Minera Kuri Kullu S.A.

Ollachea Gold Project

Puno Region, Perú

NI 43-101 Technical Report on Pre-feasibility Study

Submitted by:
Doug Corley, MAIG
John Hearne, FAusIMM,
Michael Drozd, SME Registered Member
Brett Byler, P.E.
Chris Wright, P.Geo.

Effective Date: 17 July 2011
Project Number: 166729

coffey mining
SPECIALISTS FROM BOARDROOM TO MINE FACE

CERTIFICATE OF QUALIFIED PERSON

Doug Corley MAIG.
Coffey Mining Pty Ltd.
1162 Hay Street
West Perth, Western Australia, 6005
Tel: (61) 9324 8800
Fax: (61) 9324 8877

I, Doug Corley, do hereby certify that:

I am employed as an Associate Resource Geologist, with Coffey Mining Pty Ltd.

This certificate applies to the technical report entitled Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study, dated 1 September, 2011.

I am a Member of the Australasian Institute of Geoscientists (MAIG).

I graduated with a degree from the James Cook University, Townsville, Qld, Australia, and hold a Bachelor of Science degree (with Honours) in Geology (1991).

I have practiced my profession continuously since 1991.

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 visited the Ollachea Project between the 21st and 22nd June 2010.

I am responsible for all of Items 7 to 12 (Excluding Items 10.5, 10.6 and 11.4) and Item 14 of the Technical Report entitled Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study.

I am independent of Compañía Minera Kuri Kullu as independence is described by Section 1.4 of NI 43-101.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Doug Corley MAIG.

Dated: 1 September, 2011

coffey mining
SPECIALISTS FROM BOARDROOM TO MINE FACE

CERTIFICATE OF QUALIFIED PERSON

John Hearn FAusIMM (CP).
Coffey Mining Pty Ltd.
1162 Hay Street
West Perth, Western Australia, 6005
Tel: (61) 9324 8800
Fax: (61) 9324 8877

I, John Hearn, do hereby certify that:

I am employed as the Regional Manager, Western Australia, with Coffey Mining Pty Ltd.

This certificate applies to the technical report entitled "Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study", with effective date 17 July, 2011.

I am a Fellow of the Australasian Institute of Mining & Metallurgy (AusIMM) and a Chartered Professional.

I graduated with a degree 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.

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 mining professionals have made a number of site visits to the Ollachea Project over the last two years.

I am responsible for all of Items 15 and 16 of the Technical Report entitled Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study.

I am independent of Compañía Minera Kuri Kullu as independence is described by Section 1.4 of NI 43-101.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

JHearne

John Hearn FAusIMM (CP).

Dated: 1 September, 2011

CERTIFICATE OF QUALIFIED PERSON

Michael Drozd SME. R. M.
AMEC E&C Services Inc
780 Vista Blvd, Suite 100
Sparks, Nevada
USA 89434-6656

I, Michael Drozd RM SME, am employed as Consulting Metallurgist, Associate Metallurgical Engineer AMEC Americas.

This certificate applies to the technical report entitled "Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study", with effective date July 17th, 2011

I am a registered member of the Society of Mining, Metallurgy and Exploration. I graduated from Plattsburgh State University College Bachelors Science Degree in Chemistry in 1974 and New Mexico Institute of Mining and Technology Masters of Science in Chemistry in 1976.

I have practiced my profession for 35 years. I have been directly involved in underground and open pit mining operations, hydrometallurgical and mineral processing facilities, mining project development and metallurgical research & development and design and engineering of processing facilities in USA, Canada, Brazil, Peru, Australia, Turkey, Argentina, Nicaragua, Honduras, Panama, El Salavador and CIS

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 from 17 to 19 December 2010.

I am responsible for all or parts of Section 13 and Section 17 of the Technical Report entitled Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study.

I am independent of Compañía Minera Kuri Kullu as independence is described by Section 1.4 of NI 43-101.

I had not worked on the Ollachea Project prior to December 2010.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

“signed"

Michael Drozd SME Registered Member

Dated: 30 August, 2011

CERTIFICATE OF QUALIFIED PERSON

Brett Byler, P.E.
AMEC (Perú) S.A.
Calle Las Begonias, 441, Piso 8
San Isidro, Lima 27, Perú
Tel: (511) 221-3130
Fax: (511) 222-0738

I, Brett Byler, P.E., am employed as Manager of Geosciences and Tailings, Environment and Infrastructure Division, with AMEC (Peru) S.A.

This certificate applies to the technical report entitled "Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study", with effective date 17 July, 2010

I am a registered Professional Engineer in the State of Colorado, USA (No. 39291). I graduated from the Colorado School of Mines with a Bachelor of Science Degree in Geological Engineering in 1995 and from the University of Colorado with a Masters of Science Degree in Civil (Geotechnical) engineering in 2003.

I have practiced my profession for 15 years. I have been directly involved in underground and open pit mining operations in USA, Argentina, Brazil, Peru, Chile, Bolivia and Spain.

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 for three days in August, four days in October and 13 to 19 December 2010.

I am responsible for parts of Section 18 of the Technical Report entitled Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study.

I am independent of Compañía Minera Kuri Kullu as independence is described by Section 1.4 of NI 43-101.

I had not worked on the Ollachea Project prior to June 2010.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"signed"

Brett Byler, P.E.

Dated: 1 September, 2011

CERTIFICATE OF QUALIFIED PERSON

Christopher Wright, P.Geo.
AMEC (Perú) S.A.
Calle Las Begonias, 441, Piso 8
San Isidro, Lima 27, Perú
Tel: (511) 221-3130
Fax: (511) 222-0738

I, Christopher Wright, P.Geo. am employed as Consulting Manager, Geology and Mining with AMEC (Peru) S.A.

This certificate applies to the technical report entitled Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study, with effective date 17 July, 2011.

I am a member of the Association of Professional Geoscientists of Ontario (APGO 0901). I graduated from McGill University with a Bachelor of Science Degree in Geology and Environmental Sciences in 1997.

I have practiced my profession for 14 years. I have been directly involved in underground and open pit mining operations, mineral exploration and mineral deposit evaluation in Canada, Brazil, Peru, Australia, Ecuador, Chile and Bolivia.

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 and MKK Juliaca Core Storage Facilities for three days in June, four days in October and four days in December 2010.

I am responsible for all or parts of Section 1 to 6, Items 10.5, 10.6 and 11.4 and Item 19 to 27 of the Technical Report entitled Ollachea Gold Project, Puno Region, Peru, NI 43-101 Technical Report on a Pre-feasibility Study.

I am independent of Compañía Minera Kuri Kullu as independence is described by Section 1.4 of NI 43-101.

I had not worked on the Ollachea Project prior to June 2010.

I have read NI 43–101 and this report has been prepared in compliance with that Instrument.

As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Christopher Wright P.Geo.

Dated: 1 September, 2011

PROFESSIONAL GEOSCIENTISTS
CHRISTOPHER S. WRIGHT
PRACTISING MEMBER
0901
ONTARIO

IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report for Compañía Minera Kuri Kullu S.A.) by AMEC E & C Services Inc. (AMEC) and Coffey Mining Pty Ltd (Coffey Mining). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in AMEC's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by MKK subject to the terms and conditions of its contracts with AMEC and Coffey Mining. Except for the purposes legislated under Canadian provincial securities law, any other uses of this report by any third party is at that party's sole risk.

CONTENTS

TABLES

FIGURES

1.0 Summary

The Ollachea Project (the Project) is located in the Puno Region of southern Peru. Minera Kuri Kullu S.A. (MKK), a wholly owned subsidiary of Minera IRL S.A., currently owns the Project and retained AMEC Peru S.A. (AMEC) and Coffey Mining Pty Ltd (Coffey Mining) to conduct a Pre-feasibility Study on the viability of mining the deposit from underground and processing ore in a 3,000 t/d facility to produce gold doré.

1.1 Property, Access and Permits

The Ollachea Project consists of 12 concessions covering an area of 8,698.98 ha. Six of the concessions were acquired from Rio Tinto Mining and Exploration Limited Sucursal del Peru (Rio Tinto) in exchange for cash payments and a 1% vendor royalty. The remaining six concessions were originally applied for by MKK. All concessions are currently held by MKK and are in good standing.

