jumbo	120
Multiple heading maximum per jumbo	250

Figure 16-11 shows the average lateral development metres per month split by development type. The number of metres per month is approximately 800, which is equivalent to requiring four jumbo crews per shift for a period of four and half years.

Figure 16-11: Lateral Development Metres per Month



The vast majority of mine development is scheduled to be completed by the end of 2020 with production scheduled to extend until 2024. Mine development is completed early due to the requirement to split the western part of the mine into two producing areas towards the end of the mine life. This requirement reduces the impact of the life of mine production tail.

Table 16-16 shows the development rates that were used to schedule vertical development activities. Geotechnical assessment has recommended that all raise bore holes greater than 3 m be supported to provide life of mine integrity. The four surface raise bore holes that form a major part of the primary ventilation system have been assumed to require ground support over 75% of their length.

Stope slot raises are not explicitly defined. The timing to develop these was incorporated in an all-inclusive stope turnaround time.

Table 16-16: Vertical Development Advance Rates

Development Type	Dimensions (m)	Scheduled Work Rate (drill m/day)	Scheduled Work Rate (support m/day)
Raisebore (surface)	4.5 dia.	2.10	2.56
Raisebore (surface)	4.0 dia.	2.78	2.75
Raisebore (UG)	3.0 dia.	3.00	
Raisebore (UG)	1.5 dia.	2.50 (short) 3.00 (long)	

The majority of vertical development is associated with the primary ventilation system and this is completed before full production.

16.12 Mine Production

The production strategy employed for the FS is as follows:

The mine extraction sequence adopted is bottom up, i.e. lower stopes are extracted first and then filled. The filled stopes are then used as the platform to extract the next stopes in the upward sequence;

- in general, a mining level is completed before the next mining level is started;
- for the eastern part of the mine, which contains a significantly smaller production tonnage than the western part of the mine, the bottom up method has been modified to improve productivity and allow consistent parallel production with the western part. Four mining levels have been grouped together as a mining panel and this is mined with a bottom up sequence. This essentially allows multiple areas to be mined simultaneously. It does however create the requirement for artificial sill pillars (stopes that have been previously mined and filled) that will require additional engineered support to allow safe extraction of the stopes located directly below the pillar. These artificial pillars will be filled with high strength paste fill.
- To minimize the life of mine production tail the western part of the mine has been split into two mining areas. The sill pillar level is located on 2940 mRL.

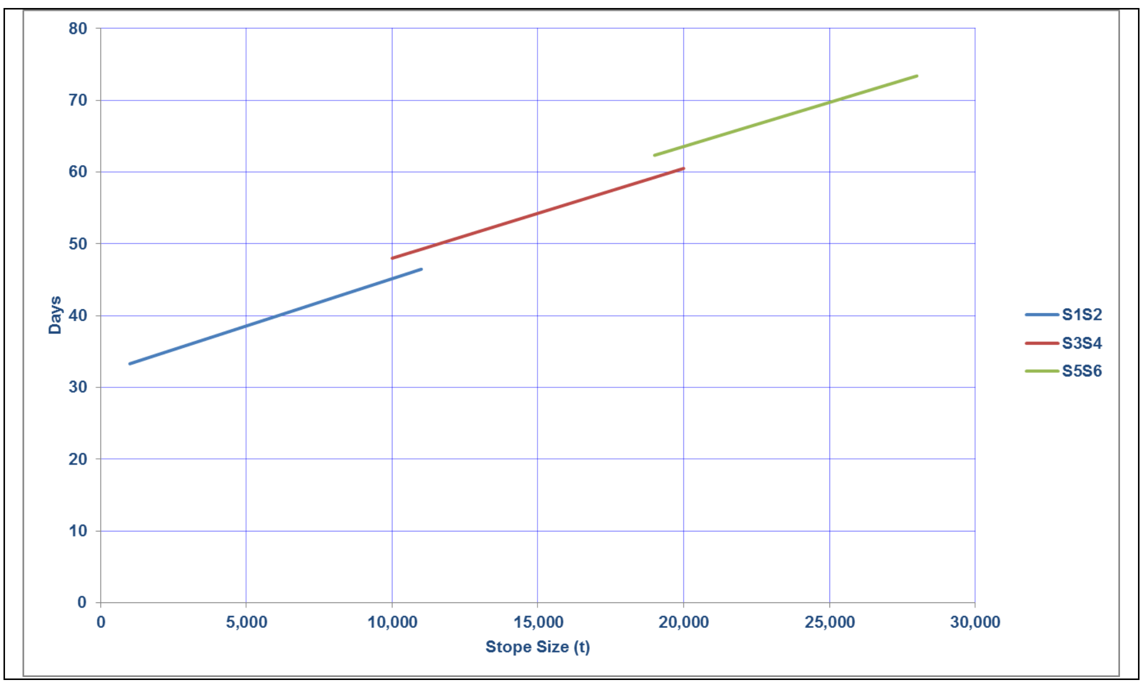
For the FS, an all-inclusive stope production duration was estimated based on fixed and variable stope activity rates. These were then compiled into a simple stope tonnage algorithm that was used to determine the all-inclusive extraction, fill and curing duration for each individual stope. Three algorithms were determined for each of the three stope configurations. Table 16-17 shows the assumptions that were used to derive the three stope duration algorithms.

Table 16-17: Stope Production Rates

Stoping Activity	S1S2 2m to 18.6m Fixed (day)	S1S2 2m to 18.6m Variable (t/day)	S3S4 18.6m to 33.6m Fixed (day)	S3S4 18.6m to 33.6m Variable (t/day)	S5S6 >33.6m Fixed (day)	S5S6 >33.6m Variable (t/day)
Cable Bolt and Grout	Not considered part of the stope cycle. Assumed to be completed pre stoping.					
Slot Drive (ave.)	1.0		4.0		6.0	
Slot Raise	7.0		6.0		6.0	
Production Drilling		2,565		3,098		3,339
Blasting/Charging	2.0	26,889	3.0	25,833	4.0	25,273
Loading		1,761		1,761		1,761
Barricades	5.0		5.5		6.0	
Fill		3,107		3,107		3,107
Curing	14.0		14.0		14.0	
Floor Base	Not considered part of the stope cycle. Completed post stoping on a level.					
Delays	3.0		3.0		3.0	
Totals	**32.0**	**0.00132d/t**	**35.5**	**0.00125d/t**	**39**	**0.00123d/t**
Duration Algorithm	0.0132*t+32		0.00125*t+35.5		0.00123*t+39	

Figure 16-12 shows the graphical form of the stope duration algorithms.

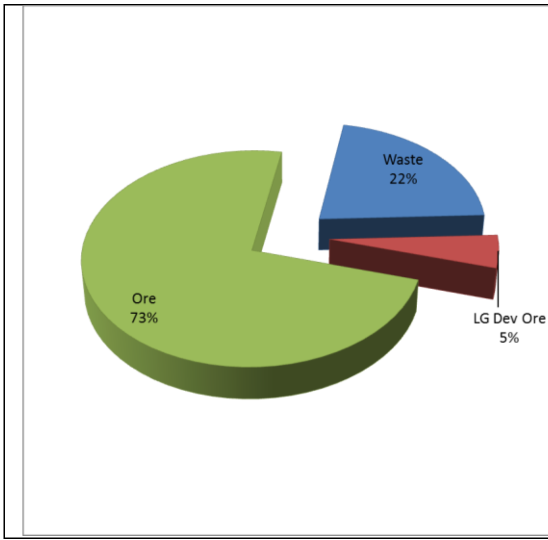
Figure 16-12: Stope Productivity Estimate



16.13 Mine Physicals Summary

FS LOM underground mining physicals are summarised in Table 16-18 and Figure 16-13 shows LOM percentage tonnage splits.

Figure 16-13: LOM Tonnage



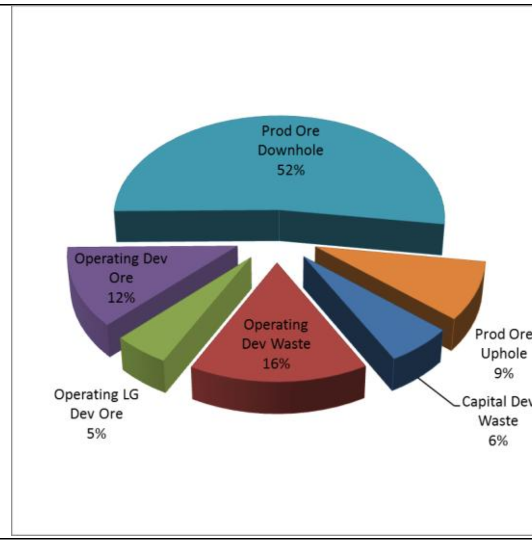


Table 16-18: Mining Physicals Summary

Physical		Units	LOM	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024
Lateral Dev.	Capital dev.	m	8,019	1,939	3,308	718	875	163	97	524	367	28			
	Operating dev.	m	56,987	10	3,374	8,851	8,723	9,435	9,551	8,160	6,953	1,372	240	252	66
Vertical Dev.	Capital dev.	m	1,586	533	465	145	178	63			202		1,586	533	
	Operating dev.	m	220			30	20	50	30	40	50		220		
Production	Total Mined	Mt	11.9	0.2	0.5	1.0	1.3	1.5	1.5	1.5	1.4	1.1	0.9	0.7	0.2
	Waste	Mt	2.6	0.2	0.5	0.3	0.4	0.4	0.3	0.3	0.2	0.0			
	Ore	Mt	8.7		0.0	0.6	0.8	1.1	1.1	1.1	1.1	1.1	0.9	0.7	0.2
	Contained Gold	koz	983		3	65	80	123	129	128	136	127	90	83	19
	Gold Grade	g/t	3.5		3.1	3.5	3.1	3.5	3.7	3.6	3.8	3.6	3.2	3.5	2.9
	Low Grade Dev. Ore	Mt	0.6		0.0	0.1	0.1	0.1	0.1	0.1	0.1	0.0			
	Contained Gold	koz	28		1	5	4	3	5	4	6	1			
	Gold Grade	g/t	1.5		1.6	1.5	1.5	1.5	1.4	1.5	1.5	1.5			
	Cable drill	kdm	662	2	7	35	55	78	77	81	74	89	73	69	22
	Production drill	kdm	697			32	58	83	80	88	84	101	81	70	19
	Haulage	Mtkm	30.1	0.2	0.9	1.8	2.7	3.6	3.8	4.1	4.0	3.4	2.7	2.3	0.6
	Backfill Void (Paste)	km³	3,012			136	246	349	346	377	371	442	356	306	84

Totals may not sum due to rounding

16.14 Mine Waste Balance

Waste material generated from the development of the exploration incline and pre-production waste development will be hauled to the lower portal and dumped on the site of a permanent waste dump that will be located in close vicinity to this portal.

Waste rock from the development of the upper portal and decline prior to decline/incline breakthrough will be deposited temporarily on an upper waste dump then backhauled during the LOM to the waste dump located close to the lower portal. All other waste rock generated throughout the mine life will be hauled to the lower portal waste dump, used to cap paste filled stopes or hauled to the TSF for co-deposition.

The LOM of waste balance is outlined in Table 16-19.

Table 16-19: LOM Waste Balance

Description		Tonnage (Mt)
Lower portal waste dump design capacity		+2.45
	UG waste development	*-2.67*
	Paste stope waste capping	*+0.11*
	Road base (non-permanent haulageways)	*+0.03*
Lower portal waste dump final capacity		-0.08
Tailings storage facility (TSF) design capacity		+5.85
	Total tailings	*-9.34*
	Tailings to paste	*+3.66*
	UG waste to TSF	-0.08
Tailings storage facility (TSF) final capacity		+0.09

16.15 Mine Equipment

The Project will require a standard, medium scale, underground mobile production fleet of jumbos, LHDs, trucks and drills. The primary, direct and indirect equipment used as a part of the basis to design the underground mine is shown in Table 16-20.

Table 16-20: Primary, Direct and Indirect Underground Mobile Equipment

Generic Description	Type or Size
Primary	
Development jumbo	Twin boom electro-hydraulic
Underground loaders	14 t for development and production (tele-remote)
Underground trucks	25 t (6x4) on-highway tipper trucks (ore and waste)
Underground trucks	34 t (8x4) on-highway tipper trucks (tailings)
Production drill rig	Top hammer (76mm and 89mm)
Cablebolt rig	Dedicated cablebolt rig (drill (64mm) and install)
Direct	
Scissor Lift	4wd UG specification
Charge-up vehicle	4wd dedicated UG charge up vehicle (dev. and production)
Shotcrete sprayer	4wd UG specification
Shotcrete transmixer (carrier)	4wd UG specification
Indirect	
Grader	6wd UG specification
Maintenance/fuel truck	4wd UG specification
Backfill services loader/IT	4wd UG specification
Flat bed truck (materials)	2wd UG specification
Light vehicles	4wd UG specification

All mobile and fixed plant equipment will be purchased, operated and maintained by MKK.

MKK will be responsible for the routine maintenance of all their mobile plant and fixed plant. Major services are to be completed on surface in a dedicated workshop with minor servicing to be completed in a basic underground service bay.

The fleet of primary mobile equipment units was calculated directly from equipment productivity rates and scheduled mine physicals from the final mine design. Direct and indirect was estimated based on Coffey Mining experience and scheduled mine physicals.

Project sustaining capital for equipment replacement has been estimated based on industry standards and original equipment manufacturers (OEM) recommendations.

Table 16-21 shows the annualised LOM mobile equipment schedule for primary, direct and indirect mobile equipment.

Table 16-21: LOM Mobile Equipment Schedule

Equipment Description		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024
Primary	Development jumbo	1	3	4	4	4	4	4	3	1	1	1	1
	Underground loaders	1	2	3	4	4	4	4	4	4	3	3	2
	Underground trucks (ore, waste)	1	2	3	4	6	6	6	6	5	4	4	1
	Underground trucks (tailings)			2	2	3	3	4	4	4	3	3	1
	Production drill rig			1	1	1	1	1	1	1	1	1	1
	Cablebolt rig	1	1	1	1	2	2	2	2	2	1	1	1
Direct	Scissor Lift	1	2	2	2	2	2	2	2	1	1	1	
	Charge-up vehicle	1	1	2	2	2	2	2	2	2	2	1	1
	Shotcrete sprayer	1	1	1	1	1	1	1	1	1	1	1	
	Shotcrete transmixer (carrier)	1	1	2	2	2	2	2	2	2	2	2	
Indirect	Grader		1	1	1	1	1	1	1	1	1	1	
	Flat bed truck (materials)	1	1	1	1	1	1	1	1	1	1	1	
	Maintenance/fuel truck		1	1	1	1	1	1	1	1	1	1	
	Backfill services loader/IT			1	1	1	1	1	1	1	1	1	1
	Light vehicles	5	13	15	16	16	16	16	15	12	12	10	4

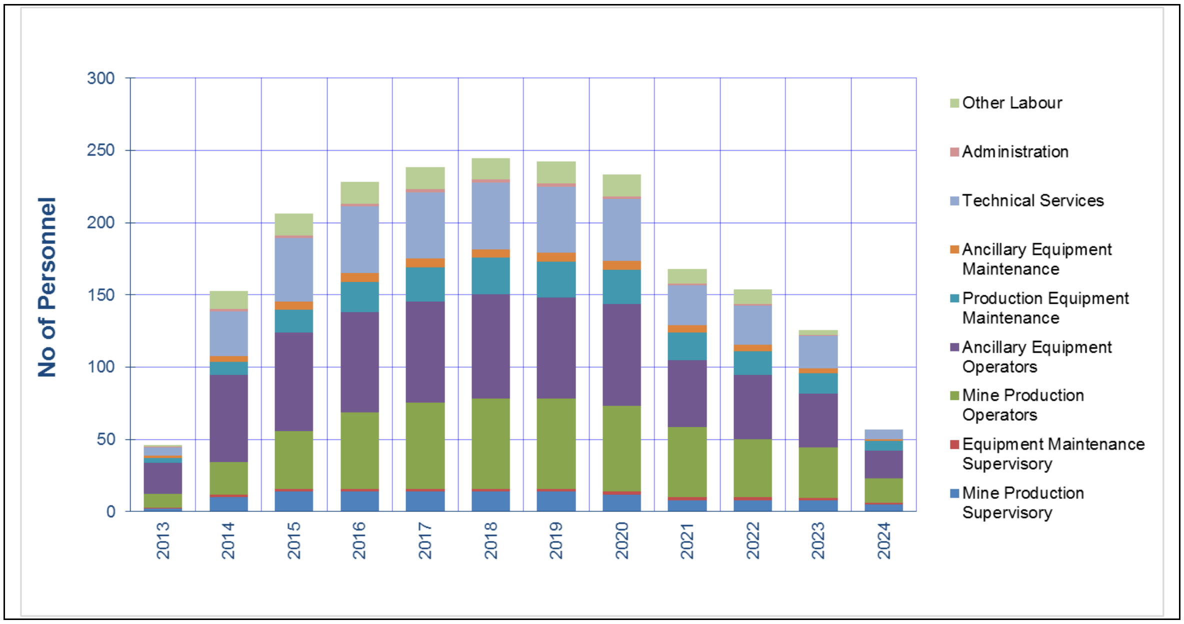
16.16 Organisation and Structure

The mine is planned to be owner operated. Specialist contractors would be used for specialised activities such as raise boring.

The mine is planned to operate 24 hours per day, 365 days per year and mine operators will work a 14 days on, 7 days off roster. Shifts will be of 12 hours duration.

Figure 16-21 shows the estimated mine workforce complement for the life of mine. This includes operators, supervisors, technical and mining management.

Figure 16-14: LOM Mining Workforce Complement



16.17 Underground Infrastructure and Services

MKK will install, operate and maintain all underground infrastructure and services.

Underground infrastructure and services proposed for the mine include:

16.17.1.1. Controls and Communication System

- Leaky feeder system and telephone network.

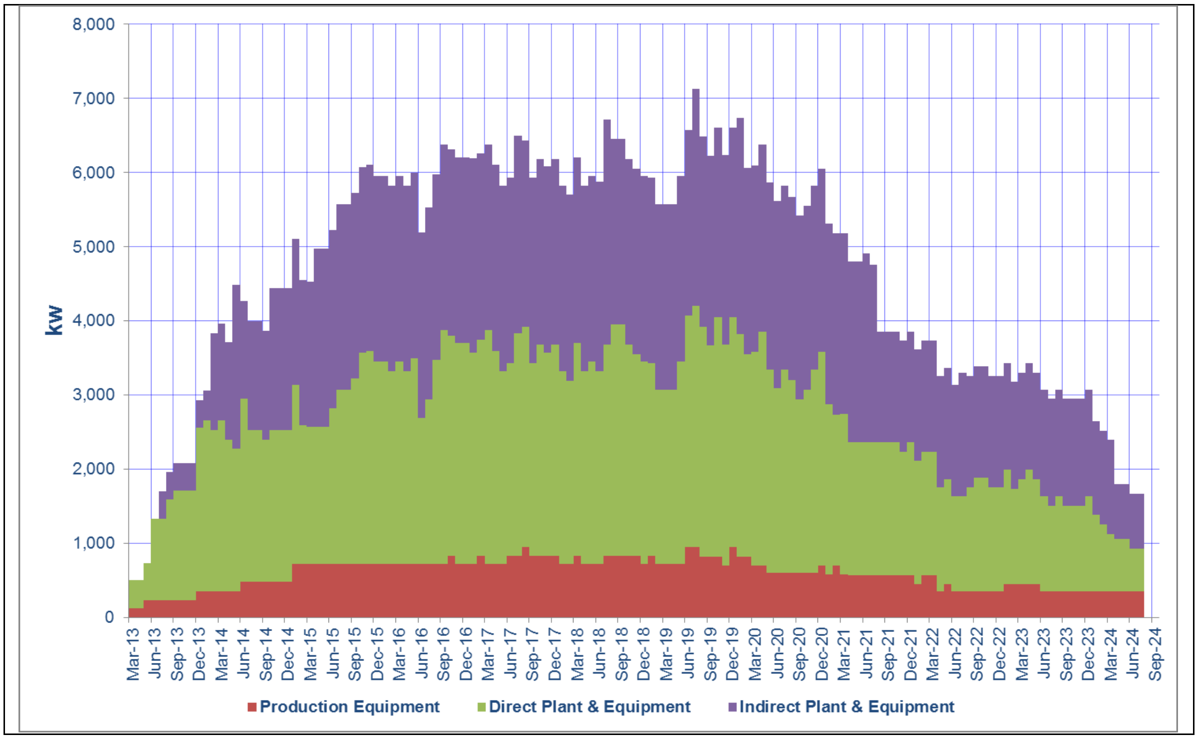
16.17.1.2. Underground Power

- A 13.8 kV HV cable running from the lower portal to the upper portal and surface fans via access incline/decline and dedicated service holes.

- 460 kV step down transformers strategically located to efficiently distribute and supply local equipment demand and numerous distribution boxes, jumbo, pump and fan starter boxes.

A LOM installed power estimate for the underground mine is shown in Figure 16-15.

Figure 16-15: LOM Underground Installed Power



16.17.1.3. Compressed Air

- Compressor to be located on surface close to the lower portal with a distribution network to fixed installations and main level accesses.

16.17.1.4. Potable Water

- To be supplied in bottles and available in the underground lunch room.

16.17.1.5. Service Water

- Service water to be supplied from a water tank located above the planned underground workings at Minapampa. This will also supply the shotcrete batchplant and paste plant.

- Estimated peak demand is approximately 8.5 L/s.

- Distribution network from the surface tank to underground workings.

16.17.1.6. Mine Dewatering

The majority of the Ollachea mine will utilise a gravity-fed dewatering system, while dewatering of the eastern part of the mine, located below the primary incline access, will be undertaken by a combination of submersible and progressing cavity pumps. Settlement sumps will serve to remove excess sediment and oil contaminants prior to discharge to surface. Gravity will be used where possible to feed polypipe reticulation, which will run the length of the exploration incline to the lower portal where it will be directed to a water treatment plant for processing.

Mine dewatering requirements are based on 3D hydrogeological modelling completed as part of the FS.

The main water sources for the mine are:

- ground water (modelled), early peak of approximately 44 L/s that drops to a LOM average of 33 L/s;
- service water, average 5.1 L/s, maximum 8.4 L/s; and
- water for flushing paste backfill distribution line (30 m^3 every second day).

The mine dewatering system consists of:

- Progressive cavity pumps for the lower decline area in the eastern part of the mine, piggy backed where and when required;
- Face and sump pumps for development headings and production areas; and
- Drainage network including sumps, pipes and drain holes.

16.17.1.7. Underground Lunch Room

Underground workers will use a dedicated meal room for mid-shift breaks.

16.17.1.8. Refuelling and Service Bay

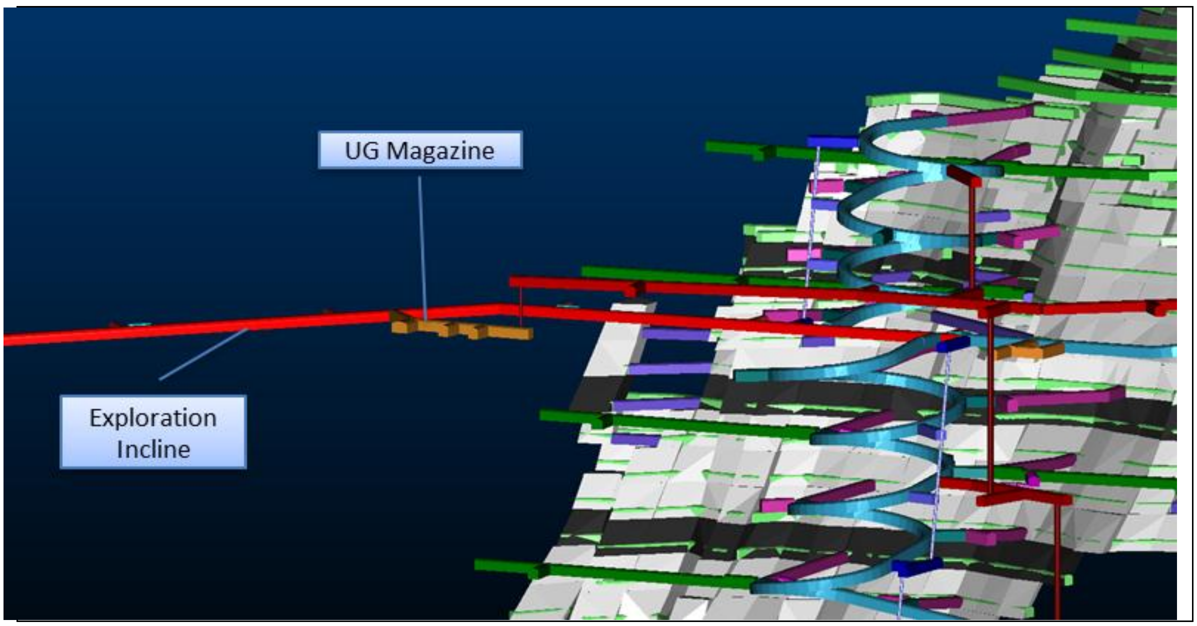
Underground drilling equipment, jumbos and production drills are to be refuelled by a mobile fuel truck. All other mobile equipment used underground will be refuelled on surface.

Minor servicing of primary underground equipment will be completed in a dedicated underground service bay. Major servicing and repairs will be carried out at the mine workshop which will be constructed near the portal.

16.17.1.9. Explosives Magazine

Explosives will be stored in a dedicated underground magazine located off the primary mine access incline and away from active areas. The magazine will be connected directly to the primary return airway system.

Figure 16-16: UG Magazine Location



17 RECOVERY METHODS

The Ollachea mineral processing plant will include circuits for crushing, grinding and classification, batch gravity concentration of cyclone underflow for gravity recoverable gold and continuous gravity concentration of cyclone overflow. Continuous gravity concentrates will be leached in a dedicated CIL circuit. Tailings will recombine with concentrate and be processed in a separate CIL circuit. Gold recovery from CIL solutions will be by carbon elution, electrowinning and refining to produce doré on site. Tailings will be treated by the Air/SO_2 process for cyanide detoxification, followed by iron precipitation by zinc sulphate addition, then thickened and filtered to produce a filter cake for disposal at a dry-stack tailings storage facility (TSF) or for use as a paste backfill. The plant will further incorporate water treatment, carbon regeneration, reagent preparation, oxygen generation and supply, compressed air and water services.

17.1 Plant Design Criteria

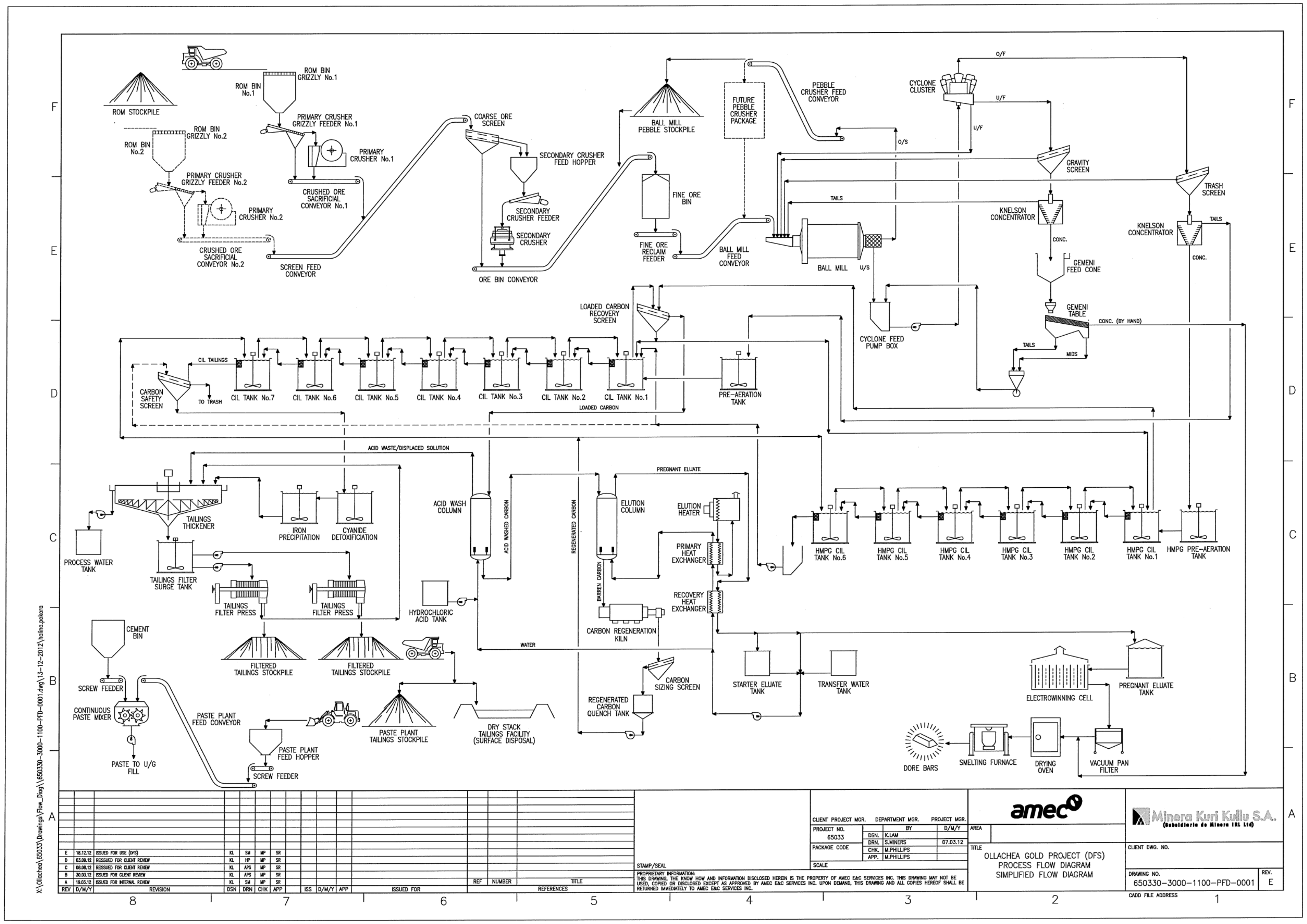
The design parameters of the processing plant are:

• Plant throughput:	1.1 Mt/y, or 137.5 t/h
• Plant availability:	91.3% or 8,000 hours per year
• ROM feed size:	F100 600 mm, F80 270 mm
• Final product grind:	P80 of 106 µm
• Design head grade:	3.65 g/t Au
• Head grade (LOM Average):	3.37 g/t Au
• Residue grade (LOM Average):	0.30 g/t Au
• Overall recovery (LOM Average):	91.1%
• HMPG CIL residence time:	24 h
• CIL residence time:	36 h
• Final tailing cyanide destruction:	SO2/Air/Cu^{2+} Catalyst + $ZnSO_4$
•	

17.2 Plant Design

A simplified process flow diagram is provided in Figure 17-1.

Figure 17-1– Simplified Process Flow Diagram



17.2.1 Crushing

Ore coming from the mine will have a maximum size of 600 mm and will be transported by road trucks of 26.4 t capacity. The trucks will dump ore directly to a fixed grizzly, with 600 mm openings, located on top of a ROM ore bin (120 t). From the ROM bin, ore will be processed by a mobile crushing plant, which incorporates the vibrating grizzly feeder and the crusher. Ore will be withdrawn from the ROM bin by a vibrating grizzly feeder. Oversize from the grizzly feeder will feed the primary jaw crusher (1,000 mm x 760 mm). As necessary, ore can be dumped to a ROM stockpile located at the primary crusher ROM bin. Stockpiled ore can be reclaimed by a front–end loader (15 t) to the ROM bin. In the event that the delivery of ore from the mining operation to the primary crushing plant is interrupted, the stockpile live capacity (6,000 t) will provide sufficient ore for 24 h of continuous operation of the entire downstream process. Provision has been made for the installation of a second primary crushing mobile unit for the ramp up in crushing circuit throughput.

The primary crushed material (P_{80} of 80 mm) will be directed to a double deck screen. The oversize from both the top deck and lower deck will discharge to the secondary crusher (Vibrocone). A series of tramp metal magnets and metal detectors will protect the secondary crusher from tramp metal ingress. The discharge from the secondary crusher will recombine with the undersize from the screen. The final crushing circuit product will have a product size P_{80} of approximately 10 mm and will be conveyed to the grinding circuit fine ore storage bin (1,100 t).

Crushed ore will be reclaimed from the fine ore bin, via a single variable speed belt feeder (2,000 mm x 10,000 mm). Reclaimed ore will discharge to the ball mill feed conveyor, which will convey ore to the ball mill located within the grinding circuit.

Dust emission within the crushing circuit will be controlled by a series of dust collection systems. Dust collected from the various transfer points, screening circuit and crushing circuits will be returned to the crushing/grinding circuit.

17.2.2 Grinding and Gravity Recovery

A single stage ball mill (5.3 x 7.3 m, drive 3.45 MW), operating in closed circuit with a cyclone cluster (10 duty/2 standby x 250 mm), will be utilised to grind the ore from a feed F_{80} of 10 mm (F_{100} of 15 mm) to a P_{80} of 106 µm. The mill will be run at a fixed speed.

The ball mill feed will constitute crushed ore, return cyclone underflow slurry, process water and kerosene; addition of the latter is required to inhibit the adsorptive capacity of gold by carbonaceous material present within the ore. Mill discharge will pass through the ball mill trommel screen, with undersize being pumped to the cyclone cluster.

Oversize (pebbles) from the trommel screen will be conveyed to a pebble stockpile. Pebbles will be recovered from the stockpile, by front end loader, and returned to the ball mill feed conveyor, via the emergency reclaim facility.

A portion of the cyclone underflow (25%) will be diverted to the gravity recovery circuit with the remainder returning directly to the ball mill. Cyclone underflow, feeding the gravity circuit, will pass over the gravity scalping screen (1,800 mm x 3,600 mm) which will remove the +2 mm material. Oversize material from the screen will return to the ball mill feed hopper. Gravity screen undersize will constitute the feed to a centrifugal Knelson concentrator (1,200 mm) for recovery of free gold. Gravity tailings will be directed back to the ball mill feed hopper. Free gold concentrate generated by the Knelson concentrator will be discharged into a storage hopper from where it will be upgraded utilising a Gemeni table. The Gemeni table middlings and tailings streams will be returned to the ball mill feed box. Gemeni table gold concentrate will be transported to the gold room for drying and direct smelting to doré. The Gemeni table will be located in a secure area.

Overflow from the cyclone cluster (P_{80} of 106 µm) will gravitate to the CIL circuit feed trash screen (1,200 mm x 3,600 mm, 0.6 mm aperture). The trash screen will remove entrained trash, including woodchips, fibre and plastics, which will report to screen oversize and be retained in a bin for disposal. Cyclone overflow, and ultimately CIL feed will be indirectly controlled at 42% solids by directly controlling the cyclone feed density at 64% solids.

Trash screen undersize will be subjected to a continuous Knelson concentrating step, for the recovery of sulphide associated gold. Concentrate from the continuous Knelson concentrator will constitute the feed to the high mass pull gravity (HMPG) CIL circuit pre-aeration tank. Tails from the continuous Knelson will constitute the feed to the mainstream CIL circuit pre-aeration tank.

Pre-aeration is required to reduce the soluble iron levels within the slurry, by oxidation of the iron species, thus preventing the formation of a hexa-ferrocyanide complex which would adversely affect the cyanide detoxification process. Pre-aeration will also contribute to a reduction in cyanide consumption through the passivation of cyanicides.

Balls will be delivered in bags (2,000 kg) and will be loaded to the ball mill via a hoist and ball loading hopper. A sump pump will be installed within the grinding area to collect spillage from the ball milling and gravity circuit. Spillage collected in this area will be returned to the ball mill discharge hopper.

17.2.3 High Mass Pull Gravity Carbon in Leach

The HMPG CIL circuit will comprise six tanks, with a total leach capacity of 480 m^3, equating to a total residence time of 24 hours at a nominal mass pull to concentrate of 7 tph solids. The HMPG CIL circuit will be fed from the HMPG pre-aeration tank overflow.

The dimensions of the individual HMPG CIL tanks will be 4.5 m x 5.0 m, equating to an individual tank volume of 80 m^3, each with a residence time of 4 h. The HMPG CIL tanks will be mechanically agitated. To aid with the dissolution of precious metals, oxygen will be added to the slurry via the agitator shaft.

Within the HMPG CIL circuit, slurry will be introduced to the first HMPG CIL tank (No. 1) and fresh carbon will be introduced into the last HMPG CIL tank (No. 6). The HMPG CIL tanks will all be fitted with mechanically swept, carbon inter-stage screens and airlifts for inter-tank carbon transfer (exception of the first HMPG CIL)

The slurry will gravitate from HMPG CIL Tank No. 1 to the consecutive CIL tanks via the inter–stage screens (0.5 m^2, 0.8 mm aperture) until being discharged from HMPG CIL Tank No. 6 to the HMPG CIL tails hopper, from which the slurry will be pumped to the distributor box of the main CIL circuit.

To maintain the desired carbon concentration within the respective HMPG CIL tanks, carbon will be advanced from any CIL tank to the adjacent upstream CIL tank, counter–current to the slurry flow, by the carbon transfer air lifts. This carbon transfer will be conducted daily over an 18 h period.

Slurry containing loaded carbon will be pumped intermittently (approximately four hours each day) from the first live HMPG CIL tank to the common loaded carbon recovery screen. Screen undersize will be returned to the respective HMPG CIL tank and the washed “loaded” carbon will discharge directly into the acid wash column.

Quenched regenerated carbon will be returned to the HMPG CIL section from the regeneration carbon quench tank. The carbon regeneration kiln discharges directly to the carbon sizing screen (1800 mm x 900 mm, 0.8 mm aperture), with screen oversize reporting to the quench tank. Fresh carbon will also be introduced to this circuit, via the carbon quench tank.

The HMPG CIL area will be serviced by two sump pumps delivering spillage back to the HMPG CIL feed distribution box.

17.2.4 Carbon in Leach (CIL)

The CIL circuit will comprise seven tanks, with a total leach capacity of 9,450 m³, equating to a total residence time of 36 hours at 137.5 tph solids. The CIL circuit will be fed from the CIL pre-aeration tank overflow.

The dimensions of the individual CIL tanks will be 12.0 m x 14.0 m, equating to an individual tank live volume of 1,526 m³, each with a residence time of 5.5 h. The CIL tanks will be mechanically agitated. To aid with the dissolution of precious metals, oxygen will be added to the slurry via the agitator shaft.

Within the CIL circuit, slurry will be introduced to the first CIL tank (No. 1) and fresh carbon will be introduced into the last CIL tank (No. 7). The CIL tanks will all be fitted with mechanically swept, carbon inter-stage screens and carbon transfer pumps for inter-tank carbon transfer.

The slurry will gravitate from CIL Tank No. 1 to the consecutive CIL tanks via the inter–stage screens (4 m², 0.8 mm aperture) until being discharged from CIL Tank No. 7 to the carbon safety screen. The CIL tailings slurry will be screened by the carbon safety screen (1,800 mm x 3,600 mm, 1 mm aperture) to recover misreporting carbon, which will drain to a carbon fines bin.

To maintain the desired carbon concentration within the respective CIL tanks, carbon will be advanced from any CIL tank to the adjacent upstream CIL tank, counter–current to the slurry flow, by the carbon transfer pumps. This carbon transfer will be conducted daily over an 18 h period.

The HMPG CIL and CIL circuits will share the elution and regeneration facilities.

Slurry containing loaded carbon will be pumped intermittently (approximately four hours each day) from the first live CIL tank, by the respective loaded carbon recovery pump, to the common loaded carbon recovery screen (1,200 mm x 3,600 mm, 0.8 mm aperture). Screen undersize will be returned to the respective CIL tanks and the washed "loaded" carbon will discharge directly into the acid wash column.

The pH of the slurry is a measure of its acidity/alkalinity. At a pH below 9.3, the cyanide will begin to decompose, with the following detrimental consequences:

- Insufficient cyanide will be available to dissolve gold, resulting in increased loss of gold to the tailings, elevated cyanide requirements and thus an increase in operating costs.
- Formation of highly toxic hydrogen–cyanide gas, posing a health threat to workers.
- As the pH drops further, the slurry becomes corrosive, resulting in equipment damage.

Consequently, it is necessary to add milk of lime to the process in order to maintain protective alkalinity (i.e. a pH of 10.5 or above). Control of the CIL circuit pH will be achieved via the addition of milk of lime to the ball mill feed, CIL feed distribution box, CIL Tank No. 1, CIL Tank No. 3 and CIL Tank No. 5.

Cyanide will be used in the plant to affect gold and silver dissolution. Cyanide will be added to the circuit as a 30% sodium cyanide (NaCN) solution. Cyanide solution will be added to CIL Tank No. 1 and HMPG CIL Tank No. 1. The ability to add cyanide solution directly into subsequent CIL tanks to trim the cyanide concentration will also be available. The cyanide consumption will be 1.35 kg/t ore.

A tower crane will be installed within the CIL area. This crane will be used to conduct the necessary maintenance tasks and will also be used to periodically remove the inter–stage screens for cleaning. Maintenance frames will be provided to house the screens, for cleaning with high pressure washers.

The CIL area will be serviced by two sump pumps delivering spillage back to the CIL feed distribution box.

17.2.5 Acid Wash and Elution (Desorption)

The desorption circuit will be shared by the HMPG CIL and CIL circuits. It will consist of separate acid wash and elution columns. A cold acid wash will be utilized. Following acid wash, gold will be eluted from the carbon, utilizing a split Anglo American Research Laboratory (AARL) elution process. The desorption circuit will be designed to operate for a single cycle per 24 hour period. An average carbon loading (gold + silver) of approximately 2,000 g/t will be achieved, based upon the completed test work program. This corresponds to a required carbon movement of 6 tonnes per day.

During the desorption process, carbon will firstly be acid washed to remove accumulated calcified scale. After acid wash, the carbon will be transferred to the elution column where the loaded carbon will be conditioned with strong caustic-cyanide soak solution. The gold and silver will then be stripped (desorbed) from the carbon by eluting with a heated solution. This will produce a batch of pregnant eluate, from each cycle, which will be recycled through the electrowinning circuit.

The acid wash column fill sequence will be initiated by pumping carbon from the first tank from the respective CIL tank farms into the acid wash column via the loaded carbon screen. Carbon will gravitate from the loaded carbon screen directly to the acid wash column. Once the acid wash column is filled to the required level, the carbon fill sequence will be stopped.

The acid wash cycle will utilise a 3% w/v hydrochloric acid solution. This dilute acid will be prepared by the addition of raw water and neat (32%) hydrochloric acid into the hydrochloric acid tank. The acid wash sequence will involve the injection of the dilute acid solution into the column, by the hydrochloric acid circulating pump, via the feed manifold located beneath the column. The pump will continuously run and allow the acid to circulate through the column, back to the tank. Acid circulation will occur for 1 h. Upon completion of acid circulation, the acid rinse cycle will be initiated by pumping water through the column to displace the spent acid solution to the tailings thickener. For the rinse cycle, four bed volumes (4 BV) of water, at 2 BV/h, will be pumped through the column. Upon completion of the acid rinse cycle, the carbon will be hydraulically transferred to the elution column.

The elution sequence will commence with the injection of a set volume of water into the column along with the simultaneous injection of cyanide and caustic solution. A set amount of cyanide and caustic will be added in order to achieve a 2% w/w NaOH and 2% w/w NaCN solution. Both reagent additions will be automatically stopped once the prescribed volume has been added. The pre–soak period will then commence. During this period, the caustic solution will be circulated through the column and pre–heated to 95°C. Upon completion of the pre–soak period the elution process will commence and the gold will be stripped from the carbon.

The elution process will operate for a fixed time period to allow a set number of bed volumes of transfer water (or low gold concentration starter eluate) to pass through the column (8 BV, 2 BV/h). Prior to reaching the elution column manifold this solution will pass through a heat recovery exchanger and the primary heat exchanger (diesel–fired) to raise the solution temperature to 120°C. Initially, eluate emerging from the elution column (4 BV) will be directed to the pregnant eluate tank for electrowinning. The latter eluate solution (4 BV) will be directed to the starter eluate tank, for re-use in subsequent elution cycles.

A temperature probe, located after the elution heater, will monitor the eluate solution temperature and will be used to control the heater output. Eluate exiting the elution column will pass through a recovery heat exchanger. The recovery heat exchanger will reduce the discharge eluate temperature whilst indirectly pre–heating the strip solution inflow. The recovery heat exchanger will also sufficiently reduce the temperature of eluate reporting to electrowinning, thus reducing the occurrence of flashing within the electrowinning cells.

Upon completion of the elution process, the elution heater will be switched off. This will allow the column and its contents to cool down to below 100°C, prior to carbon being transferred to the carbon regeneration kiln.