MKK has a surface rights agreement with the Community of Ollachea allowing it to use the property covering the area of interest of the Project for exploration activities. The agreement allows the Community of Ollachea to carry out artisanal mining activities on the property until a production MKK also currently holds permits which allow them to continue exploration activities and develop an exploration access drive as part of their exploration program.

The property is crossed by Interoceanic Highway which allows ford year-round highway access to the Regional Capital of Puno, an airport in Juliaca with daily scheduled flights to Lima and Arequipa, 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 allow exploration activities to be carried out year-round.

The property position and surface rights are sufficient to allow MKK to continue to explore and carry out study work on 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 Minapampa and Minapampa East 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 in carbonaceous sediments cut by quartz-carbonate-sulphide veins and veinlets within a shear zone. Metasediments in the shear zone are characterized by well-developed slaty cleavage. Mineralized quartz and quartz-carbonate veins and veinlets occur within the slate in the shear zone and are broadly concordant with the slaty 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. The current database consists of 153 diamond drill holes totalling 60,846 m in length. Samples were prepared and analyzed at 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 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 using ordinary kriging to estimate block grades into 20 mE x 20 mN x 4 mRL parent blocks, from 2 m composites. 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 Project at a 2.0 g/t Au lower cut-off grade consist of 10.7 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 3.0 g/t Au containing 0.3 million ounces gold. This Mineral Resource has been estimated by Doug Corley, MAIG, of Coffey Mining, Perth, Qualified Person under National Instrument 43-101 and has an effective date of 31 May, 2011 (Table 1-1). These Mineral Resources replace the Mineral Resources reported in Coffey (2011a) and are inclusive of the Mineral Reserves reported in Section 1.3.

Table 1-1: Mineral Resources for the Ollachea Project

Mineral Resources above a 2.0 g/t Au Cut-off Grade	Tonnage (Mt)	Au Grade (g/t)	Contained Au (Moz)
Minapampa			
Indicated	9.3	4.0	1.2
Inferred	2.4	3.0	0.2
Minapampa East			
Indicated	1.4	3.9	0.2
Inferred	0.9	3.0	0.1
Total			
Indicated	10.7	4.0	1.4
Inferred	3.3	3.0	0.3

Note: Mineral Resources are estimated by Doug Corley, MAIG, QP, of Coffey Mining and have an effective date of 31 May, 2011. 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.

Exploration targets on the Project include the Concurayoc Zone, westward along strike from the Minapampa Zone, the eastern extension of Minapampa East, beyond where drilling from surface is impractical for topographic reasons, and the down-dip extension of the Minapampa and Minapampa East Zones.

There are 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 sub-level open stoping. Backfill will consist of pastefill and waste rock, with pastefill used to fill approximately 80% of the mining void. In general, stopes will be mined with transverse accesses in wider zones and longitudinal access in zones of less than 7 m width. The minimum horizontal mining width is 2.6 m, including dilution.

Major mine development will be accessed via an access adit which will be completed as an exploration adit prior to project commitment and is not considered as part of the scope of facilities of the Pre-feasibility study. This will be followed by ramp developments, ventilation raises, level accesses and haulage drifts on 15 m levels both above and below the main access adit.

A cut-off grade (COG) of 2.0 g/t Au was used for the Pre-feasibility Study mine design. Considering the operating costs, gold recovery, gold price and selling costs developed in the Pre-feasibility Study, the 2.0 g/t Au cut off is approximately 15% higher than the estimated break-even grade for Ollachea.

The development schedule consists of:

- Decline development – 5,166 m
- Level development – 6,548 m
- Vertical development – 1,405 m
- Operating development – 46,224 m

Probable Mineral Reserves totalling 9.5 Mt grading 3.6 g/t Au and containing 1.1 million ounces of gold are declared based on the results of the Pre-feasibility Study and the application of appropriate mining factors, and taking in to account relevant processing, metallurgical, economic, marketing, legal, environmental, socio-economic and government factors. Mineral reserves are based on a gold price of US$ 1,100/oz, an exchange rate of 2.72 (Peruvian Nuevo Sole / US $), life of mine (LOM) average site operating costs of US$ 46.61/t and LOM average metallurgical recovery of 91%. Mineral Reserves have an effective date of June 26, 2011. Mineral Reserves have been reviewed by John Hearne, FAusIMM, of Coffey Mining, who is the Qualified Person for the estimate.

The Pre-feasibility Study mine production schedule is based on mining the Probable Mineral. The mine will produce approximately 1.1 Mt/a over a mine life of eight years with two production ramp-up years and one ramp-down year (Table 1-2).

Table 1-2: Ollachea PFS Mine Production Plan

	Unit	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	LOM
Mine Production													
Mine Production	kt	57	332	945	1,101	1,098	1,096	1,102	1,095	1,100	1,098	453	9,477
Contained Gold	koz	6	39	102	128	132	132	146	135	132	116	44	1,112
Grade	g/t	3.2	3.7	3.4	3.6	3.7	3.8	4.1	3.8	3.7	3.3	3.1	3.7

1.4 Metallurgical Testwork and Process Design

The interpretation of results from metallurgical testwork carried out in five campaigns carried out between 2008 and 2011 has been used to guide process plant design. Testwork suggests 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 can be used to achieve gold recovery of over 90% from the Ollachea mineralization.

The flowsheet applied comprises three stages of crushing followed by overflow ball milling with the mill circuit closed with hydrocyclones. Hydrocyclone overflow will then be leached for approximately 50 hours in a pure CIL circuit prior to cyanide detoxification by SO_2/air.

A gravity circuit employing a centrifugal concentrator will treat a split of the hydrocyclone underflow, the concentrate from which will be intensively leached. The circuit has been designed to accept up to 20% gravity recovery.

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

Tailings will be thickened after cyanide detoxification using a high-rate thickener and filtered using press filters. The filter cake will be routed to a paste plant at the plant site 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 that will treat mine drainage, river water and water recovered from mineral processing to provide make-up water for the plant. The mine power supply will be by connection to the San Gaban – Azangaro high tension line that runs over the Project area. Reagents including hydrated lime for pH control, Portland cement for backfill, sodium cyanide for leaching, kerosene for blanking, hydrochloric acid for carbon washing and copper sulphate and sodium bisulfite for cyanide detoxification will be procured from suppliers in Juliaca, Arequipa and Lima and prepared and distributed by reagent preparation circuits considered in the PFS plant design.

Metallurgy and mineral process design has been supervised and reviewed by Michael Drozd, R.M. SME. of AMEC, Reno, who is the Qualified Person under NI 43-101 for this work.

1.5 Tailings Disposal Facility

AMEC completed a Pre-feasibility Study design and cost estimate for a filtered tailings storage facility (TSF). The design includes a principal TSF located within 10 km of the process and filter plants, as well as a contingency TSF that would provide short-term tailings storage near the plant. Filtered tailings will be transported by truck from the plant to the TSF via the Interoceanic Highway. Development of an approximately 1,000 m long off-highway haul road will be required at the principal TSF site.

The location of the TSF is not specified in this report as negotiations for the surface rights for one or more potential sites are currently underway. AMEC considers it a reasonable assumption that surface rights for a suitable site can be acquired by MKK.

The TSF design includes a prepared foundation with an under-drain system, rock-fill toe buttress, surface water diversion channels and sedimentation and seepage collection ponds. Filtered tailings will be placed in two zones: (i) a formally compacted structural zone, and (ii) a nominally compacted zone for “off-spec” tailings (tailings that do not meet the moisture requirements). Overall tailings slopes of 2.5 : 1 (horizontal : vertical) were demonstrated analytically to have acceptable safety factors for stability based on the currently-available information. These slopes are also consistent with acceptable values from comparable sites that use filtered tailings.

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 Minera IRL S.A., from AMEC and Coffey Mining cost estimation databases, and from quotations for major reagents and wear parts.

A life-of-mine staffing schedule was built and indicates that peak operating staffing will be 519 including 109 contract staff for the tailings storage facility, perimeter security, and catering functions. As much as 30% of the workforce could be locally based with the remainder being based nationally. It is not expected that expatriates will be required for the long-term operating team.

Mine operating costs average US$18.48/t ore including backfill. Plant operating costs total US$24.26/t ore processed (including tailings disposal) and G&A costs average US$3.87/t ore. Total site operating costs are US$46.61/t ore or US$436/oz of gold (Table 1-4).