17.2.6 Carbon Regeneration

The carbon will be hydraulically transferred from the elution column to the carbon regeneration circuit. Water will be delivered to the base of the elution column at a pressure suitable to initiate carbon transfer. The eluted carbon will pass over a dewatering screen, which will allow excess water to drain off the carbon, prior to the carbon being fed to the carbon regeneration kiln feed hopper. Carbon will be withdrawn from the feed hopper, by screw feeder, and fed to the carbon regeneration kiln at a rate of approximately 350 kg/h. The carbon regeneration kiln will operate at 650°C. The hot carbon will discharge from the kiln, via a carbon sizing screen, directly into the carbon quench tank. The quenched carbon will then be pumped to the respective CIL circuits.

17.2.7 Electrowinning and Refining

Once sufficient pregnant eluate is available within the pregnant eluate tank, the electrowinning sequence will be initiated, by starting the pregnant eluate pump. This pump will transfer the pregnant solution to the electrowinning cell (sludging basketless type, 1,000 mm x 1,000 mm cathode, 18 cathodes, rectifier 2,500 A) located within the gold room. The gold will be electroplated onto steel wire mesh cathodes. The electrowinning cell discharge will be continuously returned the eluate tank until the precious metals have been recovered from the eluate.

Upon completion of the electrowinning cycle, the plated cathodes will be removed from the electrowinning cells and the gold sludge washed off the cathodes. The gold bearing sludge will be recovered by, and filtered, using a mobile pan vacuum filter. Filtered gold sludge will be dried in a calcine oven. The dried calcine will then be mixed with a prescribed flux, prior to being charged into the diesel–fired crucible smelting furnace. The gold will be poured into moulds, cooled and cleaned, after which they will be weighed and stored within the vault. The slag produced will be recycled back to the grinding circuit.

The electrowinning and refining functions will be enclosed in a secure area with limited access. A secure room with vault door will be provided to store dried gold bearing sludge and gold bars.

17.2.8 Tailings Handling and Cyanide Detoxification

CIL tailings will gravitate directly to the cyanide detoxification tank where sodium metabisulfite, air, copper sulphate and milk of lime will be added to complex the residual cyanide or oxidise it to cyanates. The sodium cyanide in the detoxified tailings will be reduced to below 1 mg/L total cyanide maximum (0.8 mg/L CN_{total} average). Pulp residence time within the detoxification tank will be 90 minutes.

Detoxified slurry from the cyanide detoxification tank will gravitate to the zinc iron (cyanide) precipitation tank, where residual iron cyanide levels will be reduced, by the addition of zinc sulphate. Pulp residence time within the iron cyanide precipitation stage will be 60 minutes. Slurry exiting this stage, with total cyanide content of below 0.8 mg/L, will gravitate to the high rate tails thickener (18 m diameter).

Tailings thickener underflow, at a solids content of 58% solids, will then feed one of the two pressure filters, located within the filtration circuit. The tails filtration circuit will produce a filter cake with moisture of about 14.8% w/w. Filter cake will discharge directly to the filtered tails stockpile. Depending on the need for backfill, the filtered product will be transported to either the paste plant or the filtered tailings storage facility.

17.2.9 Paste Backfill Plant

The paste backfill plant will be serviced by a 1,000 t filtered tails stockpile. Tails will be reclaimed from the filtered tails stockpile, by front end loader, to a reclaim hopper. Filtered tails will be conveyed from this reclaim hopper directly to a continuous paste mixer. Cement will be added to the paste mixer, from the cement storage bin, via a weigh-feeder and screw feeder combination. The paste mixture will be discharged to a paste holding hopper, from which the paste will be distributed underground, by means of a positive displacement pump.

17.2.10 Tailings Storage Facility

Filtered tailings not used for paste backfill manufacture, will be stored at a dry stack TSF. The filter cake will be loaded into standard road trucks, at the filter plant, for haulage to the tailings disposal facility. Once discharged tailings will be compacted and profiled with water management structure put in place to control seepage and run-off.

17.2.11 Reagents Handling and Preparation

The reagents that will be used within the plant are:

- Hydrated lime for control of pH
- Flocculant for thickening
- Sodium cyanide for dissolution and desorption
- Copper sulphate pentahydrate for cyanide detoxification
- Sodium metabisulfite for cyanide detoxification
- Zinc sulphate heptahydrate for iron (cyanide) precipitation
- Antiscalant to reduce fouling in the carbon wash and stripping circuit
- Fluxes for smelting charge preparation
- Hydrochloric acid for washing the carbon
- Caustic soda for neutralisation and pH control
- Kerosene for use as blanking agent to passivate the surfaces of activated carbon in carbonaceous ores
- Various fluxes for smelting of the doré.

Reagents will be delivered to the Ollachea site by road transport. Reagents will be mixed, to the required concentration in dedicated mixing tanks, prior to gravitating to their respective holding tanks. Reagents will be metered to the respective process usage points by dedicated reagent metering pumps.

17.2.12 Plant Water Circuit

The process plant will utilise both raw water and process water. Process water will predominantly consist of tails thickener overflow and tailings filtrate. Tailings thickener overflow will be pumped to the process water tank (850 m^3, 4 h storage). From the process water tank, process water will be distributed to the grinding, gravity and CIL circuit, the predominant users. Any shortfall in process water will be made up from the raw water circuit. Any excess process water will be bled from the process circuit, by directing this water to the water treatment plant.

Raw water for the paste plant, process plant and mining operation will be sourced from the Oscco Cachi River (150 m^3/h). River water will be collected and gravity fed to the Minapampa water tank, from where it will supply water to the shotcrete plant, paste plant and mine. Water from the Minapampa water tank will also gravitate to the process plant raw water tank, and forms the process plant raw water supply (24 m^3/h). The plant raw water tank (850 m^3) will provide combined raw water and fire water reserve, of 16 h and 460 m^3 respectively. Raw water from this tank will predominantly be used in the acid wash and elution circuits, for reagent make-up, as fire water and for gland seal water. The safety shower network will also receive its water from the raw water tank.

17.2.13 Water Treatment Circuit

The site wide and process plant water management plan will utilise a number of diversion drains and surface drains, aimed at isolating contact water from non-contact water. Further, the process plant site will incorporate three water management ponds: a plant run-off pond, a sediment pond and an environmental control pond.

Surface run-off from the crushing and grinding benches, which has not come into contact with deleterious elements, will be captured within the dedicated plant run-off pond.

The CIL bench will contain the CIL circuit, acid wash, elution and regeneration circuit, gold room and reagent mixing and storage areas. Run-off from this bench may contain contaminants and will be directed to the sediment pond. Potentially acidic mine water, drainage from the ROM pad, drainage from the lower waste dump, TSF seepage water and overflow from the plant run-off pond will discharge to the sediment pond. The sediment pond will provide surge capacity ahead of the water treatment plant.

Water will be recovered from the sediment pond, to the water treatment plant, by submersible pumps with the option of being pumped to the process water tank if there is a requirement for process water make-up.

The process plant water balance indicates a positive balance; hence, a bleed stream from the process water tank will be directed to the water treatment plant. However, in the case that make-up water will be required within the process water balance, this make-up water will preferentially be supplied from the sediment pond, prior to utilizing water from the raw water tank.

Dynamic modelling indicates that the anticipated water egress from the mine will be equivalent to a peak flow rate of 160 m³/h. This water will be low in pH and will contain ferrous iron, which will require oxidation to ferric iron, in order to allow for the precipitation of an insoluble ferric hydroxide.

Consequently, the water treatment facility will consist of an oxidation tank, for oxidation of iron, a lime neutralisation tank, for iron precipitation and pH adjustment and a clarifying thickener to recover precipitates. Both the oxidation and neutralisation tanks will have a 60 minute residence time. Slurry pH adjustment will be achieved by milk of lime addition to the lime neutralisation tank.

Overflow from the neutralisation tank will gravitate to the water treatment plant clarifying thickener. Thickener overflow will gravitate to the site environmental control pond, prior to discharge off-site. Underflow from the thickener will either be recycled to the oxidation tank, until a suitable underflow density is achieved, or pumped to the tails filtration area surge tank for disposal within the final tails filter cake.

Cuncurchaca, the location of the TSF, will incorporate two water ponds: a run-off pond and a seepage pond. The TSF surface will be partially covered during construction (rain coats) and run-off will be captured in a dedicated run-off pond. TSF seepage will be collected in a seepage pond and pumped to the TSF’s water treatment plant (carbon columns).

17.2.14 Compressed Air Circuit

High pressure air (700 kPa) will be reticulated throughout the process plant, by two rotary screw compressors, operating in a lead-lag configuration. The entire high pressure air supply will be dried, and will be used for both instrument air and all plant air services.

Low pressure air will be supplied by dedicated compressors. Low pressure air will be used in the HMPG CIL circuit for airlifts, and as a source of oxygen for CIL pre-aeration, HMPG CIL pre-aeration, water treatment pre-aeration and for cyanide detoxification.

Oxygen, to the CIL circuits, will be supplied by a pressure swing adsorption (PSA) oxygen generator. Oxygen distribution to the CIL tanks will be achieved via the agitator shaft.

17.3 Product/Materials Handling

The doré bars produced at Ollachea will be transported by a security vehicle to Puerto Maldonado. From Puerto Maldonado, the shipment will be air freighted to Lima airport for transfer to the refining company. The refining company takes responsibility at this point and transfers the doré to the selected refinery's location via international air freight.

17.4 Energy, Water, and Process Materials Requirements

The Ollachea process plant will have an installed power of 8,996 kW, with an operating power draw of 5,001 kW. This equates to an annual energy consumption of 43,813,015 kWh/annum or 39.83 kWh/t of ore processed.

Make-up water requirements, from the Ossco Cachi River, are estimated at 150 m^3/h; and predominantly cater for process plant raw water (24 m^3/h), mine water, paste (41.4 m^3/h) and shotcrete plant water demands.

Process material requirements are limited to reagents and consumable items, as summarized in Table 17-1.

Table 17-1 Reagent and Consumable Consumption Summary

Reagent/ Consumable	Type	Reagent Consumption			Packaging
		kg/t	kg/day	t/month	
Hydrated Lime	$Ca(OH)_2$	2.26	6,780	203	20t Tanker
Sodium Cyanide	NaCN	1.35	4,050	122	1t Bulk Bag
Sodium Metabisulfite	$Na_2S_2O_5$	3.1	9,300	279	1t Bulk Bag
Copper Sulphate Pentahydrate	$CuSO_4.5H_2O$	0.06	180	5	1t Bulk Bag
Caustic Soda	NaOH	0.13	390	12	1t Bulk Bag
Zinc Sulphate Heptahydrate	$ZnSO_4.7H_20$	0.43	1,290	39	1t Bulk Bag
Flocculant	Magnafloc 10	0.03	90	3	25kg Bag
Hydrochloric Acid	HCl	0.26	780	23	250kg Drums
Kerosene		0.1	300	9	Tanker
Ball Mill Media (75mm)	Cast, Mild Steel	0.9	2,700	81	2t Bulk Bags
Activated Carbon	Coconut Shell	0.05	150	5	Bulk Bag

17.5 Comments on Section 17

The proposed Ollachea process plant design will use gravity and CIL technology appropriate to achieve reasonably high recoveries of coarse and fine gold from the preg-robbing pyrrhotite-bearing slate host. The process design is based on metallurgical test work including comminution, gravity concentration, leaching, thickening and filtration work completed to date. The plant throughput rate selected (1.1 Mtpa) is appropriate given the mine plan presented in Section 16.

18 PROJECT INFRASTRUCTURE

18.1 Roads and Logistics

Road access for continued exploration activities, mine development and operation, plant access and project infrastructure including construction and operations camp sites and tailings storage facility is from the Interoceanic Highway. Access to the Ollachea Project is relatively straightforward, although road construction to provide access to the mine, plant, camp and TSF will be required.

The proposed Ollachea Project process plant site is immediately to the west of the Interoceanic Highway. A road of approximately 1.3 km long was built to the exploration access portal in late 2011. This road will also be used to build and access the plant site.

The Ollachea camp site will be located east of the process plant and will require an approximate 540 m-long access road from the Interoceanic Highway.

The access road to the TSF from the Interoceanic Highway is approximately 1.8 km long. In an effort to minimize cuts and fills on steep slopes, the TSF access road is designed for one lane traffic, with intermittent traffic pullouts.

Access roads to the Minapampa area for exploration activities pass through the town of Ollachea. A new access road bypassing the town of Ollachea is planned that will merge with the existing access road. This road will be used to provide access to ventilation raise surface breakthrough locations above the mine to allow for installation and maintenance of ventilation fans. Also, the road will provide access to the paste and shotcrete plants in the Minapampa area for cement delivery. The Minapampa access road will intersect with the existing exploration road and will be approximately 2.3 km long at sustained grades of 11 to 12%. The road has been designed as a single lane with intermittent traffic pullouts and will be constructed during the first half of 2013.

No additional road construction is contemplated for the Project.

18.2 Waste Storage Facilities

According to the feasibility study mine waste schedule, the Ollachea project will require permanent disposal of 2.45 Mt of waste rock and temporary storage of 0.6 Mt of low-grade ore.

A waste rock disposal area has been has been selected at Challuno (Lower Portal Waste Rock Dump) based on its proximity to the lower portal of the mine access drive. The lower portal waste rock dump has been designed to contain the permanent waste rock disposal requirements of the feasibility mine waste schedule (2.45 Mt). The lower portal waste rock storage facility is located immediately south of the process plant, near the lower portal.

Low-grade ore will be temporarily stockpiled at the upper portal in the Minapampa area (Upper Portal Ore Stockpile). A stockpile for low grade development ore has been designed for up to 300,000 tonnes of storage near the upper portal of the mine access drive. The stockpile will be located immediately west of the paste plant at Minapampa. The low-grade development ore will be transported through the mine to the plant site for processing, as required. The maximum size of the stockpile will be 298,000 t (design capacity of 300,000 tonnes).

18.2.1 Geotechnical Site Investigations – Waste Storage Facilities

Lower Portal Waste Rock Dump

The lower portal waste dump will be located within 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 waste rock dump 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 investigations at the lower portal waste dump included test pits and geotechnical boreholes. Three test pits excavated at the lower portal waste dump 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.

Four boreholes were drilled in the vicinity of the lower portal waste dump to depths of 21 to 40 m in 2012. 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 approximate depths of 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.

Upper Portal Ore Stockpile

Six test pits were excavated in the Minapampa area, with two of the test pits located within the projected ore stockpile footprint. Soil conditions typically consisted of 0.2 to 0.7 m-thick topsoil layer overlying clayey sand and gravel and silty sand and gravel (SC, GC, SM and GM). The native soils are medium dense to dense, non-plastic, and dry to wet. Groundwater was encountered in four of the excavations at depths of 1.1 to 2.0 m. Organic (bofedal) soils were mapped along the southeastern toe of the stockpile. The bofedal is inferred to be relatively thin (generally less than 1 m thick).

18.2.2 Lower Portal Waste Dump Design

The FS mine waste schedule includes permanent surface waste rock disposal requirements of 2.45 Mt for the life of mine. A waste rock storage facility has been designed to permanently contain the life of mine waste rock. The waste rock storage facility is located immediately south of the process plant, near the lower portal.

The ultimate waste dump will have a maximum height of approximately 125 m 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 1.9H:1V. Total waste rock storage is estimated to be 2.45 Mt, based on an estimated bank bulk density of 2.80 and a 35% (compacted) swell factor.

A platform will be constructed in cut at the toe of the waste dump for improvement of dump stability. Topsoil from the waste dump footprint will be salvaged for use in reclamation.

Waste dump inter-lift benches will include surface water ditches to capture and shed contact water off the waste rock storage facility and direct it to a catchment pond located north of the facility. The waste rock storage facility will include an effluent collection system consisting of gravel drains at the foundation surface. Effluent from the facility will be captured at the toe and directed to the catchment pond for monitoring and treatment as necessary.

18.2.3 Upper Portal Ore Stockpile Design

Low-grade ore will be temporarily stockpiled at the upper portal in the Minapampa area for processing throughout the life of mine. Throughout the life of mine, a total of 0.6 Mt of low-grade development ore will be temporarily stockpiled and subsequently transported through the mine to the process plant for processing.

The upper portal ore stockpile will be located west of the proposed paste and shotcrete plants at Minapampa. The ore stockpile has been designed for a maximum height of approximately 42 m, providing a maximum ore storage capacity of 300,000 tonnes. The stockpile will be constructed in 8 to 10 m- thick lifts with 6 m-wide interlift benches, for an overall slope of 1.9H:1V. Processing of the low-grade ore throughout the life of mine will allow the mine to maintain a stockpile no greater than 300,000 tonnes.

The upper portal ore stockpile will include inter-bench and perimeter channels to capture and shed contact water off the stockpile and direct it to the contact water pond located at the toe. The ore stockpile will also include a foundation drain system that will serve to drain the foundation soils as well as capture seepage from the stockpile. The drain system will consist of a drain constructed along the toe of the stockpile with finger drains extending from the toe drain to the interior of the stockpile footprint. The drains will consist of drain gravel wrapped with non-woven geotextile and the toe drain will include a perforated collection pipe. The drain system will route seepage by gravity flow to the contact water pond located at the toe of the stockpile.

18.3 Tailings Storage Facilities

18.3.1 Tailings Management

The TSF presented in this subsection has been designed to store 5.85 Mt of tailings corresponding to 11 years of mine operations, as taken from the FS mine plan. Tailings management for the project will include both surface storage, as filtered tailings, and underground paste backfill. Surface tailings storage will account for approximately 61% of the LOM tailings, while paste backfill will account for the remaining 39% of the LOM tailings stream. Considering LOM tailings production of 9.34 Mt, the TSF requires storage for 5.70 Mt of filtered tailings. Table 18-1 presents the mine tailings production criteria used for design of the TSF.

For surface disposal, filtered tailings disposal was selected as the most suitable tailings management option for the Project. The primary criterion for selecting filtered tailings was to obtain the required storage volume. 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 TSF seepage/effluent.

Table 18-1 Mine Tailings Production Criteria

	Unit	Value
Total tailings production	Mt	9.34
Mine life	Year	11
Percentage surface tailings disposal	%	61
Percentage underground tailings disposal	%	39
TSF storage capacity	Mt	5.85

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%, which is the approximate optimum moisture content as determined by the standard Proctor compaction.

Filtered tailings will be transported approximately 4.0 km from the plant site to the principal TSF 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 TSF sites.

A contingency tailings management area has been designated for temporary tailings storage adjacent to the TSF access road.

18.3.2 TSF Location

A TSF site selection study considering five alternative sites for the TSF was completed by AMEC (2011). The site selection study ranked the five potential TSF sites based on perceived economic, technical, and social factors. At the completion of the study, two viable options were identified for the TSF. The Cuncurchaca site was selected as the preferred site for the TSF.

The tailings transport route to the TSF is along the existing Interoceanic Highway. Construction of an access road to the TSF will be required from the Southern Interoceanic Highway to the TSF, starting from a point approximately 2 km north of the process plant. The TSF access road has been designed as a one-lane road with intermittent traffic pullouts. Access to the temporary contingency tailings management facility is also provided by this road. The access road is approximately 1.8 km long with approximate maximum grades of 10%. A new bridge will be required to cross the Cuncurchaca creek for the access road. The bridge will have an approximate 30 m span.

An electrical transmission line currently crosses the TSF location with a tower located within the lower portion of the TSF footprint. The transmission line will be re-routed around the TSF to allow construction and operations of the TSF.

18.3.3 TSF Geotechnical Site Investigations

Ten geotechnical boreholes were drilled at the TSF footprint in 2012. The boreholes ranged in depth from 40 to 85 m, and included 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 movement.

Ten test pits were excavated for geotechnical characterization of surface conditions at the TSF site and TSF access road alignment 2012.

The TSF area consists of Quaternary sandy gravel and cobble deposits resulting from a series of debris flow events from the Cuncurchaca drainage basin. Soil samples recovered during drilling were generally 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 raveling and shallow slope failures have been observed, likely a result of construction cuts for the Southern Interoceanic highway. Otherwise, no indications of deep-seated or active large-scale slope movement were observed during the field reconnaissance and terrain hazard assessment.

18.3.4 TSF Design

The TSF 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 145 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.

Foundation preparation will consist of clearing and grubbing of significant vegetation in the TSF shell area (“Zone A”) and removal of topsoil and other organic material as is practical and deemed required by the engineer. The topsoil will be stockpiled for reuse at closure.

A starter buttress will be constructed at the toe of the TSF for stability and erosion protection. The starter buttress will be constructed of compacted colluvial soils sourced from within the TSF footprint.

The TSF design considers two zones for tailings placement: (i) perimeter TSF tailings shell (Zone A), and (ii) interior TSF zone (Zone B).

The compacted “Zone A” tailings zone will form a structural “shell” that will be located at the exterior perimeter area of the TSF. Zone A tailings will be placed in 0.3 m-thick lifts that will be compacted to achieve at least 95% of the maximum dry density as determined by the standard Proctor test (ASTM D698).

The Zone B tailings are located in the interior of the TSF, between the compacted tailings Zone A and the natural ground. The Zone B tailings are located in less critical areas compared to Zone A as pertaining to stability of the TSF. Zone B tailings will be compacted in maximum 0.5 m-thick loose lifts to at least 90% of the maximum dry density as determined by the standard Proctor. Zone B will allow some flexibility for tailings placement during wet weather or upset plant conditions.

Stability evaluations of the TSF indicate acceptable factors of safety for static and seismic (pseudo-static) loading conditions.

Seepage analyses conducted for the TSF demonstrate that effluent from the TSF will be in the order of 1.0 to 2.5 L/m. Performance of filtered tailings facilities on other projects indicates similar seepage rates relative to the TSF footprint area. The TSF design includes a layer of compacted filtered tailings that will serve as a low-permeability base liner. Laboratory testing on bench scale tailings samples prepared during the feasibility study indicate the hydraulic conductivity of compacted filtered tailings to be on the order of 10^{-7} cm/sec, which is similar to a conventional compacted clay liner system. Finger drains consisting of free-draining gravel wrapped in filtering geotextile will be constructed on top of the base tailings layer to provide drainage to the filtered tailings. Seepage water will be directed to a lined seepage collection pond for monitoring and treatment, as required. A synthetic liner system is not considered necessary, considering the low hydraulic conductivity of the compacted filtered tailings, negligible anticipated seepage from the TSF, and deep groundwater at the site.