1.6.2 Capital Cost Estimates

AMEC prepared an AACE Class 4 capital cost estimate with a precision of ±25% for the Pre-feasibility Study. The capital cost estimate is based on:

- Capital cost estimates for the underground mine from Coffey Mining,
- Major equipment and material quotations along with recent construction contractor quotations,
- Material take offs (MTO) information from mechanical, civil and electrical engineering,
- Unit costs for earthworks, concrete works, structural steel fabrication and equipment installation were prepared from material costs, labour and construction equipment rates and productivities from the AMEC cost estimation database with input from MKK.

Capital costs include direct and indirect costs for the mine, process plant and infrastructure. Project direct capital costs total US$113.8 M. The total indirect cost is US$19.6 M and includes indirect mine costs, engineering, procurement, and contract management (EPCM), temporary facilities, duties and freight. Owner's costs projected to be incurred between project commitment and prior to commissioning are estimated to total US$7.5 M. A 20% contingency is placed on direct and indirect capital costs for the mine, plant and surface infrastructure. Design growth allowances for civil, structural and architectural (CSA) disciplines were estimated as percentages of the estimated costs of earthworks (15%), concrete works (10%), structural steel (5%) and process equipment (2%). The total contingency and design growth allowance for the project is US$28.6 M.

Table 1-3: Consolidated Estimated Operating Costs

Consolidated Operating Costs		2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	LOM
Mine Production																
Mine Production	kt			57	332	945	1,101	1,098	1,096	1,102	1,095	1,100	1,098	453		9,477
Mine Production Contained Gold	koz			6	39	102	128	132	132	146	135	132	116	44		1,112
Mine Production Grade	g/t			3.2	3.7	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.0		3.6
Stockpile Movement																
Stockpile (Closing)	kt			57	172	17	18	16	12	14	9	10	8	0		
Stockpile Contained Gold	koz			5.9	20.4	1.9	2.0	1.7	1.6	1.8	1.0	1.1	0.8	0.0		
Stockpile Grade	g/t			3.2	3.6	3.6	3.5	3.5	3.8	3.9	3.6	3.6	3.4	0.0		
Commissioning																
Commissioning Feed	kt				41											
Commissioning Contained Gold	koz				4.3											
Commissioning Head Grade	g/t				3.2											
Plant Production																
Plant Feed	kt				176	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	460		9,477
Plant Feed	koz				20	121	127	132	132	145	135	132	117	45		1,112
Plant Feed Head Gold Grade	g/t				3.6	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.1		3.65
Plant Operating Quarters					1	4	4	4	4	4	4	4	4	4		
Plant Avg. Operating Capacity	%				100	100	100	100	100	100	100	100	100	64		
Gold Production	**koz**				22	110	116	121	121	134	124	121	106	41		1,015
Consolidated Project Operating Costs																
Mine Operating Costs																
Mine Operating - Development Direct	US$ M				4.8	9.3	8.5	7.5	5.1	5.6	5.5	3.7	2.2	0.0		147.2
Mine Operating - Production Direct	US$ M				4.3	10.5	11.3	11.0	11.6	10.5	10.2	9.5	10.7	5.3		101.8
Mine Operating - Development Indirect	US$ M				1.0	1.3	1.0	0.9	0.6	0.7	0.7	0.4	0.2	0.0		28.0
Mine Operating - Production Indirect	US$ M				0.4	1.6	2.0	2.1	2.4	2.4	2.4	2.8	2.9	2.3		196.5
Total Mine Operating Costs	US$ M				10.5	22.7	22.7	21.5	19.6	19.2	18.8	16.3	16.0	7.7	0	175.1
Plant Operating Costs																
Wear Parts	US$ M				0.41	2.54	2.54	2.54	2.54	2.54	2.54	2.54	2.54	1.06		21.8
Reagents (Process Plant)	US$ M				2.20	13.76	13.76	13.76	13.76	13.76	13.76	13.76	13.76	5.76		118.1
Consumables	US$ M				0.03	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.07		1.5
Power (Average Demand)	US$ M				0.57	2.32	2.32	2.32	2.32	2.32	2.32	2.32	2.32	1.16		20.3
Fuel - Diesel	US$ M				0.04	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.21		8.2
Elution Circuit Thermal Oil (Heating Oil)	US$ M				0.000	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.001		0.02
Assays And Quality Control	US$ M				0.01	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.04		0.7
Others/Miscelaneous	US$ M				0.01	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.02		0.3
Plant Mobile Fleet - Spares And Cons.	US$ M				0.01	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.02		0.4
Maint Supplies Processing Plant	US$ M				0.10	0.62	0.62	0.62	0.62	0.62	0.62	0.62	0.62	0.26		5.3
Maint. Supplies General	US$ M				0.03	0.17	0.17	0.17	0.17	0.17	0.17	0.17	0.17	0.07		1.5
Plant Manpower	US$ M				0.30	1.24	1.24	1.24	1.24	1.24	1.24	1.24	1.24	0.62		10.8
Contract Tailings Haulage and Placement	US$ M				0.94	4.75	4.75	4.75	4.75	4.75	4.75	4.75	4.75	1.99		40.9
Total Plant Operating Cost	US$ M				4.65	26.75	26.75	26.75	26.75	26.75	26.75	26.75	26.75	11.27	0	229.9
Unit General and Administrative Cost																
Unit G&A	US$ M				1.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	2.4	0.0	36.7
Total Operating Cost	US$ M				16.2	53.6	53.6	52.4	50.5	50.1	49.7	47.2	46.9	21.4	0.0	441.7
Unit Operating Costs																
Mine Production Unit Cost	$/t				59.88	20.62	20.66	19.56	17.84	17.49	17.11	14.85	14.53	16.67		18.48
Plant Production Cost	$/t				26.40	24.32	24.32	24.32	24.32	24.32	24.32	24.32	24.32	24.50		24.26
G&A Cost	$/t				6.03	3.77	3.77	3.77	3.77	3.77	3.77	3.77	3.77	5.26		3.87
Total Site Operating Cost	$/t				92.30	48.71	48.75	47.65	45.93	45.58	45.20	42.94	42.62	46.42		46.61
Total Cash Operating Cost	$/oz				724	488	461	434	418	375	401	391	443	525		436

Table 1-4: Life of Mine Capital Cost Summary

Consolidated Capital Costs		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	LOM
Mine Production													
Mine Production	kt	57	332	945	1,101	1,098	1,096	1,102	1,095	1,100	1,098	453	9,477
Mine Production Contained Gold	koz	6	39	102	128	132	132	146	135	132	116	44	1,112
Mine Production Grade	g/t	3.2	3.7	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.0	3.6
Stockpile Movement													
Stockpile (Closing)	kt	57	172.5	17.1	18.1	15.6	11.8	13.9	9.4	9.6	7.5	0.0	
Stockpile Contained Gold	koz	6	20.4	1.9	2.0	1.7	1.6	1.8	1.0	1.1	0.8	0.0	
Stockpile Grade	g/t	3.2	3.7	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.0	
Commissioning													
Commissioning Feed	Kt		41										
Commissioning Contained Gold	Koz		4										
Commissioning Head Grade	g/t		3.7										
Plant Production													
Plant Feed	Kt		176	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	460	9,477
Plant Feed	Koz		20	121	127	132	132	145	135	132	117	45	1,112
Plant Feed Head Gold Grade	g/t		3.7	4.7	5.7	6.7	7.7	8.7	9.7	10.7	11.7	12.7	12.7
Plant Operating Quarters			1	4	4	4	4	4	4	4	4	4	
Plant Avg. Operating Capacity	%		100	100	100	100	100	100	100	100	100	64	
Gold Production			0	0	0	0	0	0	0	0	0	0	2
Capital Cost Schedule													
Mining Equipment	US$ M	16.6	10.4										27.1
Mining	US$ M	14.6	9.3										23.9
Mining - Backfill & other	US$ M	1.3	1.3										2.5
Site Development	US$ M	0.9	0.9										1.8
Process Plant	US$ M	22.4	22.4										44.9
Site Utilities	US$ M	1.2	1.2										2.4
Ancillary Buildings	US$ M	4.2	4.2										8.5
Tailings System	US$ M	0.4	2.2										2.6
Indirect Costs	US$ M	9.8	9.8										19.6
Indirect - Owners Costs	US$ M	2.6	4.9										7.5
Design Growth Allowance		1.0	1.0										1.9
Contingency - Mining (excl. backfill)	US$ M	6.2	3.9		-	-							10.2
Contingency - Directs/indirects (excl. Owners costs)	US$ M	8.1	8.4										16.5
Sustaining Capital	US$ M	-	6.3	7.7	4.2	5.5	5.3	6.0	4.7	3.5	3.9	0	47.0
Closure Cost											0	0	0
Total Pre-production capital cost	US$ M	89.4	80.1										169.5
Total Sustaining Capital Cost	US$ M	0.0	6.3	7.7	4.2	5.5	5.3	6.0	4.7	3.5	3.9	0.0	47.0
Total Capital Cost	US$ M	89.4	86.4	7.7	4.2	5.5	5.3	6.0	4.7	3.5	3.9	0.0	216.5