Protection of the tailings slope from water and wind erosion will be required as the filtered tailings are considered to be highly erodible. An erosion protection layer will be progressively placed on the tailings slope during operations and will become part of the reclamation cover. The erosion protection layer will be sourced from colluvial soils from the TSF footprint. Exposed tailings surfaces are expected to require moisture control during dry periods. Surfactants may be considered for access roads on the TSF as well as for areas of inactive tailings placement.

Surface water runoff that has not come into contact with tailings (“non-contact” water) will be captured by perimeter diversion channels and routed around the TSF to discharge to the natural drainage down-gradient of the facility. Since the TSF footprint will increase over the operations lifetime, construction of temporary diversion channels for different expansion phases of the TSF will be required.

The exposed tailings surface will be compacted and sloped to minimize infiltration. Contact runoff water will be directed off the TSF to channels conveying the runoff to a lined sedimentation pond for monitoring to ensure compliance with quality standards prior to discharge to the environment.

18.3.5 TSF Closure Considerations

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

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

18.4 Water Management

Water management for the mine, plant and TSF sites and water treatment facilities are considered in the mineral processing plant design through the use of a Goldsim model. The results of this model have demonstrated that the Ollachea project water balance is a positive water balance with excess water requiring discharge into the environment. The total water usage required by the Project is estimated to be 84 m^3/h. The total water inflow into the project area is estimated to be approximately 388 m^3/h, exceeding the Project requirement significantly. The estimated excess water of 304 m^3/h, most of which is groundwater seepage is likely to require treatment before being discharged into the environment. The total water outflow from the project is estimated to be 391 m^3/h.

As mentioned above, the main inflow of water into the project boundary is seepage into the mine. Seepage into the mine was estimated using Modflow USG (unstructured Grid), which is a 3-dimensional groundwater flow model developed by the USGS and AMEC. This version provides the ability to incorporate nested grids at any location and at variable scales within the model. In the case of the Ollachea groundwater model, USG allows for complex features and a high degree of discretization in the area of the mine for accurate mine drainage simulation.

All effluent from the site will be required to meet maximum permissible limits regulations. Any stream not meeting these requirements will be routed to the treatment plant. The water quality parameter of concern based on geochemical laboratory testing is iron in the mine seepage. The water treatment facility design criteria are for a treatment rate of up to 350 m^3/h, which is approximately the estimated maximum potential mine inflow rate (AMEC, 2012).

18.5 Camps and Accommodation

A permanent operations camp facility has been designed and will be located south of the Challuno area, in the vicinity of the lower portal and within 500 m of the Interoceanic Highway. The camp will have catering and accommodation capacity for approximately 275 persons

18.6 Power and Electrical

The Project will connect to the 138 kV transmission lines from San Gaban to Azangaro that passes over the Ollachea project. The San Gaban II hydroelectric generating station is located on the Ollachea River approximately 10 km from the Project. A 138 kV supply line will be installed from the main transmission to the plant site, and will have a length of approximately 1.2 km. This line will feed a substation that will distribute power to the plant site, the underground mine, the camp site and other auxiliary buildings.

No electrical power supply is anticipated for the tailings disposal facilities.

18.7 Fuel

Diesel fuel will be required for underground and surface mobile equipment, process equipment, and onsite emergency power generation equipment. A fuel storage facility will be located at the plant site and fuel trucks will be used to distribute fuel underground

18.8 Water Supply

Water for underground mine operations will be re-circulated from sumps within the mine where possible. Mine drainage will be diverted to a water treatment plant at the plant site where it will be combined and treated with water discharged from the mineral processing facility. Plant make-up water and all other water supply for the plant and other surface infrastructure will be supplied from the water treatment plant and drawn from the Oscco Cachi and Ollachea Rivers as required.

18.9 Comments on Section 18

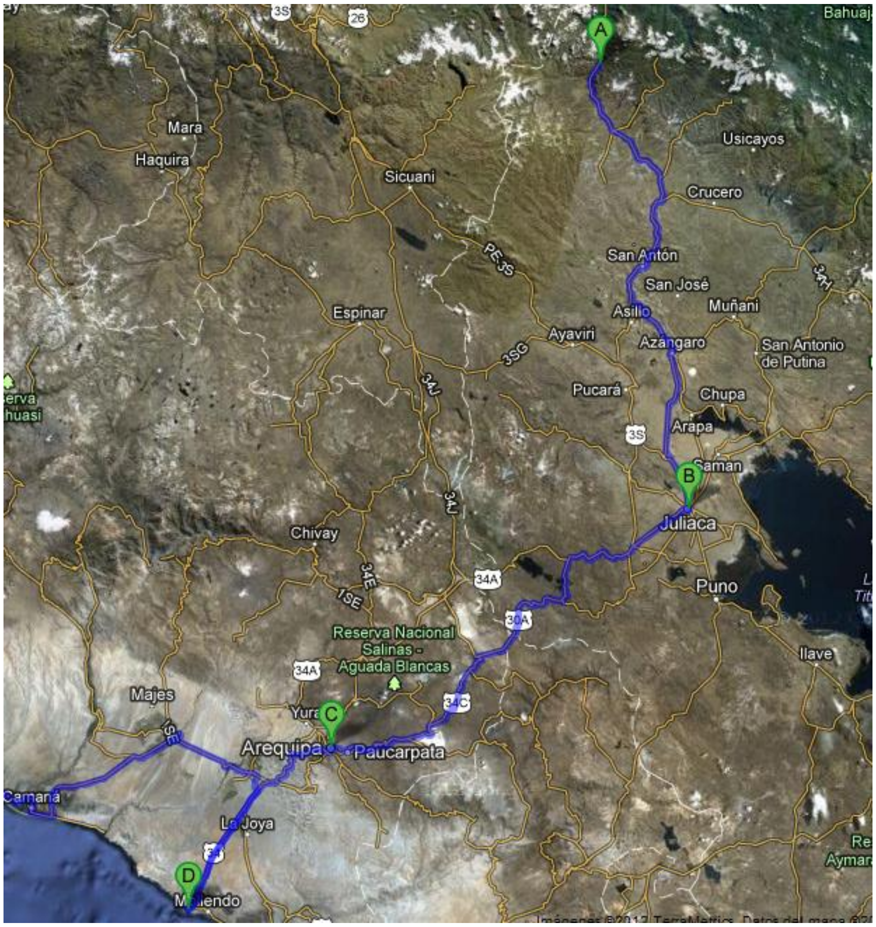
18.9.1.1. Accesses and Roads

The mine may be accessed by the Inter ocean road that facilitates transport from the Coast to the site, 200 meters from the site. No rail transport is available for the mine site. The nearest airport is in Juliaca with daily connections to Lima. The port used by MKK for maritime shipments is mainly Matarani port site.

The access to the project will be made through the Matarani –Arequipa – Juliaca –Ollachea (891 Km) route and the journey takes approximately 15 hours.

Figure 18-1 shows the routes defined to access the project facilities.

Figure 18-1: Ollachea Project Routes



(Image from Google map)

Table 18-2 shows the distances between the points of interest of the project. This matrix allows knowing how far the project is from major cities and town in Peru, and studies the different sources of resources for the project.

Table 18-2: Matrix of Distances

Distance (Km)	Ollachea Project	Juliaca	Arequipa	Matarani	Callao
Ollachea Project	0	234	492	596	1512
Juliaca	234	0	258	363	1279
Arequipa	492	258	0	116	1029
Matarani	596	363	116	0	1060
Callao	1512	1279	1029	1060	0

18.9.1.2. Ports

The Matarani Port is located at 170 km from Moquegua and 596 km from Ollachea Project and it has been the major port where the cargos will arrive. The systematic and continuous operation during all year as well as its location and capacity, makes it the most feasible port for this duty.

Figure 18-2: Location Map of Matarani Port

(Image from Google map).

19 MARKET STUDIES AND CONTRACTS

19.1 Gold Marketing

Gold production is likely to be sold on the spot market, by marketing experts retained by or on behalf of MKK. Gold can be readily sold on numerous markets throughout the world and it is not difficult to ascertain its market price at any particular time. Since there are a large number of available gold purchasers, MKK would not be dependent upon the sale of gold to any one customer. Gold could be sold to various gold bullion dealers or smelters on a competitive basis at spot prices.

Project gold production will be in the form of gold doré bars and will likely be refined via a contract with a refining company in Europe or North America. There are several large gold refineries in North America and Europe that have a long history of service to the mining industry. Although MKK has not explicitly contracted any of these companies, the primary refineries that will likely be considered are:

- Johnson Matthey – Salt Lake City, Utah or Brampton, Ontario
- Canadian Mint - Ottawa, Ontario
- Metalor – Marin, Switzerland

Based on typical rates from refining companies, the following terms should be achievable:

- Gold payment would be based on 99.9% of the London gold price.
- A treatment charge of US$1.10 per troy ounce of doré is anticipated based on a delivery point of refinery.

A recognized security company will be responsible for transporting the doré from the mine site to a regional airport for shipment to the selected refiner. The major refiners will normally make arrangements for the shipment from a major airport to the final destination. The doré will initially be transported by road from the mine site to Puerto Maldonado and thence to Lima International Airport.

MKK plans to make shipments on a bi-monthly basis. Transport costs, including insurance to Lima airport, are anticipated to be US$4,130 per shipment plus US$0.89 per US$1000 of shipment value plus US$1.18 per kg doré.

19.2 Comments on Section 19

In the opinion of the QPs MKK will be able to market gold produced from the Project at terms corresponding to industry norms.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

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

20.1.1 Physical Baseline

The study area is located in the Ollachea river sub-watershed located in the Inambari river watershed, which flows to the Atlantic Ocean basin. The maximum monthly average flow in the Ollachea river, adjacent to the property, is 71.15 m^3/s in February, while the minimum monthly average flow of 7.62 m^3/s is observed in August.

Results of water quality monitoring in the study area indicate that water quality generally meets the national water quality standards. Exceptions include instances of high iron, aluminum, coliform bacteria and thermotolerant coliform bacteria concentrations in the Oscco Cachi stream. Exceptions for the Ollachea river include high iron, manganese, and aluminum concentrations plus some bacteriological contamination in springs.

Air quality has been measured using 12 monitoring points in which the parameters analyzed meet the Peruvian environmental regulations for lead, arsenic, PM10, PM2,5 SO2, NO2, H2S and O3 concentrations. However in 2010, 2011 and 2012 it was found that 3 points of monitoring for carbon monoxide (CO) exceeded the ECA (10000ug/m3-8horas). The three unexpected readings were most likely caused by their location near the Interoceanic highway. These locations are: Air-05 (Facilities Intersur), Air-06 (crusher Intersur) 12,235 ug/m3 in the month of May 2011, Air-04 (Ollachea) with 15,812 in the month of March 2010. These results could be attributed mainly to vehicular traffic near the monitoring points (Air-05, Air-06) and the activities of the population in the place (Air-04). It should be noted that in 2012 these monitoring points' levels were below the Environmental Quality Standard for Air.

Baseline noise levels registered in the industrial areas of the study area were below the daytime and night time national environmental noise standards. Noise levels recorded in the town of Ollachea were above daytime and night time standards, mainly due to Interoceanic Highway traffic.

Current land use in the study area consists of natural grassland, artificial or plantation of woodlands and unused or unproductive lands. The land use potential has been identified as land suitable for forest production, grazing, permanent farming and protection land.

20.1.2 Biological Baseline

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

A total of 72 plant species were identified in the study area, grouped in 34 families of vascular and non-vascular plants. The only species of flora identified is considered 'vulnerable' according to the list of Peruvian protected species is the *Escallonia resinosa*.

Eleven species of birds pertaining to 10 families have been identified in the study area. One type is categorized as 'endangered', the Vultur griphus which is not within in the environmental direct area of influence of the project.

Additionally, five species of wild animals have been observed in the study area. Of those five species, two are protected, the Tremarctos ornatus is endangered and the Puma concolor is near threatened. Both of the protected species are not within in the environmental direct area of influence of the project.

The water bodies observed contained 11 species of macrozoobenthos, 54 species of phytoplankton and 16 species of zooplankton. A low density of the *Oncorhynchus mykiss* trout was also observed.

20.1.3 Anthropological Baseline

Preliminary anthropological surveys were carried out by MKK to support the semi-detailed EIA carried out in 2007 and 2008 for exploration drilling permits and those carried out in 2010 for modifications of the exploration permits to support additional exploration drilling and the excavation of the exploration access drive. Surveys were also carried out prior to the construction of the Interoceanic Highway.

A reconnaissance of archaeological sites has been carried out on the Project area. A few archaeological sites have been identified in the Challuno process plant site and Cuncurchaca TSF. These sites have now been cleared of archaeological remains.

20.1.4 Socioeconomic Description

The socioeconomic study area consists of the Ollachea district which comprises the Ollachea settlement, located near the Project area.

The population of the study area amounts to 4,919 inhabitants, with decreasing population trend from 2005 to 2007. More than half of the population consists of men, while the median age of the population is 25 years old. The majority of the population are Quechua speakers (83.96%) and the most important religion is Catholic.

The majority of the houses in the study area (71.46%) are located in rural areas. Most houses (87.37%) consist of independent houses and the main building materials are adobe (25.95%) and stones and clay (63.34%). Although the district of Ollachea has access to electricity, only 34.14% of the population gets access to this service in their household. The main source of water in the households (93.76%) comes from rivers and springs. Only 1.05% of the households are connected to the public sewage system.

Several schools are located within the study area, the majority being for primary education. Nonetheless, only 57% of the population between 3 to 24 years old currently attends an education centre. The majority of the population (50.17%) has primary education as the highest education level, while 25.89% of the population does not have any formal education. A total of 72.00% of the population in the study area are literate. Literacy rates are higher amongst men in the study area.

There are a total of 10 health centers in the health unit which covers the study area, with only one health centre located directly within the study area. 60.52% of the population in the study area do not have health insurance, 33.40% are affiliated with the Seguro Integral de Salud while 4.03% are registered with ESSALUD. The main health issues in the study area consist of acute respiratory illnesses, pneumonia, and mouth infections and there is a high rate of malnutrition in children under the age of 6.

The main activities in the Ollachea settlement consist of artisanal mining, followed by agriculture and raising livestock. At the district level, the main activity consists of agriculture and the main crops grown are corn, potatoes, beans, ‘ocas’ and hot peppers.

According to a UNDP study done in 2006, the Ollachea district has a Human Development index of 0.393, which is one of the lowest in the Puno region

20.2 Environmental Issues

Current liabilities for the project are limited to the re-vegetation of drill platforms that are currently in use and closure of artisanal mine workings. Previously-used drill platforms have been formally closed and reclaimed.

The artisanal mine workings are restricted to an area measuring approximately 500 m x 100 m on the north flank of the Oscco Cachi River.

As part of the current surface rights agreement with the Community of Ollachea, MKK is monitoring the artisanal miners and taking actions to mitigate further environmental liability associated with the small-scale mining activities. This monitoring includes regular water quality determinations both up- and down-stream of the mine to monitor for possible contamination related to mining activities.

All requirements and plans for waste and tailings disposal, site monitoring, and water management both during operations and post mine closure is covered in section 18 of this report.

20.3 Closure Plan

A formal closure plan has been developed as part of the feasibility work plan for the Project.

The extent of closure plan for Ollachea is restricted to the mine portal and mineral processing plant areas and the closing of these areas require relatively limited work, given that the mine is an underground mine and the TSF will be progressively closed as it is developed. A budget of US$ 3.1 M for closure activities has been estimated as part of the capital cost estimate for the Project.

20.3.1 Lower Portal Waste Rock Dump

Primary objectives and criteria for closure of the lower portal waste dump include:

- Long-term physical stability;
- Geochemical stability;
- Erosion protection;
- Surface water control; and
- Restore to the extent possible the original land use.

To achieve these objectives, the following activities will be carried out at closure of the waste rock dump:

- To the extent possible during operations, potentially acid generating waste rock will be covered with non-acid generating waste rock on exterior slopes of the waste dump.
- Re-grade waste rock dump to 2H:1V overall slopes, with emphasis to restore the topography to landforms that replicate the natural landscape to the extent practical.
- Compact waste rock surface as practical to form low-permeability barrier layer to reduce infiltration of precipitation into the waste rock. It has been assumed that the slate waste rock will break down under compactive effort to form a low permeability cap on the waste rock dump. Should the waste rock not adequately break down to form a barrier layer, a low-permeability soil layer will be placed over the waste rock.
- Placement of an organic layer over the waste rock dump will be re-vegetated with native grasses and shrubs. The established vegetative cover will serve to reduce erosion due to surface runoff.
- Unnecessary access roads and diversion channels will be decommissioned and reclaimed. This task will include re-grading and re-vegetation of disturbed areas.

- Upgrade diversion channels and surface water drainage courses for closure storm events, with an emphasis to minimize post-closure maintenance requirements.

- The effluent collection system at the toe of the waste dump will be maintained during closure to monitor effluent volume and quality with respect to Peruvian water quality standards. Depending on effluent quality, the collected effluent will be directed to the environmental pond or treatment plant located at the process plant site.

- A monitoring and maintenance program will be implemented to address erosion, sedimentation, management of surface water, vegetation and management of effluent.

20.3.2 Upper Portal Ore Stockpile

Low-grade ore in the upper portal ore stockpile will be completely processed prior to closure. The stockpile area will be re-graded to restore natural topography and provide positive drainage to the Oscco Cachi drainage. An organic soil layer will be placed over disturbed areas and re-vegetated with grasses and shrubs native to the area.

20.4 Permitting

- MKK currently holds permits allowing the company to carry out exploration activities on the property including the development of the exploration adit which was in progress at the time of writing this report. Permits in place include an Authorization by the National Water Authority or Autoridad Nacional de Agua (ANA) to discharge residual water from the Ollachea Project to the Corani River and Oscco Cachi stream.

- Authorization by ANA for MKK to use water resources from the Oscco Chachi River and Maticuyoc Cucho spring for the purpose of mining exploration studies until 31 December, 2012.

- Authorization from the Community of Ollachea to use the land covered by the Ollachea Concessions for exploration activities for a term of five years from 25 November, 2007. However, on 30 May, 2012 it was extended for a period of 30 years

- Authorization from the MEM to carry out exploration activities outlined in MKK's Semi Detailed Environmental Assessment (SEA) of the Ollachea Project approved in 2008 with subsequent modifications approved in 2010 and June 2011.

For construction and operation of the mine, plant and other surface infrastructure MKK will require

- an approved EIA, compilation of which is currently in progress,

- a mine closure plan,

- an approved mine plan,

- a beneficiation concession,
- permits for water use, process and drainage water discharge,
- permits for use of explosives and powder magazines,
- permits for storage and use of chemical reagents,
- permits for storage and direct use of hydrocarbons (diesel, kerosene), and
- construction permits for the facilities.

20.5 Considerations of Social and Community Impacts

MKK has conducted a continuous programme of community awareness workshops and communications, and worked closely with the Community of Ollachea since it entered into agreement to acquire the property from Rio Tinto in 2006. The company's cooperation in formalizing illegal mining on the property and its surface rights agreement with the Community of Ollachea are part of a plan to incorporate to the maximum possible the community in the advancement and future operation of the Project.

One issue that precisely sets this project apart from others in the region is the fact that the surface rights agreement between MKK and the Ollachea community is coupled with a strategic social management plan that involves the different interest groups around the project. Among them are the Ollachea farming and artisanal mining communities.

20.6 Comments on Section 20

MKK has received permits to continue to operate exploration activities on the property and to excavate an exploration adit to provide underground access to the mine for core drilling.

Given the current permitting and community agreement status for the project, environmental, archaeological and social baseline work carried out to date, and the well-established permitting process in Peru, there are no currently known social, environmental or archaeological issues that could materially impact MKK's ability to extract the Mineral Reserves on the Property.

MKK is not required to post performance or reclamation bonds.

There is an expectation that there will be environmental liabilities associated with artisanal mining activities, decline and drill pads. MKK has a mitigation program in place, which consists of regular water quality determinations both up- and down-stream of the mine to monitor for possible contamination related to mining activities.

Additional permits will be required to support mine development and operations. MIRL has successfully obtained these permits for its Corihuarmi operation, and MKK expects that it is a reasonable assumption that the company can obtain the appropriate permits for operations at Ollachea.

21 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

The Ollachea FS capital cost estimate consists of estimates of direct and indirect capital costs for the underground mine and paste backfill system, the mineral process plant, auxiliary buildings and surface infrastructure, including electrical power supply, camp site and TSF.

Capital costs for the underground mine, including the portion of the paste fill system installed underground, were estimated by Coffey Mining. Capital cost estimates for the remaining items, including all surface infrastructure, TSF and process plant, were estimated by AMEC. Estimates have been combined for the purpose of developing an integrated project capital cost estimate. The accuracy of this estimate is within -10/+15%.

21.1.1 Basis of Estimate

21.1.2 Process Plant and Infrastructure

The estimate for the process plant and infrastructure including the TSF and waste rock stockpile was prepared using project information contained in, but not limited to the following documents:

- Design Criteria
- Process Flow Diagram
- Equipment Lists
- Plant and infrastructure layouts
- General arrangements and 3D model
- Preliminary engineering drawings and sketches
- Electrical single line diagrams
- Geotechnical advise and report
- Topographic maps provided by MKK
- Project implementation plan and project schedule

The mechanical and electrical equipment costs were obtained from written quotations submitted by vendors. Estimates for minor equipment were based on AMEC in-house data for recently completed or quoted projects.

Engineering material take-offs (MTOs) were derived from 3D models, drawings and sketches. Local contractor rates were used to estimate costs. Recent historical quotes were used when rates were unavailable.

21.1.3 Mine Development

The estimate is based on an owner operator strategy with contractors used for specialised activities, such as raise boring, and for early project development.

The estimate is based on quotes from suppliers of equipment, material and consumables, contractors, estimates from recent other projects and data supplied by MKK and AMEC. Costs have been developed from first principles using the scheduled mine physicals for the Project as a base.

Mine capital costs are defined as those costs occurring prior to the commencement of stope production. Thereafter, costs are classified as sustaining capital costs and operating costs. First stope production has been scheduled to start in January 2015. Mine indirect costs have been split between development and production on a tonnage pro rata basis.

Capital development is considered as development that is used for LOM mine access and service. This includes, main access inclines and declines, all primary ventilation development (lateral and vertical), underground infrastructure (magazine, fuel/service bay, lunch room) and development required for dewatering.

21.1.4 Labour Costs

21.1.5 Process Plant and Infrastructure

Labour rates and productivities for each of the surface infrastructure construction areas were estimated from AMEC's database for Latin American mining projects. The labour rates assume that all contractor craft personnel will be Peruvian but that they will not be locally based.