1.7 Financial Analysis

A financial evaluation of the Project was undertaken using the discounted cash flow analysis approach. Cash flows were projected for the life of mine (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 and taxes, were subtracted from the cash inflows to estimate the net cash flows (NCF). A financial model was constructed on a quarterly basis to estimate the NCF over the LOM. The NCF were summarized on an annual basis. The cash inflows and outflows are assumed to be in constant second quarter 2011 US dollar basis.

The Project was evaluated on a project stand-alone, 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 were considered to be sunk costs. The financial results also exclude any expenditure between completion of the Pre-feasibility Study and commencement of construction. The analysis was done on a forward-looking basis, with the exception of the sunk costs to date, which were taken into account for tax calculations.

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

The base case gold price used in the financial evaluation was US$1,100/oz. The financial evaluation was also undertaken using a gold price of US$1,500/oz to show the impact of a higher gold price on the Project cash flow.

The model includes Peru government royalty, a vendor royalty, credit & debt tax, income tax and workers' profit participation. The Peruvian taxation system IGV (sales tax) was assumed to be incurred on the initial project capital cost and to be recovered once in production. Once in production, IGV was excluded from the operating assumptions. Since the Project involves export of goods, IGV is assumed to be immediately recoverable, consistent with Peruvian established practice.

A summary of the financial results is presented in Table 1-6.

A sensitivity analysis was performed on the Base Case NPV, using a 7% discount rate, and IRR (Figure 1-3 and Figure 1-4). 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.

Table 1-5: Summary of Ollachea Financial Results

Parameter	Unit	Base Gold Price US$1,100/oz	Upside Gold Price US$1,500/oz
Net Cash Flow before tax	US$ M	419	808
NPV @ 5% real (before tax)	US$ M	270	561
NPV @ 7% real (before tax)	US$ M	226	486
NPV @ 10% real (before tax)	US$ M	170	393
IRR (before tax)	%	28.1	46.5
Payback (before tax)	Years	3.1	1.9
Net Cash Flow (after tax)	US$ M	280	531
NPV @ 5% real (after tax)	US$ M	167	354
NPV @ 7% real (after tax)	US$ M	133	301
NPV @ 10% real (after tax)	US$ M	91	235
IRR (after tax)	%	20.5	34.1
Payback (after tax)	Years	3.8	2.5

Note:
1. NPVs as at commencement of construction.
2. NPVs are based on mid period discounting.
3. Before tax is before Workers' Participation Profit of 8% and Income Taxes of 30%.
4. Payback starts from the commencement of production.

Figure 1-1: NPV @ 7% Sensitivity Analysis

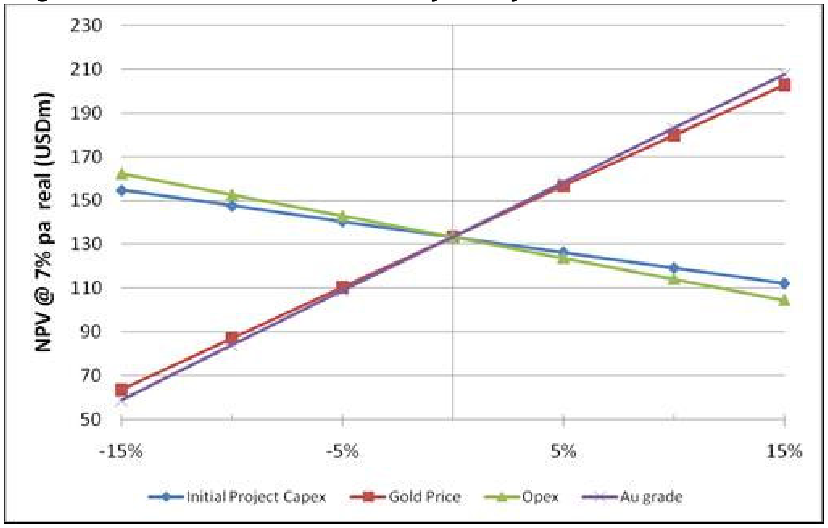
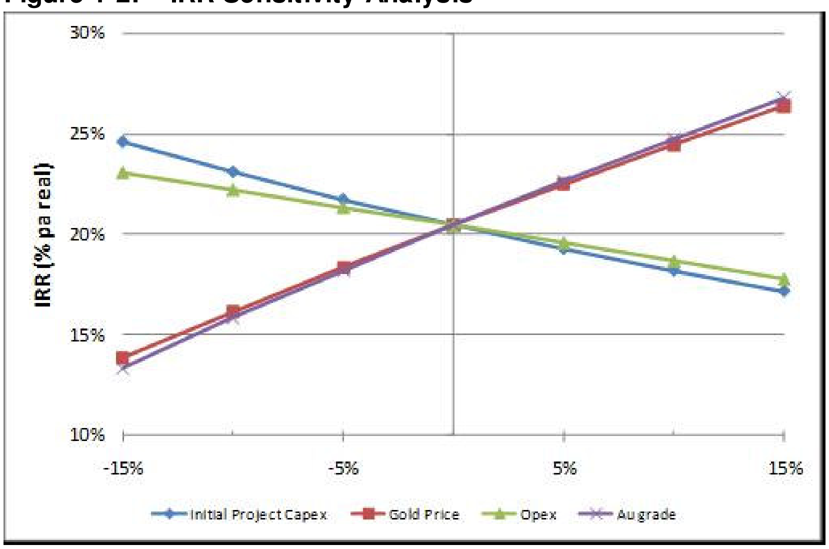


Figure 1-2: IRR Sensitivity Analysis



A summary of the analysis of the LOM average unit cost of production is provided in Table 1-7.

Table 1-6: LOM Unit Cost of Production per Ounce of Payable Gold

Parameter	Unit	Cost
Mining	US$/oz	173
Processing	US$/oz	226
G&A	US$/oz	37
Total Site Cash Operating Costs	**US$/oz**	**436**
Realization Costs	US$/oz	5
Royalties	US$/oz	28
Total Cash Costs	**US$/oz**	**470**

1.8 Conclusions

The results of the Pre-feasibility Study indicate that the Ollachea Project, under the assumptions in the study, is a viable underground mining and mineral processing operation.

1.9 Project Risks and Opportunities

1.9.1 Risks

High Water Inflow and Mine Drainage

Management of water inflow to the mine is a significant risk to the Ollachea Project that was identified in the Pre-feasibility Study. Water inflow rate and quality has major repercussions on:

- Permitting: Changes to the flow rate of the Oscco Cachi River and the spring north of Minapampa due to mining will be a potential environmental and social impact of the Project.
- Mining: Water management and pumping may be a burden on the operation during peak inflow years and may also have an impact on operating costs and productivity in the mine.
- Process plant: A water treatment facility to treat mine drainage will be required at the plant site. The nature of the composition of mine drainage is not well understood at this stage of the Project and the technology required for water treatment will need to be defined during Feasibility-level studies.

The risk of high water inflows and related environmental and social impacts can be mitigated by taking the following measures into account during the feasibility work program:

- Provision of a new water supply for the town of Ollachea westward and up the valley from Minapampa: MKK has already completed engineering on the water supply and it is planned to be implemented before mining is scheduled to commence.
- Lining of the Oscco Cachi river bed to limit water inflow into the mine: Design of a lining system has been initiated by MKK.
- Hydrogeological study including installation of additional piezometers, incorporation of data from a planned exploration tunnel, additional hydrology baseline data, three dimensional structural geology modelling, numerical modelling of flow rates
- Evaluation of a grouting program to reduce water inflow to the mine including field testing in the exploration tunnel
- Optimization of mine design to avoid areas that may be susceptible to high inflows, and in pumping and water management design to ensure that the mine has the capacity to efficiently handle likely peak water flows
- Determination of minimum ecological flow for the Oscco Cachi River.
- Determination of potential water inflow composition and mine drainage considering pH, dissolved solids, suspended solids and other parameters necessary for water treatment plant design.

Leach Extraction

The use of the proposed flowsheet under the conditions typically experienced in such a circuit has shown repeatable recoveries over 90% for samples from along strike and down dip of the various ore lenses. Further work is required to explore variability of the ores and to quantify if any significant issues exist with regard to long-term application of the proposed flowsheet.

Pastefill

Pastefill has been selected as the backfill technology for the Ollachea Project. Initial thickening, filtration and tailings characterization work indicate that plant tailings have granulometric, mineralogical and geochemical characteristics that are favourable for the production of filtered tailings and paste backfill. However, rheology, binder, and strength test work are not yet complete. The viability of the proposed pastefill system has not been completely demonstrated in the following areas:

- Rheology for pumping requirements.
- Binder content requirement.
- Curing time for stope cycle considerations.
- Strength for mining secondary stopes against pastefill walls.

To mitigate the Project’s risk due backfill considerations AMEC recommends:

- Finalizing the current pastefill testwork campaign.
- A trade-off study of paste plant and pumping configurations based on the results of the current pastefill testwork campaign.
- Additional tailings characterization and pastefill testwork based on mineralized composites and using larger volumes of sample to more precisely define strengths and slump rates for the paste.

Schedule

Approval of the Project's EIA is on the critical path of its execution schedule. The schedule considers 120 days for Ministry review of the study, 60 days for MKK to address the Ministry's observations as a result of the study review, and 30 days for the Ministry to reconsider the study and approve the Project. Complications in addressing the Ministry's observations or additional rounds of observations may cause a delay in the projected Project Approval timeline.

1.9.2 Opportunities

Exploration Potential

There is potential to add additional tonnage to the mine production plan by continuing to explore the Concurayoc Zone to the west of the Minapampa Zone. The potential to discover additional tonnage down-dip at Minapampa also exists as well as the potential to identify mineralization to the east of Minapampa East. This area will be drilled from underground. Significant exploration discoveries have the potential to add considerably to the mine life. If mine scheduling permits, the inclusion of one or more additional zones may also support plant expansion.

Gold Price

A gold price of US$1,110/oz has been used for financial modelling for the Pre-feasibility study. On 11 July, 2011, the spot gold price was quoted at US$1,521/oz which is approximately 40% higher than the price used for the study. To take advantage of record-high gold prices in the near term, consideration should be given to advancing the Project in a rapid but orderly fashion so as to maximise potential revenue from higher commodity prices during mine ramp-up and operation.

Plant Design Optimization

During the completion of the Pre-feasibility Study MKK identified a number of plant design optimizations that could be undertaken to save on plant capital cost. Future design work should attempt to capitalize on these comments to reduce the project capital cost.

Use of atypical, but commercially applied, flow sheets such as resins and minor elevation of temperature showed improved leaching behaviour in the laboratory. These techniques will be explored further as part of feasibility-level studies.

1.10 Recommendations

A Feasibility Study is recommended for the Ollachea Project.

The recommended work plan for the Feasibility Study begins in August 2011 and includes the following activities:

- Drilling (US$2.4M) to collect data and sample and data for:
  - resource model update
  - geomechanical study
  - hydrogeology
  - geotechnical characterization of tailings and plant site locations
  - sample for metallurgy, tailings and backfill test work
- Mineral Processing testwork program including process flowsheet optimization, pastefill, and tailings testwork (US$0.4M).
- Geotechnical, geomechanical and hydrogeological study (US$0.4M)
- An updated Mineral Resource Model incorporating exploration data to improve confidence in Mineral Resources (US$0.1M).
- An updated mine design and mine schedule incorporating new hydrogeological, and geomechanical data and backfill testwork. (US$0.5M).
- Feasibility study including process and infrastructure design, engineering, capital and operating cost estimation and financial analysis incorporating results of the geotechnical, hydrogeological, mine design and mine schedule and metallurgical test work (US$1.5M)
- Field expenses to continue with environmental base line study, property maintenance, field staff and overheads (US$1.0)

The recommended feasibility work plan will require a budget of approximately US$6.3M.

2.0 Introduction

The Ollachea Property is located in the Puno Region of southern Peru. Minera Kuri Kullu S.A. (MKK), a wholly-owned subsidiary of Minera IRL S.A., (IRL) currently owns the Property and retained AMEC Peru S.A. (AMEC) and Coffey Mining Pty Ltd (Coffey Mining) to conduct a Pre-feasibility Study (PFS) on the viability of mining the deposit from underground and processing ore in a 3,000 t/d 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 18 July, 2011 press release issued by IRL titled: *Minera IRL Announces Positive Prefeasibility Study, Ollachea Project, Peru.*

2.2 Qualified Persons

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

- Doug Corley, MAIG; Coffey Mining Associate Resource Geologist; Geology and Resources QP; responsible for Items 7-12 (excluding items 10.5, 10.6 and 11.4), and 14
- John Hearne, FAusIMM; Coffey Mining Regional Manager (Western Australia; Mining and Mineral Reserves QP; responsible for Items 15, 16
- Mike Drozd, R.M. SME; AMEC Americas Ltd. Associate Mineral Processing Engineer; Metallurgy and Mineral Processing QP; responsible for Items 13, 17
- Brett Byler, P.Eng.; AMEC (Peru) S.A., E&I Civil Engineer; Infrastructure QP; responsible for Item 18
- Chris Wright, P.Geo.; AMEC (Peru) S.A., M&M Consulting Manager, Study Manager, and QP responsible for Items 1 to 6, Items 10.5, 10.6 and 11.4 and Items19 to 27

2.3 Site Visits and Scope of Personal Inspection

From 21 to 22 June, 2010, at the beginning of the Ollachea Pre-feasibility Study, a site visit was held to review the project layout, geology and drilling practices. During the site visit the Ollachea core storage facility in Juliaca was also inspected to review drill core core storage conditions for the project. Doug Corley, and Chris Wright, took part in this site visit.

In August 2010, Brett Byler, P.E. and Dr. John Lupo, P.E., AMEC employees, carried out field reconnaissance for a tailings storage facility (TSF) trade-off study. Dr. Lupo provided additional input on these areas to the Mr Byler.

On 6 and 7 October, 2010 a field visit was undertaken to review drilling progress, to review project geology, and discuss infrastructure locations. Chris Wright took part in this site visit.

From 17 to 19 December, 2010 a site visit was held to discuss updates on mine planning, drilling, metallurgical test results and process flowsheet options and tailings infrastructure locations. Brett Byler, Mike Drozd and Chris Wright participated in this site visit.

From January to March AMEC staff visited the property to carry out hydrogeological study work and inspection of site conditions for civil, mechanical and electrical engineering. These staff provided input to the AMEC QPs on these areas of review.

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 17 July, 2011. The dates for critical information used in this report are:

- The updated Mineral Resource estimate and mineral resource block model were completed on 31 May, 2011
- The Mineral Reserve estimate for the project was completed on 26 June, 2011
- The final PFS mine plan was issued 7 July, 2011
- PFS Mineral process engineering and capital cost estimation were completed 8 July, 2011
- The PFS financial model was finalized 17 July, 2011

There were no material changes to the scientific and technical information on the Project between the effective date and the signature date of the Report.

2.5 Information Sources and References

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

- Draft PFS Study Final Report (AMEC, 2011d)
- Mineral Resource Block Model File - OLMAY11M.dm (Coffey, 2011b)
- Mine Schedule File - MINEWPER00466AG_Rev 1.xlsx
- Capital Cost Estimate File - Cx_OllacheaGold_17072011_Final.xls
- Exploration Access Drivel 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

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.