21.1.6 Mine Development

Labour costs, inclusive of on-costs, used for the cost estimate were supplied by MKK; these were derived from a combination of Peruvian benchmarking and current labour costs at IRL's operating mine at Corihuarmi.

21.1.7 Summary of the Estimate

The total estimated cost of the overall project as detailed in this document is USD$223.3 million**.** The estimate base date is Q3 2012. This total has been compiled as shown in Table 2.1:

Table 21-1 Capital Cost Estimate Summary

INITIAL CAPEX	US$(M)
Mine	55.1
Process Plant	72.0
EPCM	18.6
Other Indirects (incl Contingency & Owners costs)	31.8
Total Capital Cost Estimate	**177.5**
SUSTAINING CAPITAL	
Mining Sustaining	38.3
Waste Dump Closure	2.0
TSF Closure	2.2
Process Plant Sustaining	3.2
Total Sustaining Capital Estimate (Life of Mine)	**45.7**
Project Total	**223.3**

21.1.8 Currency and Exchange rates

The estimate was developed in Q3 2012 price levels, in United States dollars. Foreign currencies are expressed in American dollars, based on foreign exchange rates provided by MKK as nominated in Table 21-2.

Table 21-2 Foreign Exchange Rates

Currency	Rate
US$/EUR	0.760
US$/CAD	0.990
US$/CHF	0.950
US$/AUD	0.970
US$/GBP	0.620
US$/PEN	2.650

21.1.9 Project Contingency

The value of 12% for contingency was calculated from a thorough risk and opportunity analysis. This contingency factor has been applied to the mining, process plant and infrastructure capital estimates.

21.1.10 Process Plant Capital Cost

21.1.10.1. Summary

Process plant capital costs summarised by area are shown in

Table 21-3.

Table 21-3 Process Plant Capital Cost by Area

Area	Title	US$(M)
2000	Site Development	3.2
2100	Plant Area Preparation	0.3
2200	On-Site Roads	0.4
3000	Process Plant	8.2
3100	Power Supply Distribution & Control	2.1
3170	Services	3.4
3180	Diesel	0.3
3200	Crushing & Ore Receipt	9.7
3210	Grinding & Classification	7.8
3300	Gravity	1.4
3600	CIL – Carbon In Leaching	10.5
3650	HMPC CIL	0.8
3750	Acid Wash, Elution & Carbon Regen	2.0
3760	Electrowinning & Refining	1.1
3850	Cyanide Detox & Tailings Handling	6.0
3860	Paste Plant	3.0
3900	Reagents	1.7
5000	Ancillary Buildings	0.4
5100	Site Camp	2.0
5200	Plant & Mine Office Buildings	0.4
5300	Administration Office Buildings	0.2
5400	Maintenance Buildings – Mining Workshop	0.2
5500	Warehouses	0.5
5700	Facilities	0.1
6300	Tailings Dam Facility	5.3
6400	Tailings Dam Equipment	0.4
Total Process Plant Capex by Area		**72.0**

21.1.10.2. EPCM Services

EPCM costs were determined from first principles based on the schedule and organisation charts. The engineering component of the estimate was based on the deliverables list and using historical hours against these deliverables.

21.1.10.3. Freight

International sea freight and domestic freight has been included for all major mechanical equipment. International sea freight allowance has been included at 5% of the ex works cost and a further 3% of ex works costs has been included for inland freight. No allowance has been included for import duties. Note that a formal logistics study has not been prepared.

21.1.10.4. Process Plant and Infrastructure Sustaining Costs

Process plant and infrastructure sustaining costs include a yearly allowance, as well as the cost of the second primary crushing unit in year 3, and the replacement of light vehicles and some mobile equipment in year 5.

An allowance of $100,000 has been made for the first year of operation. For the years thereafter, an allowance of $200,000 per year has been made, except for the final year of operation where an allowance of zero sustaining cost has been made.

21.1.11 Mine Capital Costs

Mine capital cost has been split into seven categories. These are:

- Plant and equipment (incl. mobile equipment).
- Direct development waste.
- Direct development ore.
- Direct production.
- Indirect development waste.
- Indirect development ore.
- Indirect production.

shows a summary of the total mine capital cost split into the seven categories.

Table 21-4: Mine Capital Cost Summary

Description	US$M	%
Plant & Equipment (Inc Mobile Equipment)	21.6	39.1
Direct Development Waste	24.2	43.9
Direct Development Ore	0.7	1.2
Direct Production	2.1	3.9
Indirect Development Waste	6.2	11.2
Indirect Development Ore	0.4	0.7
Indirect Production	0.0	0.0
Total Mine Capital Cost	**55.1**	**100**

The cost contributors to the seven mine capital cost categories are discussed in the sections that follow.

21.1.11.1. Plant and Equipment (incl. mobile equipment)

Costs in this category include:

- Primary production mobile plant e.g. drill jumbos, LHDs, on-highway tipper trucks, production drill rigs, cable drill and install rigs.
- Major direct mobile plant e.g. scissors lifts, charge up vehicles, shotcrete sprayer and transmixers.
- Major direct fixed plant e.g. secondary fans, paste fill pump, diamond drills.
- Indirect mobile plant e.g. grader, UG stores truck, fuel truck, backfill services loader/IT, light vehicles.
- Indirect fixed plant e.g. primary fans, air compressor, pumps, substations and electrical boxes, communications, lighting, escapeway ladders, refuge chambers, underground service bay equipping, underground magazines, toilets, emergency response equipment and fire truck, underground lunch room, fixed plant PCM, technical services and production hardware and software (geology, mining, survey).

21.1.11.2. Direct Development Ore and Waste

Costs in this category include:

- All lateral owner operator development costs are inclusive of mobile and direct fixed plant operating and maintenance costs, operating and maintenance labour costs, materials and consumables costs (drill, blast, load, haul, ground support, services (including power, water, fuel), pumps and secondary fans).
- A development contractor will be used in 2013 as part of the project start up to assist with incline/decline development.

- Vertical development cost for primary, secondary and tertiary ventilation raises and escapeway raises. All raises are to be completed by a raisebore contractor

21.1.11.3. Direct Production

Costs in this category include:

- Stope delineation diamond drilling and assaying.

21.1.11.4. Indirect Development Ore and Waste

Costs in this category include:

- Service and ancillary vehicles operating and maintenance costs.
- Primary fans operating and maintenance costs.
- Air compressors operating and maintenance costs.
- Water pumps operating and maintenance costs.
- Electrical equipment operating and maintenance costs.
- Light vehicles operating and maintenance costs.
- Mine production supervisory labour which includes mining superintendent, underground manager and shift bosses.
- Equipment maintenance supervisory labour, electrical and mechanical.
- Technical services labour which includes technical services manager, mining engineers (drill and blast, ventilation, planning, paste), geologists (mine and resource) and geological technicians, geotechnical engineers and technicians, surveyors and surveyor assistants, draftsperson, grade control QAQC, surface core yard staff.
- Mining administration labour which includes trainers for mine operators.
- Other labour which includes general manual labour.
- Fixed costs which included technical services consultants and technical services training and conferences, software maintenance and core yard and geological consumables.

21.1.11.5. Indirect Production

There is no capital costs associated with this category.

21.1.11.6. Mine Sustaining Capital

Mine sustaining capital cost has been split into three categories. These are:

- Plant and equipment (incl. mobile equipment).
- Direct development.
- Indirect development.

Table 21-5 shows a summary of the total mine sustaining capital cost split into the three categories.

Table 21-5: Mine Sustaining Capital Cost Summary

Description	US$M	%
Plant & Equipment (Inc Mobile Equipment)	31.3	81.6
Direct Development	5.9	15.4
Indirect Development	1.1	3.0
Total Mine Sustaining Capital Cost	**38.3**	**100**

21.1.11.7. Plant and Equipment (incl. mobile equipment)

Costs in this category include:

- Replacement equipment for primary mobile plant e.g. drill jumbos, LHDs, on-highway tipper trucks, production drill rigs, cable drill and install rigs based on industry standard useful working life.
- Replacement equipment for direct mobile plant e.g. scissors lifts, charge up vehicles, shotcrete sprayer and transmixers based on industry standard useful working life.
- Replacement equipment for indirect mobile plant e.g. grader, UG stores truck, fuel truck, backfill services loader/IT, light vehicles based on industry standard useful working life.
- Replacement equipment for major direct fixed plant e.g. paste fill pump based on industry standard useful working life.
- Replacement Indirect fixed plant e.g. air compressor , dirty water pumps, survey equipment, computers, toilets, emergency response equipment based on industry standard useful working life.

21.1.11.8. Direct Development

The primary cost contributors for this category are the same as outlined previously in this section.

21.1.11.9. Indirect Development

The primary cost contributors for this category are the same as outlined previously in this section.

21.1.12 Owner (Corporate) Capital Costs

An estimate of Owner's costs to support the Project from project commitment to plant commissioning has been provided by MKK based on their current head-office operating costs. These costs include Owner's staff, sub-consultants, marketing, royalties, insurances, etc. These Owner's costs cover both underground mine construction and construction of surface installations for the project. Owner's costs include general and administrative costs for project support from Lima, and project management and an Owner's office in the field, as well as baseline and environmental and permitting activities. A sum of $6,891,055 has been included in the estimate for these costs. Owner's costs will be incurred over seven quarters from Q2 2013 to Q4 2014.

21.1.13 Closure Cost

21.1.13.1. Mine and Process Plant

The closure cost is relatively low as the underground mine surface footprint is not significant. The closure cost for the mine and process plant will be covered by the salvage value of equipment within the mine and the process plant.

21.1.13.2. TSF and Waste Rock Dump

The closure cost of the TSF and waste rock dump allows for long term physical and geochemical stability of the area disturbed by this project. Allowance has been made for erosion protection, surface water control and restoration to the original land use to the extent possible.

21.2 Operating Cost Estimate

The operating cost estimate includes operating costs of the underground mine, the minerals processing plant, the TSF and general & administrative (G&A) costs for the integrated operation.

21.2.1 Basis of Estimate

21.2.1.1. Responsibilities

Operating costs for the underground mine, including the portion of the paste fill system installed underground, were estimated by Coffey Mining. Operating cost estimates for the remaining items, including all surface infrastructure, TSF and process plant, were estimated by AMEC.

Currency and Exchange Rates

All project operating costs are expressed in US$. Pricing information from vendors was generally provided in US$. Where pricing information was provided in other currencies, costs were converted to US$ using the project exchange rates presented in Table 21-2.

The estimate does not make allowances for variation in exchange rates or the escalation of unit operating costs (i.e. the costs are in constant US$ as of Q3 2012).

21.2.1.2. Accuracy

The intended level of accuracy of this operating cost estimate is ±15%. No contingency has been included in the estimate.

21.2.1.3. Exclusions

Unless stated otherwise, the estimate excludes taxes from all costs.

Other exclusions include:

- Exploration and Exploration Drilling.
- Marketing and Sales Cost (in economic analysis).
- Depreciation Amortization and Royalties.
- Interest Charges.
- Doré shipping costs and refining charges for the doré bars (included in economic analysis).
- Enterprise fees, licenses and water use.
- Activities covered by the sustaining capital and closure/rehabilitation provisions.

21.2.2 Mine Operating Costs

Mine operating costs have been split into four categories. These are:

- Direct development.
- Direct production.
- Indirect development.
- Indirect production.

Table 21-6 shows a summary of the mine operating cost split into the four categories.

Table 21-6 Mine Operating Cost Summary

Description	US$M	%	US$/t ore
Development Direct	62.9	28.8	6.74
Development Indirect	17.2	7.9	1.85
Production Direct	105.5	48.4	11.32
Production Indirect	32.6	14.9	3.49
Total Mine Operating Cost	**218.2**	**100**	**23.41**

The LOM operating cost for the mine is estimated to be US$ 23.41/t ore. The key cost contributors to the four mine operating cost categories are discussed in the sections that follow.

21.2.2.1. Direct Development

The primary cost contributors for this category are the same as outlined in Section 22. The LOM direct development cost for the operating mine is estimated to be US$ 6.74/t ore.

21.2.2.2. Direct Production

The primary cost contributors for this category are:

- mobile and direct fixed plant operating and maintenance costs.
- operating and maintenance labour costs.
- materials and consumables costs (stope delineation diamond drilling, drill, blast, load, haul, backfill, ground support, services (including power, water, and fuel), pumps and secondary fans).

The LOM direct production cost for the operating mine is estimated to be US$ 11.32/t ore.

21.2.2.3. Indirect Development and Production

The primary cost contributors for these two categories are the same as outlined in Section 22.

The LOM indirect development and production costs for the operating mine are estimated to be US$ 1.85/t ore and US$ 3.49/t ore respectively.

21.2.3 Process Plant Operating Costs

Process plant LOM operating costs are summarised, by category, in Table 21-7.

Table 21-7 LOM Process Plant Operating Cost by Category

	Total LOM US$M	Process Plant Costs LOM US$/t	LOM US$/oz
Operating Supplies			
Wear Parts	20.1	2.15	21.8
Reagents And Consumables			
Reagents (Process Plant)	42.2	4.53	45.9
Consumables	2.4	0.25	2.6
Services/Utilities			
Power (Average Demand)	25.7	2.76	27.9
Fuel - Diesel	7.5	0.80	8.1
Sodium Cyanide, NaCN	53.7	5.76	58.3
Others/Miscellaneous	2.0	0.22	2.2
Sub-total	153.6	16.48	166.8
Labour			
Processing Plant	11.3	1.22	12.3
Sub-total	11.3	1.22	12.3
Others			
Maintenance Supplies	7.9	0.86	8.7
Equipment and Vehicle Hire	3.8	0.41	4.1
Sub-total	11.7	1.27	12.8
	176.7	**18.96**	**191.9**

21.2.3.1. Operating Supplies

Operating supplies costs comprises the costs of reagent and consumables, and the cost of wear parts.

The unit costs of supplies were from quotations received from local and overseas suppliers and from an internal database of similar operations.

Consumption of reagents and consumables was derived from metallurgical testwork, feed grade and throughput.

Wear parts are parts which are replaced due to normal wear and tear, such as liners. The quantity of crusher and mill liners consumed is based on information provided by equipment vendors.

Grinding media consumption was calculated from the Bond Abrasion index of the Ollachea ore types (derived from testwork).

21.2.3.2. Fuel (Diesel)

The cost of fuel covers diesel consumption for equipment, light vehicles and plant mobile fleet and is based on a diesel price of US$1.1/L.

Fuel consumption for process plant equipment of 0.61 L/t, was based on average fuel consumption rates and 5 strips of 6 t carbon per week.

21.2.3.3. Services and Utilities

The main component of the cost of services and utilities was the power cost.

Power demand for the various plant areas was calculated using the average continuous power demand derived from the Mechanical Equipment List.

A unit cost for electricity of US$0.067/kWh was used in the estimate.

21.2.3.4. Maintenance Supplies

The cost of maintenance materials was derived as a percentage of the mechanical equipment cost. This factor is based on AMEC's in-house database of operating plants. These costs allow for the maintenance of all mechanical equipment, pipes, liners, chutes, motors, electrical components and valves.

21.2.3.5. Miscellaneous

An allowance for miscellaneous plant costs was included to account for costs such as internal/external laboratory assays, elution heating oil, light vehicles and mobile equipment running costs (spares and consumables).

21.2.4 Tailings Storage Facilities (TSF) Operating Costs

Operating costs for the Ollachea TSF were estimated based on the FS tailings production of 5.5 Mt and a dry stack facility which will be located at Cuncurchaca, located approximately 5.8 km from the process plant.

The cost to operate the TSF includes labour, fuel (diesel), maintenance supplies, consumables (erosion covers, raincoats), equipment (TSF monitoring and surveillance), road and highway maintenance, equipment and vehicle hire, and water management.

21.2.5 General and Administration Operating Costs

General and administration (G&A) operating costs are presented in Table 21-8. These costs were derived from a staffing list for administrative personnel, by benchmarking to similar operations, and current operational data.

Table 21-8 General and Administration Operating Costs

	G & A Costs

	Total LOM US$	LOM US$/y	LOM US$/t ore
G&A Labour	7.2	830,000	0.78
G&A Fuel (Diesel)	0.5	50,000	0.05
G&A Maintenance Supplies	1.4	160,000	0.15
G&A Power	0.1	10,000	0.01
Investment in Sustainable Social Responsibility Programs	3.3	380,000	0.36
Third Party Services (Catering/Janitorial Contract)	6.6	760,000	0.71
External Assays	0.2	20,000	0.02
External Consulting And Software	0.3	30,000	0.03
IT And Communications	0.5	50,000	0.05
Safety/Protective Clothing (EPP)	0.4	40,000	0.04
Postage, Courier And Light Freight	0.0	0	0.00
Office/Computer Supplies/Maintenance/Supplies	0.2	30,000	0.02
Medical Assistance/First Aid	0.4	40,000	0.04
Travel/Accommodation/Camp	2.0	230,000	0.21
Access And Internal Roads Maintenance	0.6	70,000	0.06
G&A Mobile Fleet (Light Vehicles Maintenance)	0.6	70,000	0.07
Insurance/Legal Service	8.6	990,000	0.93
Recruitment And Training	0.5	60,000	0.05
Environmental Monitoring/Reporting/Programs/Management	0.5	60,000	0.06
Site Security (External)	4.1	470,000	0.44
Procurement And Importation Costs	0.4	40,000	0.04
Corporate Overheads	1.3	150,000	0.14
Mining Lease	0.8	90,000	0.08
Total	40.5	4,630,000	4.34

21.2.5.1. Investment in Sustainable Social Responsibility Programs

The estimate includes the following allowances for sustainable social responsibility programs for Ollachea:

- Payment for the use of surface land
- Sustainable development
- Health programs
- Training and supply of goods and services
- Education
- Agriculture
- Others.

21.2.6 Operating Cost Overall

21.2.6.1. Staffing List

An operations staffing list was developed to assist in scaling the camp site and other infrastructure, in capital cost estimation and in operating cost estimation.

At peak operation, the workforce will comprise 364 operations staff. Roughly two-thirds will be on site at any time.

Roster

Commissioning of the processing plant is planned to commence in January 2015. During the ramp-up in mine production rate, the plant will be operated on a campaign basis of two panels - operating the plant for 20 days followed by 10 days off. When in full production, a third panel will allow for continuous operation. Operations personnel will work for 14 days on at 12 hours per day followed by 7 days off.

Geographical Distribution of Staff

All staff employed for the Ollachea project will be Peruvian, with approximately 25% of workforce based in the local area of Ollachea. Non-local staff will be based in Juliaca or elsewhere in Peru. The geographical distribution of staff is summarised in Table 21-9.

Table 21-9 Geographic Distribution of Staff

Location	% of Workforce
Ollachea (Local)	25
Juliaca	45
Elsewhere in Peru	30

The proportion of local staff will likely increase over the life of the mine, with fewer local employees at the beginning of the project and increasing proportions of local workers as they are trained and move through the work force.

Remuneration

Annual remuneration rates for personnel were based on skill levels and responsibilities. Salaries, which were supplied by MKK, were benchmarked against salaries supplied by a survey of national salaries in Peru (Compensation Management Tool (Deloitte, 2010)) and against salaries prevailing at other local mining operations, as well as prevailing rates at MIRL's operation at Corihuarmi. Remuneration rates were provided in Peruvian soles which were then converted to US$ at the project exchange rates.

Annual remuneration rates include two additional months of gratuity.

Burden

A burden rate of 45.9% for mine workers and 28.5% for surface workers was applied to annual remuneration rates (14 months of salary), to determine the total cost of each employee.

21.2.7 Consolidated Operating Cost Schedule

A consolidated operating cost schedule for the Project is shown in Table 21-10. Mine operating costs average US$23.4/t ore processed (includes backfill). Plant operating costs total US$21.5/t ore processed (include tailings disposal), and G&A costs average US$4.3/t ore processed. Total site operating costs are US$ 49.2/t ore or US$499/oz of gold.

Table 21-10 Mine Operating Costs - Average

Operating Cost		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	Total LOM
Process Plant														
Supplies	US$M			11.8	15.5	18.4	18.5	18.4	18.6	18.7	18.5	12.2	3.6	153.6
Labour	US$M			1.0	1.2	1.3	1.3	1.3	1.3	1.3	1.3	1.0	0.6	11.3
Maintenance supplies and Misc.	US$M			1.0	1.2	1.3	1.3	1.3	1.3	1.3	1.3	1.0	0.7	11.8
Total Process Plant	**US$M**			**13.3**	**17.9**	**21.0**	**21.1**	**21.0**	**21.2**	**21.3**	**21.1**	**14.2**	**4.8**	**176.7**
TSF	**US$M**			**1.9**	**2.4**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.7**	**2.0**	**1.0**	**23.3**
G&A	**US$M**			**4.1**	**4.3**	**4.5**	**4.5**	**4.5**	**4.4**	**4.3**	**4.0**	**3.8**	**2.1**	**40.5**
Mining	US$M													
Fuel	US$M			1.6	1.9	2.4	2.4	2.4	2.3	2.2	1.9	1.6	0.5	19.2
Explosives	US$M			2.3	2.6	3.0	3.0	2.8	2.5	1.5	1.0	0.9	0.2	19.8
Maintenance Supplies	US$M			3.2	3.9	5.0	5.1	5.0	4.4	3.9	3.5	2.8	1.0	37.9
Labour	US$M			4.7	5.0	5.6	5.7	5.5	5.3	4.1	3.9	3.1	1.1	44.0
Power	US$M			2.1	2.3	2.5	2.5	2.5	2.4	1.8	1.4	1.2	0.5	19.2
Consumables	US$M			6.1	7.5	9.5	8.7	8.8	8.9	8.3	5.4	4.3	1.2	68.7
Other	US$M			1.1	1.1	1.2	1.3	1.2	1.2	0.9	0.7	0.5	0.2	9.4
Total Mining	**US$M**			**21.1**	**24.4**	**29.1**	**28.7**	**28.3**	**26.9**	**22.7**	**17.8**	**14.4**	**4.6**	**218.2**
TOTAL	US$M			**40.4**	**49**	**57.3**	**57**	**56.5**	**55.2**	**51**	**45.6**	**34.4**	**12.5**	**458.7**

22 ECONOMIC ANALYSIS

22.1 Methodology Used

A financial evaluation of the Project was undertaken using the discounted cash flow analysis approach. Cash flows were projected for LOM, which includes construction, operation and closure phases. The cash inflows were based on projected revenues for the LOM. The projected cash outflows, such as capital costs, operating costs, royalties and taxes; were subtracted from the cash inflows to estimate the net cash flows. A financial model (Model) was constructed on a monthly basis to estimate the net cash flows (NCF) over the LOM. The NCF were summarised on an annual basis. The cash inflows and outflows were assumed to be in constant 3rd quarter 2012 US dollar basis.

The Project was evaluated on a 100% project stand-alone and 100% equity-financed basis. The financial results, including Net Present Value ("NPV") and Internal Rate of Return ("IRR") do not take past expenditures into account; these are considered to be sunk costs. The analysis was done on a forward-looking basis from commencement of production commencing in January 2013, with the exception of the sunk costs to date, which were taken into account for tax calculations as an allowable deduction. Any other expenditure after 31 December 2012 not related to the Project construction has not been included.

The results of the economic analysis represent forward-looking information as defined under Canadian Securities Law. The results depend on inputs that are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Factors that could cause such differences include, but are not limited to:

- commodity prices
- exchange rate
- geotechnical assumptions in particular that the shear zone in the centre of the deposit will not significantly adversely affect extraction from neighbouring stopes
- ability to meet the mine plan as proposed
- assumptions that the dilution percentages assumed can be achieved in a production setting
- metallurgical recovery assumptions
- estimates as to projected reagent consumption

- assumptions that the proposed process route will adequately be able to account for the presence of organic carbon and refractory minerals
- permitting and construction timeline assumptions
- assumptions that land can be acquired to erect the powerline
- continued support from the local community.

22.2 Financial Model Parameters

The inputs and assumptions that form the basis of the Model include metal prices, mining schedule, mining inventory, processing throughputs, metallurgical recoveries, realisation costs, operating costs, capital costs, royalties and taxation parameters.

The base case gold price used in the financial evaluation was US$1,300/oz. The US$/PEN exchange rate used was 2.65.

22.2.1 Mineral Resource, Mineral Reserve, and Mine Life

Mineral Resources for the Ollachea FS are discussed in Section 14 of this report and total 10.6 Mt of Indicated Mineral Resources grading 4.0 g/t Au and containing 1.4 Moz of gold. Mineral Resources are inclusive of Mineral Reserves.

Mineral Reserves for the Project are discussed in Section 15 of this report and total 9.3 Mt of Probable Mineral Reserves at an average grade of 3.4 g/t Au containing 1.0 Moz of gold. The Mineral Reserves are split between ore (+2 g/t Au) and low grade development ore (+1 g/t to 2 g/t Au and are summarised in Table 22-1.

Table 22-1 Probable Mineral Reserves

Ore type	Tonnage (kt)	Grade Au (g/t)	Contained Au (koz)
Ore (+2 g/t Au)	8,730	3.50	983
Low Grade Development Ore (+1 g/t to 2 g/t Au)	590	1.50	28
Probable Mineral Reserves	**9,320**	**3.38**	**1,011**

Stope ore production will commence in January 2015 and commissioning of the process plant will commence in the same month. The mine production rate will ramp-up over a period of approximately 18 months. The life of mine is estimated to be approximately 9.6 years, with a ramp-down in mine production rate in the final approximately 2.5 years of the mine life.

22.2.2 Metallurgical Recoveries

The average LOM metallurgical gold recovery is 91.0%. Metallurgical recovery is estimated as function of tailings residue gold grade with head grade; three different metallurgical gold recovery formulas were developed for the different zone types.

Metallurgical Recovery (%) = (AuH – AuR)/AuH * 100

Where AuH is the head gold grade and AuR is the tailings residual gold grade.

- Zone 1&2 Recovery (%) = (AuH – (0.1713*AuH/4.6072+0.08000))/AuH*100
- Zone 3&4 Recovery (%) = (AuH – (0.3201AuH/4.6072+0.05388))/AuH*100
- Zone 5&6 Recovery (%) = (AuH – (0.1801*AuH/4.6072+0.31700))/AuH*100

22.2.3 Smelting and Refining Terms

The gold content of the doré produced at Ollachea is estimated to be 85%.

Estimated refining charges and payable metal are shown in Table 22-2. These are based on an analysis of existing contracts and market practice.

Table 22-2 Refinery Costs

Charge	Unit	Cost
Refining charge	US$ per oz doré	1.10
Payable	% Au content	99.9%

22.2.4 Metal Prices

A gold price of US$1,300 per ounce of gold is used for the base case in the model.

22.2.5 Operating Costs

The Project operating costs are presented in Section 21. The total operating costs estimated over the LOM are US$458.7 M. A breakdown of the LOM operating costs is presented in Table 22-3.

Table 22-3 Operating Costs

Parameter	Cost US$M	Cost US$/t ore
Mining		
Direct Development	62.9	6.7
Direct Production	105.5	11.3
Indirect Development	17.2	1.9
Indirect Production	32.6	3.5

Total Mining Costs	**218.2**	**23.4**
Processing		
Operating Supplies	153.6	16.5
Maintenance Supplies & Other	11.7	1.3
Labour	11.3	1.2
TSD and Tailings Handling	23.3	2.5
Total Processing Costs	**200.0**	**21.5**
General and Administration (G&A) Cost	40.5	4.3
Total Operating Costs	**458.7**	**49.2**

Note:
1. Costs are estimated in 3Q 2012 US dollars.

22.2.6 Capital Costs

The Project capital costs are presented in Section21. The total capital costs estimated over the LOM from commencement of construction is US$223.3 M. A breakdown of the LOM capital costs is presented in Table 22-4.

Table 22-4 Capital Costs

Parameter	Cost US$M
Pre-production Capital Cost	
Mining Equipment	21.6
Mining Development	33.6
Site Development	3.9
Process Plant	58.4
Ancillary Buildings	3.9
Tailings System	5.7
Total Direct Costs	**127.2**
Indirect Costs	23.4
Indirect costs – Owners Cost	8.3
Total Indirect Costs	**31.7**
Contingency – Mining	6.4
Contingency – Directs & Indirects	13.8
Total Contingency	**20.2**
Total Pre-production Capital Cost	**177.9**
Sustaining Capital Cost	

Parameter	Cost US$M
Mining Equipment	31.3
Mine Development	7.0
Plant & Other	3.2
Total Sustaining Capital Cost	**41.6**
Closure Capital Cost (Net)	**4.2**
Total Capital Costs	**223.3**

Note:
Costs are estimated in 3Q 2012 US dollars.

22.2.7 Royalties and Special Mining Tax

Peru Government Royalty (MMR) and Special Mining Tax (SMT)

In September 2011, Peru enacted three different tax legislation bills to establish a Modified Mining Royalty (MMR), Special Mining Tax (SMT) and Special Mining Burden (SMB), with an effective date of 1 October 2011. These three levies are in addition to the existing local country corporate income tax that applies to mining companies.

In the case of MKK, it is assumed that the Company will be subject to MMR and SMT. The following summarises the MMR and SMT:

- Modified Mining Royalty: The MMR applies on the companies´ operating income and is payable on a quarterly basis. An “operating income” to “mining operating revenue” measure (operating profit margin) is calculated each quarter and, depending on operating margin, the royalty rate increases as the operating margin increases. Marginal rates range from 1% for operating profit margins between 0% and 10% to 12% for operating profit margins greater than 80%. The system has been designed to provide both a minimum royalty and an additional amount based on the profitability of each project. A mining company must always pay at least the minimum royalty rate of 1 % of sales, regardless of its profitability.

- Special Mining Tax (“SMT”): The SMT is a tax imposed in parallel with the MMR. The SMT is applied on operating mining income based on a sliding scale, with progressive marginal rates ranging from 2% for operating profit margins between 0% and 10% to 8.4% for operating profit margins above 85%. The tax liability arises and becomes payable on a quarterly basis.

Third Party Royalty

A third party royalty of 1% net smelter revenue (NSR) is included in the model.

22.2.8 Working Capital

Working capital has been built into the Model by keeping track of the liquidities required and is calculated based on the difference between the current assets minus the current liabilities. The calculation is based on the assumption of a one month delay on account receivables and a one month delay on account payables.

22.2.9 Peruvian Taxes and Workers' Profit Participation

The details of the taxes and workers' profit participation included in the Model are set out below.

Income Tax

The Peru corporate income tax under the general tax regime is 30%

Tax losses can be carried forward for a period up to four years. It is estimated that as at 31 December 2012, MKK will have gross tax losses of approximately US$2.0 million from previous activities in Peru that can be used to offset income tax payable. Due to the carry forward rule these gross losses exclude any losses prior to 2010 year.

It is estimated that as at 31 December 2012, MKK will have capitalised prior expenditures on the Project of US$72.1 million that will be allowable deduction for tax purposes.

IGV – Value Added Tax

In Peru purchased goods and services are subject to IGV (Impuesto General a las Ventas) of 18%, which is a value added tax (VAT).

VAT paid by a company on the purchase of goods or services is generally allowed as a credit against the VAT collected from customers. The export of goods and services, such as export of gold, is not subject to VAT. A refund of VAT credit is generally allowed to the exporter. The refund is generally limited to 18% of the value of the exported goods or services. In applying for the refund, certain administrative requirements must be complied with.

There is a VAT early recovery system applicable to companies that enter into certain investment contracts or agreements with the Peruvian government. Under this regime, the VAT paid on the acquisition of new capital goods, as well as on the acquisition of immediate goods and services and construction contracts, can be recovered on a monthly basis through credit notes, without having to wait until the company makes VATable supplies. It is the intention of MKK to apply for registration under the early recovery system for the pre-operative stage of the Project on completion of the FS.

The Model assumes that 75% of the VAT incurred on the initial project capital costs is allowable for early recovery with the balance recovered once the Project comes into production. It is assume that it takes until October 2013 for the initial recovery and that the early recovery of VAT occurs thereafter on a quarterly basis.

It is estimated that, as at 31 December 2012, MKK will have incurred VAT of US$5.0 million that will be recoverable once the Project comes into production. This VAT incurred is not entitled to early recovery under the exploration early recovery system.

Once the Project is in production, VAT has been excluded from the operating estimates since the Project involves export of gold; it is assumed that the VAT will almost be immediately recoverable.

Financial Transaction Tax

The current Peruvian 0.005% proportional rate on banking operations in local or foreign currency (both debits and credits) is incorporated in the Model over the life of the Project.

Workers' Profit Participation

It is an obligation under Peru law that mining companies distribute 8% of their profit to workers in the form of workers' profit participation.

Workers' Profit Participation is deductible for income tax and is payable in the first quarter following the year end.

22.2.10 Closure Costs and Salvage Value

A closure cost of US$4.2 million is estimated for the waste stockpiles and tailings storage facilities. The closure requirements for the underground mine and the process plant areas are relatively minor; it has been assumed that the closure cost for the mine and process plant will be covered by the salvage value of equipment within the mine and the process plant

22.2.11 Financing

Costs associated with Project financing have not been considered in the Model.

22.2.12 Inflation

There is no provision made for inflation in the Model.

22.3 Financial Results

The cash inflows are based on projected revenues. The projected cash outflows, such as capital costs, operating costs and taxes; are subtracted from the cash inflows to estimate the net cash flows. A summary of the annual cash flows is presented in Table 22-5.

Table 22-5 Annual Cash Flows

Cash Flows		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	Total LOM
Inflows															
Net Revenue	US$M	-	-	82.0	101.0	144.7	154.0	152.6	163.2	152.2	121.2	98.8	22.9	-	1,192.6
Outflows - Operating															
Operating Costs	US$M	-	-	(40.3)	(49.0)	(57.2)	(57.0)	(56.4)	(55.2)	(50.9)	(45.7)	(34.3)	(12.5)	-	(458.7)
Royalties & Mining tax	US$M	-	-	(1.6)	(2.1)	(3.2)	(3.7)	(3.6)	(4.1)	(3.6)	(2.5)	(2.1)	(0.5)	-	(27.0)
Other taxes	US$M	-	-	(0.7)	(0.8)	(1.8)	(2.1)	(2.0)	(2.4)	(2.1)	(1.1)	(1.2)	(0.0)	-	(14.2)
Workers' Profit Participation	US$M	-	-	-	(1.6)	(2.1)	(4.3)	(4.8)	(4.7)	(5.4)	(4.8)	(2.9)	(2.8)	-	(33.5)
Income Tax	US$M	-	-	(3.8)	(4.8)	(12.8)	(17.7)	(17.2)	(19.9)	(17.4)	(10.5)	(11.0)	(0.7)	-	(115.7)
Total Outflows - Operating	**US$M**	**-**	**-**	**(46.4)**	**(58.3)**	**(77.1)**	**(84.8)**	**(84.1)**	**(86.4)**	**(79.5)**	**(64.6)**	**(51.5)**	**(16.5)**	**-**	**(649.0)**
Cash Flow from Operations	**US$M**	**-**	**-**	**35.6**	**42.7**	**67.7**	**69.3**	**68.5**	**76.8**	**72.8**	**56.7**	**47.3**	**6.4**	**-**	**543.6**
Outflows - Investing															
Initial Capital Costs	US$M	(67.8)	(109.8)	-	-	-	-	-	-	-	-	-	-	-	(177.5)
Initial Capital Costs – IGV	US$M	(7.3)	(4.1)	16.4	-	-	-	-	-	-	-	-	-	-	5.0
Sustaining Capital Costs	US$M	-	-	(9.2)	(7.4)	(2.9)	(5.0)	(6.8)	(6.3)	(2.5)	(1.4)	(0.1)	-	-	(41.5)
Closure Costs	US$M	-	-	-	-	-	-	-	-	-	-	-	(3.8)	(0.4)	(4.2)
Movement in Working Capital	US$M	-	-	(0.3)	(0.3)	(0.1)	0.0	0.0	(0.0)	(0.1)	0.4	(0.0)	0.4	-	-
Total Outflows – Investing	US$M	(75.1)	(113.9)	6.9	(7.7)	(3.1)	(4.9)	(6.8)	(6.4)	(2.6)	(1.0)	(0.1)	(3.4)	(0.4)	(218.3)
Net Cash Flow	**US$M**	**(75.1)**	**(113.9)**	**42.5**	**35.0**	**64.6**	**64.3**	**61.7**	**70.5**	**70.2**	**55.7**	**47.1**	**3.0**	**(0.4)**	**325.3**
Net Cash Flow before Tax	**US$M**	**(75.1)**	**(113.9)**	**47.0**	**42.2**	**81.2**	**88.4**	**85.8**	**97.5**	**95.1**	**72.1**	**62.2**	**6.5**	**(0.4)**	**488.7**

Note:
Costs are estimated in 3Q 2012 US dollars.
Net Revenue is gross revenue less realization costs (transport and refinery charges).
Net Cash Flow before tax is before Special Mining Tax, Worker's Profit Participation of 8% and Corporate Income Tax of 30%.

The Project was evaluated on a project stand-alone, 100% equity-financed basis. The base case gold price used in the financial analysis was US$1,300/oz. The base case gold price used is significantly lower than the current spot gold price of approximately US$1,700/oz and as such the financial evaluation was also undertaken using a gold price of US$1,600/oz to show the impact of a gold price closer to the current spot gold price. The NPV, IRR and payback period are presented in Table 22-6. The Project financial returns at a base case of NPV of 7% demonstrate that the Project is financially robust under the assumptions set out in this report.

Table 22-6 Summary of Ollachea Financial Results

Parameter	Unit	Base Gold Price US$1,300/oz	Upside Gold Price US$1,600/oz
Net Cash Flow before tax	US$ M	489	749
NPV @ 5% real (before tax)	US$ M	309	497
NPV @ 7% real (before tax)	US$ M	256	422
NPV @ 10% real (before tax)	US$ M	192	331
IRR (before tax)	%	29.2	40.2
Payback (before tax)	Years	3.2	2.5
Net Cash Flow (after tax)	US$ M	325	486
NPV @ 5% real (after tax)	US$ M	194	310
NPV @ 7% real (after tax)	US$ M	155	258
NPV @ 10% real (after tax)	US$ M	108	194
IRR (after tax)	%	22.1	30.2
Payback (after tax)	Years	3.7	3.0

Note:
NPVs as at commencement of construction.
NPVs are based on mid-period discounting.
Before tax is before Special Mining Tax, Workers' Participation Profit of 8% and Income Taxes of 30%.
Payback starts from the commencement of production.
The financial results are on 100% Project basis and exclude the agreement with the community for a 5% participation in MKK on commencement of production and Second Additional Payment payable by MKK and due to Rio Tinto in accordance with Mining Claim Transfer Agreement dated 23 February 2007.

A summary of the analysis of the LOM average unit cost of production on a per ounce basis is provided in Table 22-7.

Table 22-7 LOM average Unit of Production

Parameter	Unit	Cost
Mining	US$/oz	237
Processing	US$/oz	217
G&A	US$/oz	44
Total Site Cash Operating Costs	**US$/oz**	**499**
Realisation Costs	US$/oz	4
Royalties	US$/oz	46
Total Operating Costs	**US$/oz**	**549**

Note:
Costs are estimated in 3Q 2012 US dollars.
Per ounce based on payable gold.

22.4 Sensitivity Analysis

A sensitivity analysis was performed on the Base Case NPV, using a 7% discount rate, and IRR (Figure 22-1 and Figure 22-2). Positive and negative variations up to 15% in either direction were applied independently to each parameter: gold price, capital cost, operating cost and gold grade. The results demonstrated that the Project is most sensitive to variation in gold grade and gold price. Initial capital cost has the least impact on the sensitivity of the NPV @ 7%.

Figure 22-1 NPV at 7% real (post-tax) Sensitivity Chart

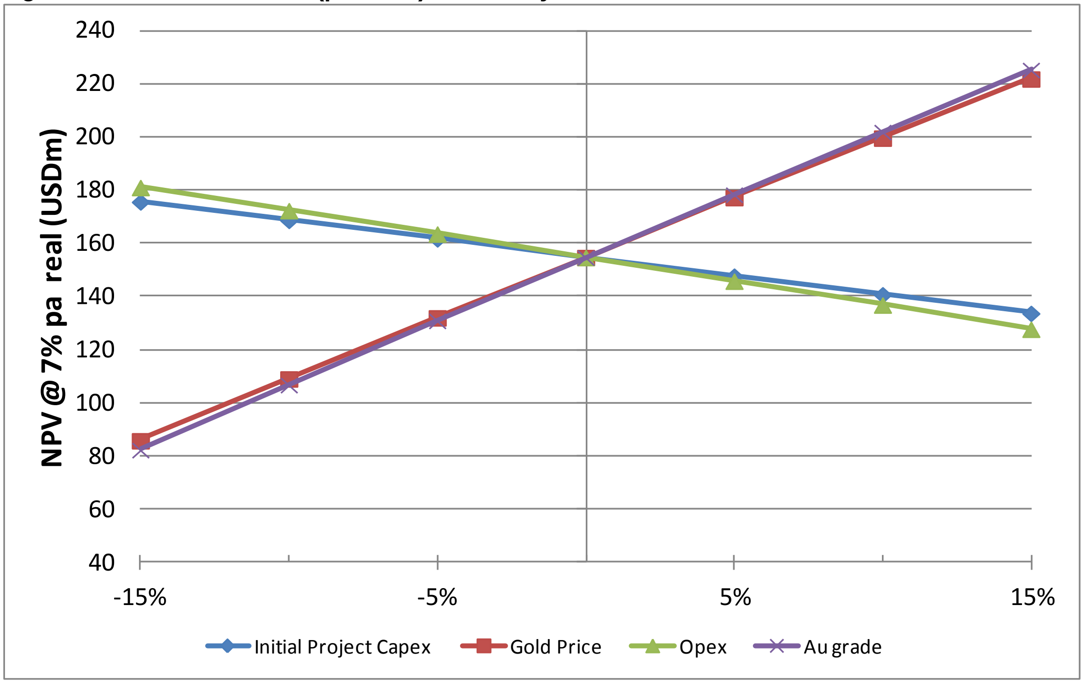
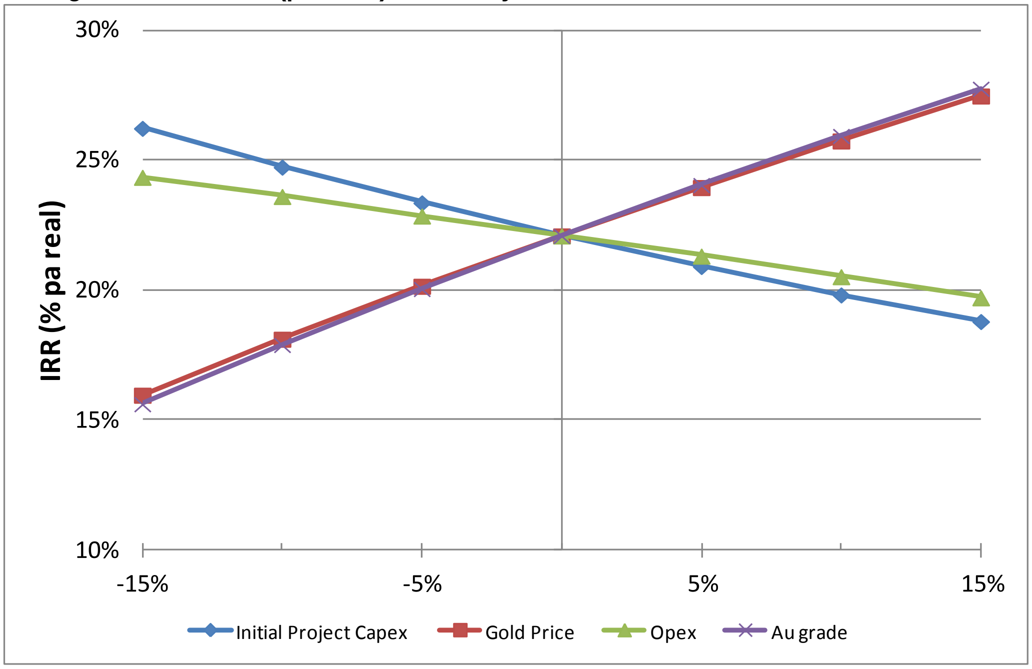


Figure 22-2 IRR real (post-tax) Sensitivity Chart



As with most gold projects, gold price is one of the most sensitive elements of the analysis. The breakeven point of the gold price for the NPV @ 7% real (after tax) is US$872/oz whereas the IRR real (after tax) reaches zero when the price of gold is US$679/oz. Table 22-8 shows the impact of different gold prices on Project returns.