2.6 Previous Technical Reports

MKK has previously filed the following technical reports for the Project as follows:

Coffey, 2010. Ollachea Gold Project National Instrument 43-101 Technical Report. Unpublished NI 43-101 Technical Report prepared for MKK by Coffey Mining QPs Beau Nichols MAIG, Bernardo Viana MAIG, Jean-Francois St.Onge MAusIMM, and Barry Cloutt MAusIMM, with effective date 6 April, 2010, supporting disclosure of results from the Ollachea Scoping Study. 120 p.

Coffey, 2011a. Ollachea Resource Update, November 2010, National Instrument 43-101 Technical Report. Unpublished Technical Report prepared for MKK by Coffey Mining QP Doug Corely and MKK VP Exploration Don McIver with effective date 14 January, 2011. 145 p.

3.0 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. AMEC has fully relied upon and disclaims responsibility for information derived from legal experts for this information through the following documents:

Tong, F., 2010. Certain mineral rights and permits held by Minera IRL S.A. and Compañía Minera Kuri Kullu S.A.. Memorandum prepared by independent legal studio Rodrigo, Elias & Medrano Abogados for Collins Stewart Europe Limited, Jennings Capital Inc., National Bank Financial Inc and Tim Miller, VP Corporate finance for Minera IRL Ltd. and dated 10 November, 2010. 27 p.

Arevalo, M., 2011. Acutal 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.

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

The independent expert legal opinion of Francisco Tong (Tong, 2010b), of Estudio Rodrigo, Elias, Medrano, a Peruvian law firm, and an internal MKK memo (Arevalo, 2011) were provided to AMEC to support MKK's possession of surface rights enabling them to conduct exploration and development activities on the Ollachea Property. Qualified Persons have relied on, and disclaim responsibility for these opinions in Section 4.4.

3.3 Permitting

The independent expert legal opinion of Francisco Tong (Tong, 2010b), of Estudio Rodrigo, Elias, Medrano, a Peruvian law firm, and an internal MKK memorandum

(Arevalo, 2011) was provided regarding the status of permits allowing MKK to conduct exploration and development activities on the Property. Qualified Persons have relied on, and disclaim responsibility for, these opinions in Section 4.6.

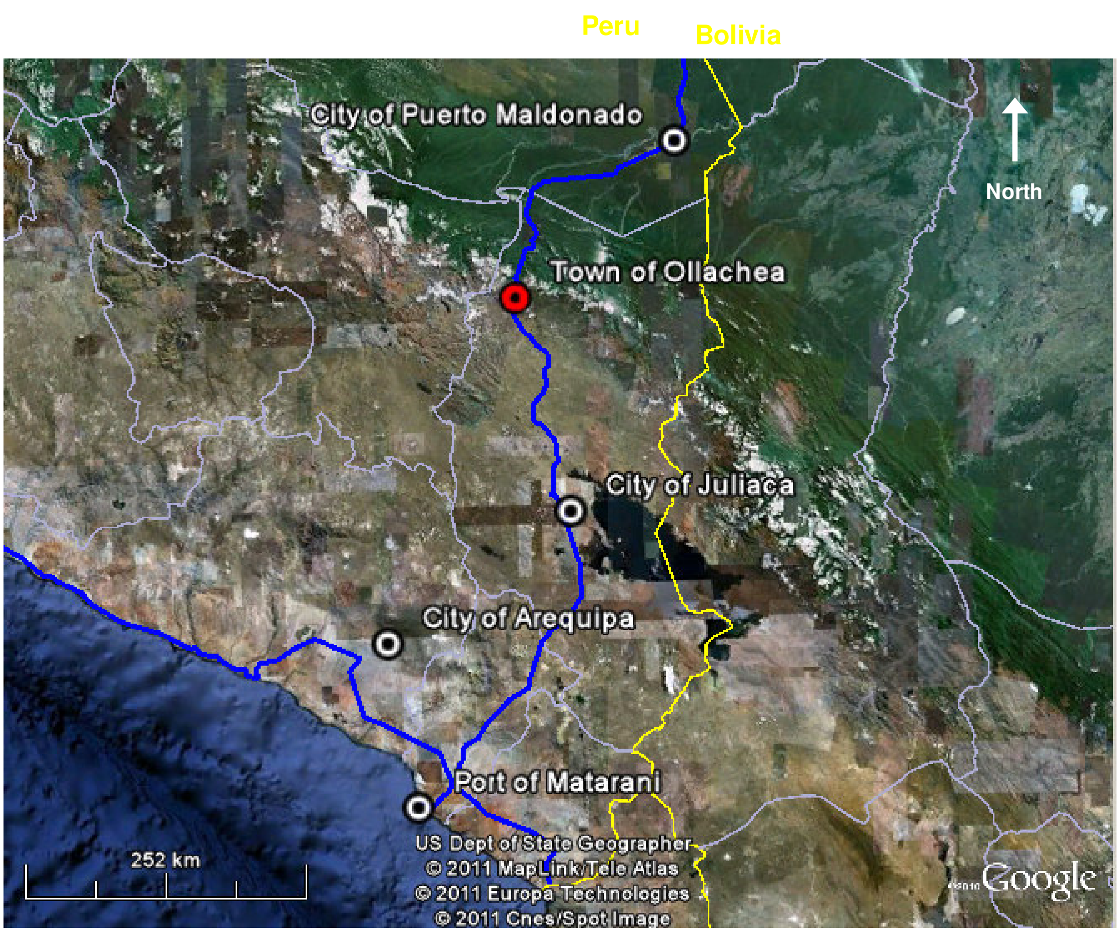
3.4 Social and Environmental Impacts

A summary the environmental baseline of the Ollachea property was prepared by AMEC E&I Environmental Specialist Sophie Bertrand 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.

4.0 Property Location and Description

The Ollachea Project is located immediately north of 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 approximately 200 m 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. The centre of the mineralized zone is located at UTM 339,500 mE, 8,474,600 mN in the WGS 84 coordinate system.

Figure 4-1: Ollachea Project Location and Access


Peru
Bolivia
City of Puerto Maldonado
North
Town of Ollachea
City of Juliaca
City of Arequipa
Port of Matarani
252 km
US Dept of State Geographer
© 2011 MapLink/Tele Atlas
© 2011 Europa Technologies
© 2011 Cnes/Spot Image
Google

4.1 Property and Title in Peru

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, 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 good standing 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.

4.2 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 (2010b) 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.

On 8 July, 2011, the INGEMMET mining concession registry website (http://www.ingemmet.gob.pe/ConsultasDM/DefaultDM.aspx?Opcion=262) listed all annual maintenance and penalty fees for the Oyaechea 1 to Oyaechea 12 concessions as paid and all concessions as in good standing.

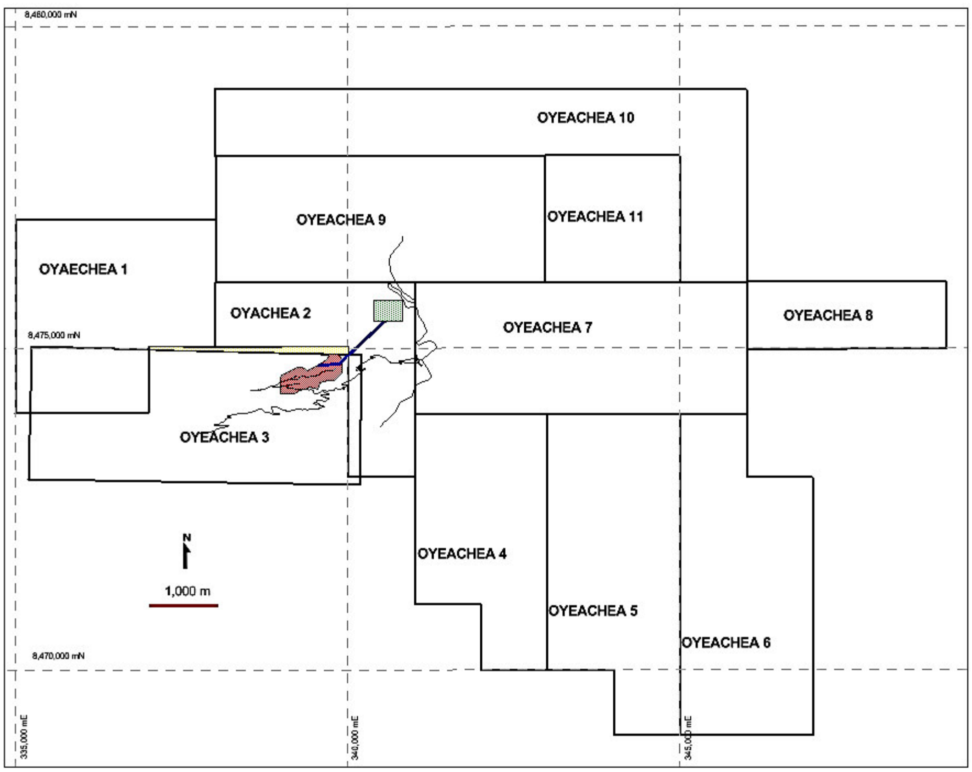
The mineralization included in the Mineral Resource and Mineral Reserves discussed in this Report occur within the Oyaechea 3 concession. The proposed plant site location will be located on the Oyachea 2 concession. The proposed portal location for the exploration access adit, which may also serve as the main mine portal wil be located on the Oyaceha 2 concession.

A gap measuring approximately 3,000 m long by 130 m wide exists between the Oyachea 2 and Oyachea 3 concessions (Figure 4-1). 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.

Table 4-1: Ollachea Concessions

Concession Name	Concession Number	Concession Holder	Area (ha)	Application Date
OYAECHEA 1	10215003	Compañía Minera Kuri Kullu SA	800	23/06/2003
OYAECHEA 2	10215103	Compañía Minera Kuri Kullu SA	500	23/06/2003
OYAECHEA 3	10218103	Compañía Minera Kuri Kullu SA	998.98	24/06/2003
OYAECHEA 4	10215203	Compañía Minera Kuri Kullu SA	700	23/06/2003
OYAECHEA 5	10215303	Compañía Minera Kuri Kullu SA	900	23/06/2003
OYAECHEA 6	10215403	Compañía Minera Kuri Kullu SA	900	23/06/2003
OYAECHEA 7	10389907	Compañía Minera Kuri Kullu SA	1000	19/08/2008
OYAECHEA 8	10389807	Compañía Minera Kuri Kullu SA	300	30/10/2007
OYAECHEA 9	10139909	Compañía Minera Kuri Kullu SA	1000	30/11/2009
OYAECHEA 10	10140009	Compañía Minera Kuri Kullu SA	1000	16/10/2009
OYAECHEA 11	10140109	Compañía Minera Kuri Kullu SA	400	16/10/2009
OYAECHEA 12	10167809	Compañía Minera Kuri Kullu SA	200	22/01/2010

Figure 4-2: Ollachea Exploration Concession Map



Note: The red polygon is the surface projection of Indicated Mineral Resources in the Minapampa and Minapampa East Zones. The green polygon is footprint of the mineral processing plant proposed in the PFS. The yellow polygon between the Oyachea 2 and Oyeachea 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.

4.3 Project Ownership

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, Minera IRL signed an agreement with Rio Tinto to acquire the original Ollachea concessions. On 27 February, 2007 the agreement was ratified and the Rio Tinto concessions were transferred to MKK (Tong, 2010b).

From 2007 to 2009 MKK filed applications for the Oyaechea 7 to Oyaechea 12 concessions which together with the concessions originally held by Rio Tinto constitute the Ollachea Project.

4.4 Surface Rights

The following is summarized from the 2010 IRL Annual Information Forum (IRL, 2011) and a legal opinion memorandum from Estudio Rodrigo, Elias and Medrino (Tong, 2010b):

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. The agreement will be in force for a maximum of five years, and will automatically revert to a development contract at the time a development decision is made. MKK will make payments for surface rights access totaling US$213,333 over the five-year period. In addition, MKK agreed to make contributions to sustainability projects and commit to social responsibility programs for the community totaling US$416,666 and 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, the 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.

4.5 Agreements and Royalties

The following is summarized from the IRL Annual Information Forum document for 2010 (IRL, 2011) and is supported by Tong (2010b).

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, 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.

Rio Tinto's clawback right lapsed in 2009 and on 15 December 2009, Rio Tinto notified IRL that IRL 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 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 a feasibility study, less 30% of the sunk costs determined after the exercise of this option.

The Peruvian government currently levies a sliding-scale royalty on gross sales from mining operations that ranges between 1% and 3%. The mining royalty payable is: 1 % for annual sales of under US$ 60 million, 2% for gross sales from US$60 million to US$120 million and 3% for gross sales in excess of US$120 million.

4.6 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.

4.7 Environment

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.

Archaeological surveys have also been carried out as part of baseline study semi-detailed environmental impact assessment studies prepared to support application for application 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 Challuno area which has been proposed as the plant site area. Consideration of these sites has been taken in the Project layout.

4.8 Comment on Item 4

Information from legal and MKK experts support that all mineral concessions, permits and community agreements are currently in good standing. Based on this information, the QPs are of the opinion that MKK will be able to conduct a feasibility-level work program on the Project.

To the extent known, there are no other significant factors that may affect access, title or the right to perform work on the property.

5.0 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

Road access to the Ollachea Project is by the Interoceanic Highway which runs 200 m east of the proposed plant site for the Project. The Interoceanic Highway is 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 moderate, with regular closures for construction and road clearing activities, but are expected to be improved once work is complete in late 2011.

A series of un-paved roads connect the Town of Ollachea to the Minapampa area and the Oscco Cachi valley and are 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 one to two hours 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 which runs from Tacna at Peru's southern border with Chile and northward to Lima and eventually to Ecuador.

5.2 Climate

The Project has a temperate sub-alpine climate with a pronounced rainy winter season and dry summer season. The rainy season extends from December to April, the dry season from June to September and the remaining months of October, November and May 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, and 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, make-up water for plant operations could be drawn from these sources, mine drainage or from the Ollachea river which has significantly greater flow rates than the Oscco Chachi River.

5.4 Physiography

The Project is located at between 2,500 m and 3,500 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. A number of sites have been identified for tailings storage facilities (TSF) within 10 km of the project and are discussed in Section 18.

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

5.5 Comment on Item 5

The Ollachea 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 and in the future, 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 or are currently being acquired by MKK.

Currently, MKK are in negotiations for surface rights for the proposed dry-stack tailings storage facility. AMEC considers that it is a reasonable assumption that these rights can be obtained through negotiation. However, in the eventuality that the preferred site cannot be used, MKK has made provision for an alternative site, where the company has acquired a significant portion of the necessary surface rights.

AMEC considers it a reasonable assumption that the necessary surface rights for the Project can be acquired in a timely fashion considering the project advancement schedule.

6.0 History

The earliest evidence of mining on the Ollachea Project is attributed to Spanish colonial activity during the 18th century. Informal mining activity was 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 2003 Rio Tinto re-discovered the area in May 2003 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 began negotiations with Rio Tinto in 2006 and following negotiation of a surface access rights agreement with the Community of Ollachea in February 2007, began work on 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). Small-scale artisanal mining continues on the Project (Figure 6-1).

Beginning in early 2007, MKK carried out bedrock sampling, geochemical sampling, mapping and structural geology based on aster image interpretation (Teluris, 2009). By October 2009, 80 diamond drill holes totaling 30,575 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 Pre-feasibility Study for the Project. By November 2010, an additional 46 drill holes totaling 17,536 m had been drilled and the Mineral Resource estimate for the Property was updated (Coffey, 2011a).

Between November 2010 and May 2011, MKK completed 14 more core drill holes totaling 5,949.6 m. This Report discusses an updated Mineral Resource estimate based on the Mineral Resource database to May 2011, and the results of a Pre-feasibility Study carried out in 2010 and 2011 and based on the May 2011 database and updated Mineral Resource estimate.