Table 22-8 Sensitivity of Financial Returns versus gold price

Gold Price (US$/oz)	Pre-tax		Post-tax	
	NPV @ 7% (US$ M)	IRR (%)	NPV @ 7% (US$ M	IRR (%)
1,000	89	15.6%	48	12.2%
1,100	148	20.6%	84	15.8%
1,200	206	25.1%	120	19.1%
1,300	264	29.2%	155	22.1%
1,400	320	33.0%	189	24.9%
1,500	377	36.7%	224	27.6%
1,600	433	40.2%	258	30.2%
1,700	490	43.5%	292	32.6%
1,800	546	46.7%	325	35.0%
1,900	601	49.7%	359	37.3%
2,000	657	52.7%	392	39.5%

23 ADJACENT PROPERTIES

There are no properties adjacent to the Ollachea Property that are of relevance to this Report.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation

An Owner's team will be formed to deliver the project through the engagement of an EPCM contractor and specialist engineering consultants, suppliers and Peruvian construction contractors.

The project will be delivered through an incentivised contracting strategy, thereby reducing interface management and minimising duplication of roles. The team will have integrated systems and procedures. A specialist EPCM provider will be responsible for the delivery of the Process Plant and Associated Infrastructure area and will provide the underlying framework for all project systems and procedures.

The contracting strategy will align with in-country contractors' capabilities and local industry practices. The strategy will allow a competitive tendering environment whilst providing sufficient flexibility to maintain control of the project. Fixed price contracts are preferred; however, unit rates contracts, where used, would be structured with incentive clauses to encourage performance. Both practices support a reduced level of performance management by the Owner's team.

The Owner's team will deliver the mining development work covering all aspects, including areas such as mine design, mine fleet selection and procurement, assembly of the mining fleet, and operations etc. Ultimately, this part of the team will transfer through to the operations team.

All the process plant, tailings, waste rock, paste backfill plant, topsoil and water management facilities will be engineered by a single consultant to minimise interface management. Bulk earthworks and access roads across the site will be delivered by a civil construction plant fleet dry-hired and managed by the Owner's team. This fleet will commence construction early in the project and be retained through to operations to continue with subsequent construction stages of the tailings management facilities.

High voltage power supply from the existing power line running above the Ollachea project site will be delivered in two phases, the first stage comprising design, including government agreement and approvals, and land acquisition, and the second stage comprising construction and commissioning. The detailed design will be provided by the EPCM contractor or a specialist Peruvian engineering consultancy firm, and the construction by a specialist construction contractor.

The accommodation camp will be constructed close to the process plant; this will house all the project construction personnel including the owner's team, EPCM contractor, mining contractor and constructors.

24.2 Project Personnel

The overall philosophy will be to source personnel from Peru. If there are insufficient trained and experienced people available in Peru then personnel will be sourced from elsewhere within South America. It is anticipated that senior management will mainly be made up of expatriate persons with extensive experience in project delivery. The philosophy for project resourcing will be prioritised as follows:

- Local
- Regional
- Peru
- South American
- Canada/USA
- Australia.

24.3 PLANNING AND SCHEDULING

A project implementation schedule shows a total project duration of approximately 24 months (includes detail design, procurement, construction) to the start of commissioning.

Key milestone dates for the project are as follows:

- Project Execution commencement – 14 January 2013
- Permit approval for the extension of the exploration decline development – 1 March 2013
- Environmental approval – 1 April 2013
- Construction permit – 1 July 2013
- Commencement of construction – 1 July 2013
- Commencement of commissioning of process plant (first feed) – 31 December 2014
- Completion of Process Plant commissioning – 1 March 2015

The approximate duration of the project phases are as follows:

- Process Plant Detailed Engineering Design – 7 months

- Long Lead Procurement – 14 months
- Process Plant Construction – 18 months
- Commissioning of Process Plant – 3 months.

The schedule is constrained by a number of critical approvals, as follows:

- EIA Approval – 1 April 2013
- Construction Permit – 1 July 2013, no site works associated with the process plant can commence prior to this date.
- Procurement and delivery of Ball Mill – 62 weeks from placement of purchase order
- Procurement and delivery of Transformer – 62 weeks from placement of purchase order
- Procurement and delivery of Mining Fleet – 44 weeks from placement of purchase order

25 INTERPRETATION AND CONCLUSIONS

MKK has land tenure, surface rights agreements, permits for water supply and discharge and exploration permits required to carry out exploration activities including the development of an exploration access drive.

For construction and operation of the mine, plant and other surface infrastructure, MKK will require an approved EIA, permits for water use, process and drainage water discharge, use of explosives and powder magazines, chemical reagents, hydrocarbons (diesel, kerosene), and an exploitation permit for the Property.

Current arrangements for Project access, communication, power and water supply and labour are sufficient to carry out year-round exploration activities, and, with the necessary upgrades, can be reasonably expected to meet needs for Project development and operations.

25.1.1 Geology

The Ollachea deposit is an example of an orogenic, lode, slate-belt or mesothermal gold deposit. The deposit occurs in seven mineralized zones in the Minapampa Zone, having an open-ended strike length of 900 m and a width of approximately 200 m, and is hosted by slates with a pyrrhotite, pyrite, arsenopyrite and chalcopyrite sulphide assemblage.

The current Mineral Resource database for the Ollachea (Minapampa) Project consists of of 155 diamond drill holes totalling 60,306 m in length. Samples have been taken at 0.5 m to 5 m lengths and have an average length of 2 m (1.3 m within the mineralised zones). Samples have been prepared and analysed at Certimin Laboratories in Juliaca and Lima with blanks, standard reference materials, pulp duplicates, coarse crush reject duplicates and core twin samples to establish assaying accuracy and precision. Data pertaining to drilling, sampling, sample chain of custody, preparation and assaying of samples in the Mineral Resource Database are reasonable and can be used to support the estimation of Indicated and Inferred Mineral Resources.

The three-dimensional geological model constructed for the deposit serves to constrain gold mineralization to the genetic model and structural interpretation for the model given the continuity of geology and grade indicated by the diamond drilling and sampling in the current Mineral Resource database.

Mineral Resources have been estimated using ordinary kriging to interpolate parent block grades into 20 mE x 20 mN x 4 mRL from 2m composites, sub-blocks of 2 mE x 2 mN x 0.4 mRL were used to volumetrically represent the mineralised zones. The composite length, sub-block size, estimation method and estimation parameters for composite selection in estimation and control of extreme grades are reasonable considering the deposit type, proposed mining method, and geostatistical characteristics of the gold mineralization at Ollachea.

Mineral Resources for the Ollachea (Minapampa) Property, reported at a 2 g/t Au cut-off grade, consist of 10.6 Mt of Indicated Mineral Resources with an average grade of 4.0 g/t Au and 3.3 Mt of Inferred Mineral Resources with an average grade of 3.3 g/t Au. Mineral Resources were estimated by Doug Corley, MAIG, R.P. Geo, a Qualified Person under National Instrument 43-101 and have an effective date of 6 July, 2012. Mineral Resources are inclusive of Mineral Reserves.

A further Inferred Mineral Resource of 10.4 million tonnes grading 2.8 g/t containing 0.9 million ounces (above a 2.0 g/t Au cut-off) has been defined at Concurayoc (with an effective date of 31 August 2011). The Inferred mineral resource was announced in a press release by MIRL on the 7 September 2011 entitled "Substantial Maiden Mineral Resource Concurayoc Zone, Ollachea Project, Peru".

Concurayoc is located approximately 400 m westward along strike from the Minapampa Zone. There have been no mining studies carried out on the Concurayoc Inferred mineral resource, and therefore it does not have any demonstrated economic viability.

Exploration targets on the Property include the Concurayoc Zone, westward along strike from the Minapampa Zone and the down-dip extension of the Minapampa Zone.

25.1.2 Mining and Mineral Reserves

The Ollachea deposit will be mined from underground using long hole open stoping (LHOS) with paste fill. All stopes will be accessed longitudinally and extracted on a level by level retreat basis. The mine will have two main mining areas, east and west. The west area contains multiple stacked lodes and contains the majority of the Mineral Reserves. The east area has a limited number of lodes and extends over a greater vertical distance than the west, resulting in a lower concentration of contained ounces. Multiple mining areas are required to be mined simultaneously in the east to support project economic objectives. All mining will use a bottom up mining direction. The minimum horizontal mining width is 2.6 m, including dilution.

Major mine development will be accessed via an exploration incline that is currently being developed. This will be followed by the development of a second portal (2nd means of egress), ramp developments, ventilation raises, level accesses and haulage drifts. All mine levels will be 15 m apart. In general, ore and waste will be transported from the underground mine by truck via the exploration incline, with waste being deposited at a waste dump located in close vicinity to the lower portal entrance and ore being deposited at the process plant ROM pad.

The primary ventilation system consists of the exploration incline, other incline and decline drives, four surface raises (two return air raises and two fresh air raises), and an internal return air system and connecting drive that services the eastern part of the mine. Primary fans will be located on the two surface return air raises. The expected peak airflow at full production will be 700 m^3/s.

A hydrogeological numerical model was developed by AMEC to understand the behavior of the groundwater system in the Ollachea project area. Initial flow rates during project ramp up will be up to 80 m^3/h, which then increase to approximately 120 m^3/h during full production. Due to the nature of the planned mine development mine dewatering will be predominately gravity assisted. The water volumes estimated are not considered sufficiently large to present mine dewatering problems.

Coffey Mining completed the FS geotechnical assessment based on data from project geotechnical core logging and core photographs, structural studies, rock test results, and personal inspection. The *in situ* stress state was determined using the Deformation Rate Analysis (DRA) technique. The Modified Rock Mass Quality Index (Q') was utilised to characterise the *in situ* rock mass. Non-linear finite element numerical modelling was undertaken using the ABAQUS/FEA program to assess rock mass damage due to mine design and proposed extraction sequence. Modelling results were used to provide criterion for FS mine design and extraction sequence.

Ground support requirements for mine development, including large excavations, have been recommended based on the Q Index. Maximum stable stope spans were determined using the stability graph method, after Potvin (1988) and Nickson (1992). Empirical analysis was used to assess crown pillar stability. Two possible locations for vertical surface ventilation shafts were subjected to a shaft stability exercise to provide an indication of raise boring risk and provide LOM support recommendations.

Recommendations arising from the FS geotechnical assessment have been used as the basis of the FS mine design and extraction sequence, and mine operating and capital costs.

Face mapping information from the current development of the exploration incline, which is now traversing the orebody host rock, has not been considered in the FS geotechnical assessment due to timing of data availability. It is recommended that this data be assessed on an ongoing basis and used to update the geotechnical analysis completed as part of the FS. This should form an integral part of the proposed project development.

Due to the "non-visual" nature of the mineralisation, diamond drilling will form a significant part of the mine grade control program. Holes will be drilled from planned hangingwall drives prior to ore development on a minimum grid pattern of 15 m by 15 m. Ore drives will then be driven primarily on survey control, backed by face and wall channel sampling. An onsite laboratory is planned and has been designed to provide a 24 hour turnaround of samples.

The LHOS mining method and extraction sequence adopted for the Project is reliant on the use of paste fill. Process plant total tailings will be used to produce the paste fill. Approximately 42% 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 paste plant will be located at Minapampa with tailings transported from the process plant to paste plant by truck via the underground mine.

Suitability of process plant tailings for production of backfill was determined from the analysis of particle size distribution, mineralogy and rheological properties. It was found that the tailings are rather fine and suitable for paste fill.

Over 200 paste UCS tests were conducted to investigate the impact of cement type, and cement and tailings content on the UCS over a testing period of seven to 182 days. Representative testing was conducted using locally sourced cements, ore tailings and tailings processing water produced during FS process plant testwork.

The mine is planned to be owner operated by MKK. Specialist contractors will be used for selected activities such as raise boring. The mine is planned to operate 24 hours per day, 365 days per year and all mine personnel will work a 14 days on, 7 days off roster. Shifts will be of 12 hours duration.

The planned mine will require a standard, medium scale, underground mobile production fleet of jumbos, LHDs, trucks and drills. All mobile and fixed plant equipment will be purchased, operated and maintained by MKK. The fleet of primary mobile equipment units was calculated directly from equipment productivity rates and scheduled mine physicals from the final mine design.

Industry standard software was used to design and schedule the planned mine. Consideration was given to geological and geotechnical design constraints determined as part of the FS.

A single LOM mine design cut-off grade (COG) of 2.0 g/t Au was used for the FS. This was selected based on a simple break even grade analysis, company strategic objectives and previous study financial outcomes.

Economic value has been identified in processing low grade development ore below the project COG at different stages of the project life. The material to be processed is sourced from stope access development that traverses through Indicated Mineral Resources but has been diluted below the Project COG of 2.0 g/t Au. As the mining cost for this material will have already been expensed, it is economic to treat through the plant. A mill COG of 1.0 g/t Au was used for the low grade development ore in the FS.

Probable Mineral Reserves totalling 9.3 Mt grading 3.4 g/t Au and containing 1.0 million ounces of gold are declared based on the results of the FS. Mineral Reserves have an effective date of November 29, 2012. The reported Mineral Reserve has been compiled under the supervision of John Hearne, FAusIMM (CP), and an employee of Coffey Mining, and who is recognized as a Qualified Person for the purposes of National Instrument 43-101.

The Project is scheduled to start in July 2013 with first stope production in January 2015. Full mine production is reached in December 2016 and extends until December 2021. Mine production will stop at the end of July 2024, which gives a mine production tail duration of 31 months. Full mill production commences in May 2016 based on processing direct supply mine ore, direct supply mine low grade development ore and stockpiled low grade development ore. Full mill production continues until November 2022 with stockpiled low grade development ore used as supplementary feed source.

Financial analysis has shown the project to be robust at a base gold price of US$ 1300/oz using the assumptions outlined in this report. Sensitivity analysis indicates the project is most sensitive to gold price and gold grade. Stope dilution will have to be controlled through good operational management and technical application to ensure project economics are not adversely affected.

25.1.3 Metallurgy and Mineral Process Design

Results from the metallurgical test work completed on the six ore lenses, comprising the Ollachea ore body, have been utilised to formulate the basis for the process plant design. Test work concludes that the crushing and grinding of the ore, to a P_{80} of 106 µm, is required to sufficiently liberate the gold from the pyrrhotitic slate host.

Gravity concentration into two distinct components, a free gold concentrate and gold bearing sulphide concentrate, followed by CIL treatment of the respective tailings streams, which host the majority of the preg-robbing carbonaceous materials present within the ore, will achieve a Life of Mine (LOM) gold residue grade of 0.30g/t, translating to a LOM gold recovery of 91.0%.

Test work indicates the presence of a significant component of gravity recoverable (GRG) gold, with GRG gold contributing 39% to overall gold recovery. Sulphide associated gold, reporting to the continuous gravity concentration step, and leached within its own dedicated CIL circuit, contributes 32% to overall gold recovery. The preg-robbing nature of the ores, necessitate the implementation of a CIL leach on the tails of the GRG and sulphide circuit, to recover the remaining gold from tailings, which contributes 20% to the overall gold recovery. Tabling of the GRG concentrates, in conjunction with the adsorption and desorption of solubilised gold onto carbon, followed by electrowinning and refining will be utilised to produce gold doré on site.

Detoxification test work concludes that the implementation of an Air/SO_2/Cu^{2+} catalyst followed by a $ZnSO_4$ treatment circuit will produce a discharge slurry containing less than 1 mg/L total cyanide, which satisfies the legislated discharge requirements. The detoxified tailings are amenable to high rate thickening and will produce thickened slurry, containing 58% solids by weight. Filtration test work completed on the thickened tails slurry concluded that pressure filtration will produce a filter cake containing 14.8 % w/w moisture. This filter cake will be will be suitable for use to produce paste fill for the underground mine. When backfill is not required, the filter cake will be hauled to a dry-stack tailings storage facility.

25.1.4 Project Infrastructure

Access to the Ollachea Project is relatively straightforward from the Southern Interoceanic highway, although road construction to provide access to the mine, plant, camp and TSF will be required. The access road to the TSF from the Southern Interoceanic highway is approximately 1.8 km long.

Access roads to the Minapampa area for exploration activities pass through the town of Ollachea. A new access road bypassing the town of Ollachea is planned that will merge with the existing access road. MKK plan to construct this road in the first part of 2013.

Plant site infrastructure includes a power supply line and substation connecting to the national power grid on the San Gaban line that passes over the plant site. Auxiliary buildings for administration, mine surface shops, and security facilities will be constructed around the plant site. A permanent operations camp facility has been designed and will be located south of the Challouno area, in the vicinity of the lower portal and within 200 m of the Interoceanic Highway. The camp will have catering and accommodation capacity for approximately 275 persons.

The Ollachea FS mine schedule has surface waste rock disposal requirements of 2.2 Mt during the life of mine. A waste rock storage facility has been designed to permanently contain the life of mine waste rock. The waste rock facility will be located immediately south of the process plant, near the lower portal. Temporary waste storage will be required at the upper portal Minapampa area until the incline and decline tunnels meet. At that time, the waste rock temporarily stored at the upper portal will be transported to the permanent waste storage facility at the lower portal.

A preferred tailings storage facility site has been located and negotiations are underway to secure the remaining surface rights to the site, with 30% of the required surface rights already obtained. AMEC considers there is a reasonable expectation that these surface rights can be obtained. The TSF site is within 1.8 km of the plant site and can be accessed from the Interoceanic Highway. A dry stack tailings facility has been designed for 5.5 Mt of tailings consisting of a toe berm, underdrains, temporary coverage system, and coverage for final TSF closure.

To support cost estimation and development of general arrangements, design of other infrastructure - including a paste plant, surface warehouse, shops and administration buildings – has been progressed.

25.1.5 Operating and Capital Cost Estimates

The Ollachea project operating costs include fixed and variable costs for mine production, plant production, tailings management and general and administrative services for the operation. Mine operating costs average US$23.4/t ore processed (includes backfill). Plant operating costs average US$21.5/t ore processed (includes tailings disposal) and G&A

costs average US$4.3/t ore processed. Total site operating costs are US$ 49.2/t ore processed or US$499/oz of gold.

Capital costs include direct and indirect project capital for the mine, process plant and infrastructure and indirects amount to US $177.5 M, the sustaining capital that includes the mine and plant sustaining capital as well as the closure costs for the waste dump and the tailings facility amount to US$45.7 M. The total capital costs for the project are estimated at US $223.3M with the estimate base date of Q3 2012.

25.1.6 Financial Analysis

A financial evaluation of the Project was undertaken using the discounted cash flow analysis approach. Cash flows were projected for LOM, which includes construction, operation and closure phases. The cash inflows were based on projected revenues for the LOM. The Project has an after tax NPV of US $155 M at a discount rate of 7% and a gold price of $1300/oz. A sensitivity analysis considering positive and negative variations of up to 15% in either direction were applied independently to: gold price, capital cost, operating cost and gold grade. The results of the sensitivity analysis demonstrate that the project is most sensitive to variation in gold grade and gold price.

Risks and Opportunities

During the course of the Ollachea FS the following risks and opportunities have been identified:

25.1.7 Geology and Mineral Resources

Opportunities

Potential exists to increase the current Mineral Resource, both along strike and down dip from the current known Mineral Resource.

Potential exists for grade control drilling to delineate additional mineralisation adjacent to known mineralised zones.

Potential exists for Inferred Mineral Resources located in close vicinity to the declared Mineral Reserves to be upgraded to the Indicated resource category with more infill drilling data.

Risks

As the mineralisation is “non-visual”, grade control delineation is critical to spatially locating mineralised zones. Grade control drilling on a minimum of a 15m x 15m grid is planned to allow ore development drives to be located optimally.

Grade control drilling and geological mapping would be required to identify any possible north – south fault offsets to known mineralised zones; not identified by the dominantly north – south orientated mineral resource drilling. Grade control drilling should be angled to intersect any potential north – south striking structures.

25.1.8 Mining and Mineral Reserves

Opportunities

Parts of the FS mine development design have been identified that could be optimised resulting in a reduction of metres, waste tonnes and ultimately cost and development time. These include:

- Diamond drilling platforms designed to provide coverage in the western part of the mine could be reduced such that a full platform on every second level would provide coverage for two levels.
- The twin surface return air raises require a lower and upper connection to allow the air flow to be balanced. Early development of the upper connection would negate the requirement for each and every level to have two connections to these raises.

Section 15 identifies several parts of the FS mine design where there is potential to recover economic material through further investigation with the current available data or when new geological and geotechnical data becomes available. These parts include:

- The dimensions of the crown pillar were determined based on limited geotechnical data. With further investigation and data collection, including mapping of the near surface artisanal mines, crown pillar dimensions could possibly be reduced. Alternatively, full crown pillar extraction using high strength paste backfill should be investigated.
- Economic stopes located in close vicinity to the widest part of a shear zone that obliquely cuts through the orebody.
- Economic stopes located in close vicinity to primary ventilation raisebore holes.
- Use of an alternative extraction sequence to negate the use of sill pillar located in the western part of the mine.

The FS geotechnical assessment has provided recommendations on stable stope dimensions based on a limited data set (but appropriate for a FS). As mining commences, capturing and analysing actual stope performance compared to this initial assessment will provide invaluable data that can be used to optimise stable stope dimensions.

Development of a linear numerical modelling capability and calibrating models against observed conditions will also assist the stope performance assessment. Any increase in stope dimensions, especially along strike, will positively impact on mine production efficiencies.

Similarly, geotechnical data collected during the initial stages of mine development will enable ground support designs to be optimised to better suit the prevailing conditions. Any reductions in fibre reinforced shotcrete that has been recommended in the FS for capital development would provide positive cost benefits to the Project.

Cement is one of the largest cost contributors to the mine operating costs through its use in paste backfill. FS testwork has indicated that there is scope for potential reductions in cement consumption based on increasing tailings solid content. The addition of sand and / or chemical additives should also be investigated as this may also provide cost savings by reducing cement requirements.

Risks

If stope performance is continually less than expected on a major scale e.g. excessive hangingwall overbreak, stope turnaround times will be negatively impacted to the extent that planned production targets may not be readily achieved. It will also increase costs due to increased tonnage to be hauled and void to be filled. Excessive overbreak would also dilute mine head grade with a resultant decline in ounces produced. The development of early stopes, or trial stopes, is a means to mitigate this risk. Close monitoring and observation would allow stope dimensions and cable bolt support patterns to be optimised.

A diamond drilling grade control programme using a minimum grid spacing of 15 m by 15m is required to allow ore drives to be optimally located for effective and efficient stope extraction. A highly variable hanging wall over short distances will present challenges to the placement of these ore drives. This may require an even smaller spacing to be drilled which would increase production cost and delay ore drive development. Increasing diamond drilling resources would mitigate time delays; however, this will further increase production costs.

The ground conditions around the widest part of the oblique shear zone have been identified as a risk resulting in economic stopes being removed from the Mineral Reserves. An increase in the size of this area would have a negative impact on the Mineral Reserves. The closely spaced diamond drilling grade control programme will provide an early indication of the scale and extent of this risk.

Four surface raisebore holes are planned as part of the primary ventilation system. A shaft stability assessment completed as part of the FS on two potential areas highlighted the potential for excavation stability risk during boring and for LOM. To mitigate this risk specific diamond drill holes must be drilled early in each of the locations selected during the

FS mine design exercise with each hole geotechnically logged to an engineering standard. Shaft stability assessments should then be completed on each location, and if satisfactory, be used as part of the raise boring tender. An unsatisfactory outcome on any of the shaft stability assessments would require consideration to be given to alternative development techniques or a new location(s) to be identified. Any change in the location would require the FS mine design and schedule to be updated to reflect the location change(s) prior to implementation.

The FS mine development and production schedule is based on the continuous extension of the exploration incline. This requires the current development contract and necessary permits to be granted. Failure to achieve any of these in a timely manner will have a direct impact on the planned project ramp-up.

The mine has been designed and scheduled to produce ore at the rate of 1.1 Mt/a. Due to the nature of the mineralisation, grade distribution and geotechnical environment, multiple stopes are to be turned over on a regular basis to meet the production target. This requires a sustained high number of parallel development and production activities to be completed. Any issues encountered when completing these activities that leads to a reduction in operating efficiency will limit the mines ability to meet the desired production target with a likely reduction in ounces produced. Good operational management and technical application is required to mitigate this risk.

The project in considered technically and operationally complex due to the nature of the orebody, geotechnical environment and required high turnover of stopes to meet planned production targets. MKK and its parent IRL at the completion of the FS have limited in-house underground experience therefore will require a major recruitment drive in the short term to hire the necessary technical and operational personnel to implement and operate the underground mine at the designed capacity.

25.1.9 Metallurgy and Mineral Process Design

During the course of the Ollachea S and process plant design, the following potential risks and opportunities have been identified.

Opportunities

- Leaching at elevated temperature has the potential to partially mitigate the impact of carbonaceous material on gold extraction. This temperature affect has not been fully considered within the current process design. Marginal benefits may be expected as mill discharge slurry temperature will approach 40°C.
- Increasing the mass pull to the HMPG CIL circuit could potentially benefit final residue grade given the consistent leaching performance of this circuit.

- During the test work program, anecdotal evidence of increased viscosity, within the HMPG CIL, was observed. Consequently, the current design allows for the leach circuit to operate at 30% solids. The use of caustic soda, as a modifier, would potentially reduce the viscosity in this circuit, offering the potential for this circuit to operate at 38-40% solids which would allow optimisation of the HMPG CIL leach tank size.

- Ore Zones 5, 6 and 7 currently yield higher residue grades, and, consequently, lower extractions, than Ore Zones 1, 2 and 3. This conclusion is based upon limited data points, from the FS test work program. Further test work, focusing specifically on these three zones, is recommended with the aim of reducing reside grade through further optimisation of reagents and leach conditions.

- The LOM recovery provided is dependent upon the mine plan utilised for its derivation, and the proportion of respective ore zones, comprising the mine plan. A change in the proportion of ore from the respective ore zones would affect LOM recovery.

- Laboratory testwork cyanide consumption is normally much higher than what is seen in the full scale plant, due to the effect of fresh carbon addition. Cyanide consumption testwork to establish the impact of carbon on cyanide consumption and to quantify a realistic operating cyanide consumption is recommended

Risk

- Test work has indicated that leach extraction is variable, with different ore zones yielding variable final residue grades, with Ore Zones 1, 2, 3 and 4 yielding significantly better extraction results than Ore Zones 5 and 6. Consequently, this could lead to variable recovery in the process plant

- The LOM recovery provided is dependent upon the mine plan utilised for its derivation, and the proportion of respective ore zones, comprising the mine plan. A change in the proportion of ore from the respective ore zones would affect LOM recovery.

- Testwork has indicated that, to ensure a consistently low final residue grade, the recovery of gold to the HMPG circuit is to be maximised, with lower residue grades experienced when the HMPG CIL contribution to overall recovery increases. Consequently, this could lead to variable recovery and cyanide consumption

- Test work indicates that the comminution characteristics of the ores are variable. Similarly, the RWI:BWI differential indicates the potential for critical size material being rejected from the ball mill, as scats. Scats build-up would impact milling throughput and would require further processing through a pebble crusher. The FS design provides for as future pebble crushing installation, but is currently not included in the plant design

- The water quality of the feed to the water treatment facility requires verification. Currently, the water treatment facility has sufficient volumetric capacity and retains reagent dosing systems typical of a high density sludge neutralisation plant. Chemical requirements, with respect to reagent addition, and consequently operating cost may be negatively impacted, should water quality be poorer than assumed. However, as poor quality water from mine infiltration to date has not been observed, this is equally an opportunity for potential savings in operating costs.

26 RECOMMENDATIONS

It is recommended that the planning for the execution of the project begins in January 2013.

It is also recommended that additional confirmatory investigations are carried out in early 2013 to minimise project risk during the execution of the project. These recommendations are described under their sections below.

26.1 Geology and Mineral Resources

Samples for bulk density determination should be taken from mineralized zones, as an ongoing exercise. A target of 100 density determinations per zone (or more) should be taken to adequately characterize variability in the mineralized zone. Once a regular underground drilling program has begun, it would be recommended to take one density determination per sample collected, to develop a large enough database, so the bulk density can be estimated in a similar way as gold is. This is something to be developed as the grade control procedure forms and would form part of the operational expenditure. Estimated LOM cost to complete this task is US$200,000.

26.2 Mining and Mineral Reserves

Due to timing, and location, of available data no cognisance was taken of exploration incline face mapping during the course of the FS. This development is now located in orebody host rock and face mapping data is being captured and reported using a RMR classification system. FS geotechnical design was completed using the Q system and specific geotechnical design methodologies based exclusively on this system (Stope Stability Chart (Mathews-Potvin), Ground Support Selection Chart (Barton-Grimstad) and Raisebore Stability Analysis (McCracken-Stacey)). Coffey Mining therefore recommends that the Q system be adopted as the project standard for rock mass classification as this system is considered the most appropriate for designing and optimising this type of underground mine. Technical personnel currently onsite will be able to complete this at no additional cost.

To mitigate risk associated with the development of surface shafts using the raise boring technique diamond drill holes centred on the location of each of the four planned holes should be prioritised and planned for early completion. Estimated cost to complete this task is US$100,000.

Several opportunities have been identified with regard to FS mine design and schedule optimisation. These should be reviewed prior to project implementation to determine the scale and possible benefit of the opportunity, and whether the outcome would warrant a change to the base FS mine design and schedule. Estimated cost to complete this task is US$50,000.

The FS development and production schedule is reliant on the continuous extension of the exploration incline. Both permit approval and an extension to the current development contract must be expedited to adhere to the project implementation schedule. Estimated cost to complete this task is US$10,000.

For the FS no formal underground equipment tenders were issued. Budget quotes were sourced from OEM suppliers and agents that sell similar types of planned equipment. Formal underground equipment tenders require to be issued as part of the implementation phase of the project. Given the planned accelerated project implementation schedule underground equipment tenders are considered a priority activity. Estimated cost to complete this task is US$25,000.

26.3 Metallurgy and Mineral Process Design

The Ollachea FS metallurgical test work program has yielded sufficient information to develop a definitive metallurgical flow sheet, with quantifiable metallurgical outcomes, with respect to product and tailings quality. Recommendations for consideration include:

- Conducting additional cyanide detoxification test work to confirm the results achieved through inclusion of the $ZnSO_4$ step, which is critical to ensuring discharge criteria are met. Estimated cost to complete this task is US$20,000.

- Conducting additional test work on Ore Zone 5 to establish whether altered process parameters, with respect to blanking reagent addition, carbon or cyanide concentration, will result in improved residue grades. Estimated cost to complete this task is US$40,000.

- Obtain additional information on the quality of the water to be treated by the water treatment plant, and perform water treatment testwork to ensure the chemical considerations are adequate. Estimated cost to complete this task is US$20,000.

26.4 Project Infrastructure

The following recommendations are provided as relating to project infrastructure:

- Slope monitoring at the TSF Cuncurchaca site should be carried out throughout the life-time of the proposed facility. Local failures, mostly due to the lack of erosion protection, on the exposed cuts along the Interoceanic highway, are a regular occurrence. Although no visible signs of general land mass movement were identified during the geohazard evaluation of the area, two inclinometers were installed immediately downhill of the proposed TSF to monitor potential ground movement. It is recommended that monitoring is conducted at least twice a year or more frequently if movement is measured. Estimated LOM cost to complete this task is US$200,000.

- Additional geotechnical investigation consisting of test pits and borings is recommended along the proposed TSF access road. Geotechnical borings are recommended for the proposed bridge crossing the Cuncurchaca River. Estimated cost to complete this task is US$150,000.

- Additional studies are recommended to identify and characterize material borrow sources in the vicinity of the project. In particular, sources of low-permeability soil for use in caps for closure of waste rock facilities were not identified in the feasibility study. Estimated cost to complete this task is US$40,000.

- On-going monitoring of piezometers installed at the TSF, waste rock dump and process plant site is recommended to characterize seasonal fluctuations of the phreatic surface. Monthly monitoring is initially recommended. Based on fluctuations of readings, the monitoring frequency may be revised. Estimated LOM cost to complete this task is US$40,000.

- Geotechnical characterization of waste rock, including grain-size distribution, durability and anticipated weathering, should be evaluated to confirm the criteria assumed for the waste dump design. Estimated cost to complete this task is US$15,000.

- A geotechnical exploration consisting of additional borings and test pits is recommended for the Minapampa access road. As it is currently proposed, the road design could require significant cuts in an area where little geotechnical information is available to determine batter angles or the need for retention systems. Estimated cost to complete this task is US$250,000.

- Results from a hydrological study of the Ollachea River basin should be used to determine the safe elevation/distance of any facilities to be constructed in the vicinity of the river such as the campsite or any other potential structures. Estimated cost to complete this task is US$40,000.

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28 CERTIFICATES OF QUALIFIED PERSONS

CERTIFICATE OF QUALIFIED PERSON

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Coffey Mining Pty Ltd.
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I, Doug Corley, BSc (Hons), MAIG R.P. Geo., am employed as a Principal Resource Geologist with Coffey Mining Pty Ltd.

This certificate applies to the technical report titled "Ollachea Gold Project National Instrument 43-101 Technical Report on Feasibility Study" an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a Member of the Australian Institute of Geoscientists (MAIG) and a Registered Professional Geoscientist in the field of Mining (R.P.Geo (Mining)). I graduated from the James Cook University of North Queensland, Townville, QLD, Australia, with a Bachelor of Science degree (with Honours) in Geology in Geology in 1991.

I have practiced my profession continuously since 1991. I have been directly involved in the mineral resource estimation for the Ollachea (Minapampa zone) as used in this Technical Report.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Ollachea project site between the 21st and 22nd June 2010.

I am responsible for items 10-12 (excluding items 10.6 and 11.2), and item 14 of the Technical Report.

I am independent of Minera Kuri Kullu S.A. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Ollachea project since 2010, and have produced various mineral resource updates over the project since this time.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 19 December 2012

Signed Doug Corley, MAIG R.P. Geo (Mining)

Doug Corley MAIG R.P. Geo. (Mining)

CERTIFICATE OF QUALIFIED PERSON

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I, John Hearne, BEng (Mining), MBA, FAusIMM, CP(Mining), am employed as the Regional Manager, Western Australia, with Coffey Mining Pty Ltd.

This certificate applies to the technical report titled "Ollachea Gold Project National Instrument 43-101 Technical Report on Feasibility Study" with an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a Fellow of the Australasian Institute of Mining & Metallurgy (AusIMM) and a Chartered Professional. I graduated from the University of Sydney, Sydney, NSW, Australia, and hold a Bachelor of Engineering degree in Mining Engineering (1984).

I have practiced my profession continuously since 1984. Over the last 28 years, I have been directly involved in undertaking and managing all facets of mining projects from pre-feasibility studies to full operational management for both underground and open cut operations.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not personally visited the Ollachea Project. Other Coffey Mining professionals have made a number of site visits to the Ollachea Project over the last three years.

I am responsible for sections 15 and 16, excluding sections 16.5 and 16.8, of the technical report.

I am independent of Minera Kuri Kullu S.A. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Ollachea Project since 2010 and was responsible for the 2011 Mineral Reserve estimate.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 19 December 2012

Signed John Hearne, BEng, MBA, FAusIMM, CP(Mining)

John Hearne, BEng, MBA, FAusIMM, CP(Mining)

coffey mining
SPECIALISTS FROM BOARDROOM TO MINE FACE

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I, Vadim Louchnikov, BEng (Hons) (Mining), MEngSc (Geotechnical), FAusIMM, am employed as an Associate Mining Geotechnical Engineer with Coffey Mining Pty Ltd.

This certificate applies to the technical report titled “Ollachea Gold Project National Instrument 43-101 Technical Report on Feasibility Study” an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a Fellow Member of the Australasian Institute of Mining and Metallurgy (AusIMM). I graduated from the University of South Australia, Adelaide, Australia, with a Bachelor of Engineering degree (with Honours) in Mining Engineering in 2001. I obtained my Master’s degree in Geotechnical Engineering from the University of Adelaide, Australia, in 2004.

I have practiced my profession continuously since 2001, predominantly at underground operating mines in various geotechnical / backfill roles. I have been directly involved in the geotechnical and backfill assessments for the Ollachea Project as used in this Technical Report.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Ollachea project site between the 16th and 20th January 2012.

I am responsible for items 16-5 (Geotechnical) and 16-8 (Backfill) of the Technical Report.

I am independent of Minera Kuri Kullu S.A. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Ollachea project since 2011, and have produced a number of reports and memorandums over the project since this time.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 19 December 2012

Signed Vadim Louchnikov, FAusIMM

Vadim Louchnikov FAusIMM

CERTIFICATE OF QUALIFIED PERSON

I, Tim Miller, MAusIMM, FFinsia; am employed as Chief Financial Office and Company Secretary with Minera IRL Limited.

This certificate applies to the technical report titled “Ollachea Gold Project, Peru, National Instrument 43-101 Technical Report on Feasibility Study” an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a Member of Australasian Institute of Mining & Metallurgy (AusIMM) and a Fellow of Financial Services Institute of Australasia (Finsia). I graduated with a Bachelor of Applied Science (Chemistry) from Royal Melbourne Institute of Technology in 1988, a Graduate Diploma Applied Finance and Investment from the Securities Institute of Australia in 1993 and Master of Applied Finance from University of Melbourne in 2002.

I have practiced my profession for over 20 years. I have been directly involved in the Ollachea Gold Project since the date of acquisition by Minera IRL Limited’s subsidiary Minera Kuri Kullu S.A.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Ollachea project site in October 2010.

I am responsible for item 19 of the Technical Report.

I am not independent of Minera Kuri Kullu S.A. and Minera IRL Limited as independence is described by Section 1.5 of NI 43–101.

I have been involved with the commercial and financial areas of the Ollachea Project since the date of acquisition by Minera IRL Limited’s subsidiary Minera Kuri Kullu S.A. in 1997.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 19 December 2012

Signed Tim Miller, MAusIMM, FFinsia

Tim Miller, MAusIMM, FFinsia

CERTIFICATE OF QUALIFIED PERSON

I, Donald Angus McIver, Geologist, FAusIMM, FSEG, am employed as Vice President Exploration with Minera IRL SA.

This certificate applies to the technical report titled "Ollachea Gold Project, Peru, National Instrument 43-101 Technical Report on Feasibility Study" an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a Fellow of both the Australasian Institute of Mining & Metallurgy (FAusIMM), as well as the Society of Economic Geologists (FSEG). I graduated with a Bachelor of Science Degree (Geology and Chemistry) from the University of Port Elizabeth in 1986, a Bachelor of Science Honours Degree (Geology) in 1987 from the same University and a Master of Science Degree (Exploration and Economic Geology) in 1996 from Rhodes University located in Grahamstown.

I have practiced my profession for over 25 years. I have been directly involved in the exploration discovery of and all subsequent phases of the resource drilling on the Ollachea Gold Project, as well as aspects relating to Geotechnical and Hydro-geological studies and also Mine Design inputs into the project's Feasibility Study.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have regularly visited the Ollachea Project site since the initiation of Exploration activities on the Ollachea Project.

I am responsible for items 6 to 9 of the Technical Report.

I am not independent of Minera Kuri Kullu S.A. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Ollachea Project since its inception with Minera Kuri Kullu.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 19 December 2012

Signed Donald Angus McIver, FAusIMM, FSEG

Donald Angus McIver, FAusIMM, FSEG

CERTIFICATE OF QUALIFIED PERSON

I, Marius Phillips, P.Eng, AusIMM (CP), am employed as a Process Engineer with AMEC Australia Pty Ltd.

This certificate applies to the technical report titled “Ollachea Gold Project, Peru, National Instrument 43-101 Technical Report on Feasibility Study” an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a Member (CP) of the Australian Institute of Mining and Metallurgy and a Registered Professional Engineer of Queensland.

I graduated from The University of Johannesburg, South Africa with an Extractiev Metallurgical Degree in 1997.

I have practiced my profession for over 20 years experience in the minerals industry and have had diverse experience in Australian and International projects including feasibility studies, technology transfer, process optimisation, process engineering design, commissioning, operations and management of major process plants..

I have technical expertise and extensive experience in operations, design, optimisation and commissioning of gold, copper and platinum processing plants.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Ollachea project site from 13 to 22 January 2012. The purpose of this visit was for familiarisation purposes and to assess layout possibilities as well as view the drill core/ conduct sample selection.

I am responsible for sections 11.2, 13, 17, 21.2.3 and 26.3 of the technical report.

I am independent of Minera Kuri Kullu S.A. as independence is described by Section 1.5 of NI 43–101.

I have had no previous involvement with Ollachea .

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 December 2012

Signed Marius Phillips, CP

Marius Phillips, P.Eng, AusIMM (CP)

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CERTIFICATE OF QUALIFIED PERSON

I, Grahame Binks, BEng (Hons Met), MEngSc, AusIMM (CP), am employed as a Strategic Business Manager with AMEC Australia Pty Ltd.

This certificate applies to the technical report titled "Ollachea Gold Project, Peru, National Instrument 43-101 Technical Report on Feasibility Study" an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a Member (CP) of the Australian Institute of Mining and Metallurgy and a Registered Professional Engineer of Queensland, #08522.

I graduated from The University of Melbourne with a Bachelor of Engineering Degree (Honours Metallurgical) in 1983 and a Master of Engineering Science in 1985.

I have practiced my profession for over 26 years experience in the minerals industry and have had diverse experience in Australian and International mineral plants their development from concept to implementation and project assessments.

I have specialist experience in precious metals, copper, lead, zinc, nickel, tin and uranium and in a wide range of operating environments. I have worked for a number of major minerals companies, including CRA, Pasminco, Electrolytic Zinc of Australasia and Zinifex. My experience includes R&D project evaluation, due diligence studies and feasibility studies. I have been involved with a number of major projects including the Tujuh Bukit Copper Gold Project, Taysan Copper Gold Project, Seminco copper Project, Intex Mindoro Nickel Project, Pasminco Iron Residue Disposal project, Zinifex Two Stage Neutral Leach, QGC Soda Ash, BHP Billiton Tails Leach Upgrade, BHP Billiton Yeelirrie and BHP Billiton Olympic Dam Expansion. I have worked recently as a consulting Process Engineer and Study Manager and have consulted primarily in relation to the evaluation and engineering of copper, lead, zinc, tin, nickel and uranium projects in Vietnam, Philippines, Indonesia and Australia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Ollachea project site.

I am responsible for sections 1-5, 20-26 (excluding 21.2.3, 26.3) of the technical report.

I am independent of Minera Kuri Kullu S.A. as independence is described by Section 1.5 of NI 43–101.

I have had no previous involvement with Ollachea .

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 December 2012

Signed Grahame Binks, CP

Grahame Binks, BEng (Hons Met), MEngSc, AusIMM (CP)

CERTIFICATE OF QUALIFIED PERSON

I, Brett Byler, P.E., am employed as a Senior Geotechnical Engineer with AMEC Perú S.A.

This certificate applies to the technical report titled "Ollachea Gold Project NI 43-101 Technical Report on a Feasibility Study" an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a Registered Professional Engineer in the state of Colorado (#39291). I graduated from with a Bachelor of Science degree in Geological Engineering from the Colorado School of Mines in 1995 and a Master of Science degree in Civil Engineering from the University of Colorado at Boulder in 2003.

I have practiced my profession for 15 years. I have been directly involved in studies, design and construction of mining infrastructure.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I most recently visited the Ollachea project site on December 5 through 10, 2010.

I am responsible for Sections 10.6, 18 (excluding 18.4) and Section 26.4 of the Technical Report.

I am independent of Minera Kuri Kullu S.A. as independence is described by Section 1.5 of NI 43–101.

I have been previously involved with the Ollachea Gold Project between June 2010 and September 2011 acting as a "Qualified Person" on a previous NI 43-101 Technical Report concerning the property.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 19 December 2012

Signed Brett Byler, P.E.

Brett Byler, P.E.

CERTIFICATE OF QUALIFIED PERSON

I, Jim McCord, am employed as a Principal Hydrogeologist Water Resources with AMEC South America.

This certificate applies to the technical report titled "Ollachea Gold Project NI 43-101 Technical Report on a Feasibility Study" an effective date of 29 November 2012 and an issue date of 19 December 2012 (Technical Report).

I am a member National Groundwater Association. I graduated from B.S., Civil Engineering, Virginia Polytechnic Institute and State University, 1981, M.S., Hydrology, New Mexico Institute of Mining and Technology, 1986, Ph.D., Geoscience, Dissertation in Hydrology, New Mexico Institute of Mining and Technology, 1989

I have practiced my profession for 29 years since graduation. I have been directly involved in Hydrologic and Hydrogeologic data analysis and modeling for the project, as well as tailings facility seepage analysis.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have not visited the Ollachea Project.

I am responsible for Water Resources of the Technical Report.

I am independent of Minera Kuri Kullu S.A. as independence is described by Section 1.5 of NI 43–101.

I have had no previous involvement with the Ollachea project.

I have read NI 43–101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: 19 December 2012

Signed Jim McCord, P.E.

Jim McCord, P.E.

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