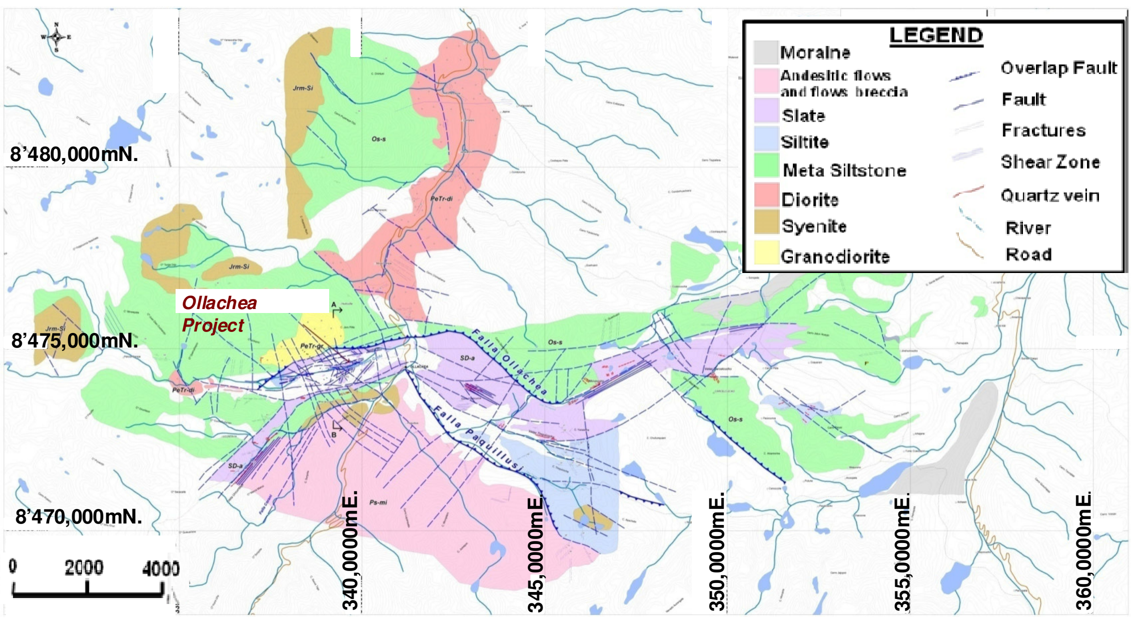
Figure 6-1: Artisanal Mine Workings at Minapampa - October, 2010

7.0 Geological Setting and Mineralization

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 respects to the Ollachea 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)

7.1 Project Geology

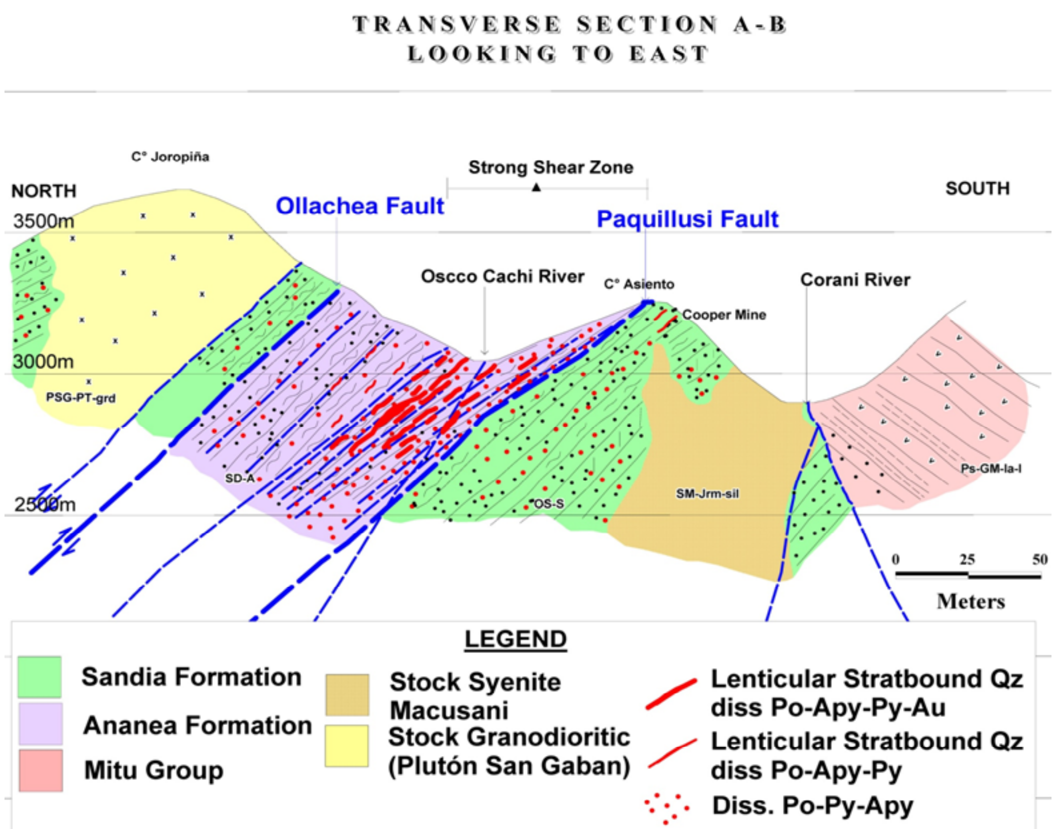
The inferred geology of the Ollachea Project is dominated by phyllites of the Devonian Sandia Formation, and variably bedded graphitic slates and shales of the Siluro-Devonian Ananean Formation. Andesitic volcanic rocks crop out south of the sedimentary units and both the sedimentary and volcanic rocks 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 7-2).

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.

Figure 7-2: Schematic Cross Section of the Ollachea Deposit



(after Ing. Valdivieso, Y., MKK, 2008. Schematic Transverse Section looking East, Ollachea Project. 1:50,000 scale)

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.

7.2 Mineralization

The principal zone of mineralization comprising the Ollachea Prospect is being extensively worked by artisanal miners (Figure 7-3).

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 from the Minapampa zone eastwards into the contiguous Minapampa East Zone. Gold mineralization has also been encountered to the west of the Minapampa Zone in a zone on the south side of the Oscco Cacchi River that is referred to as Concurayoc, located some 400 m west of Minapampa. The known mineralized zone 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. 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 in 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.

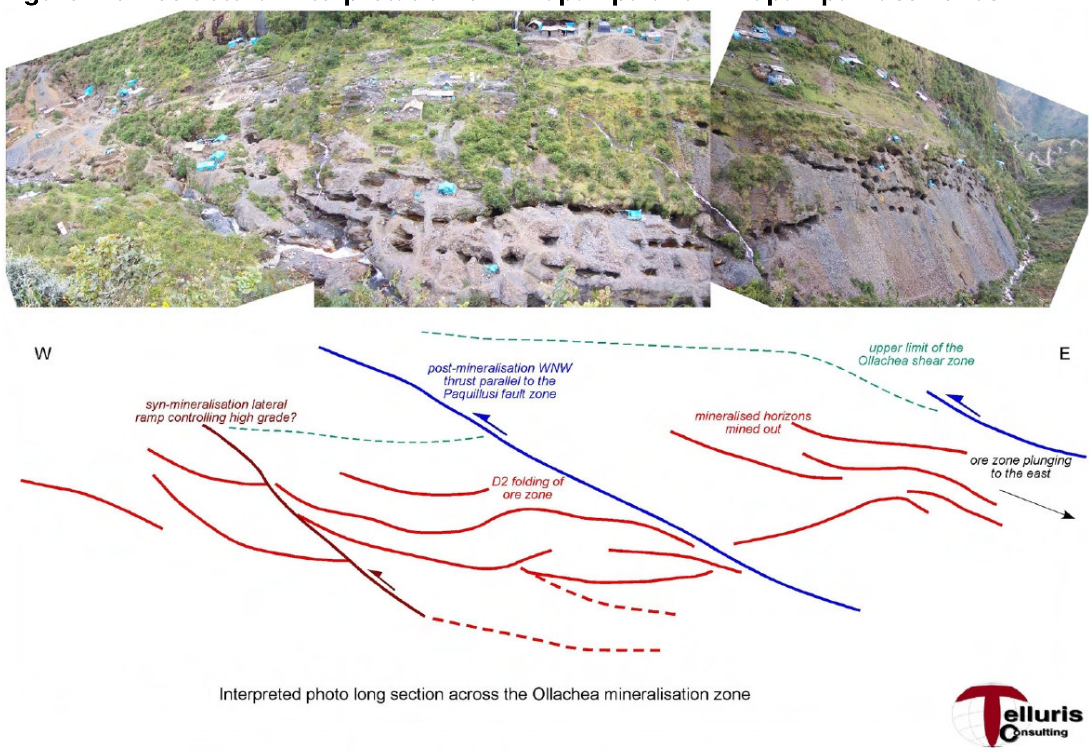
The slate sediments have been classified on a project scale using lithological codes Pz1 to Pz6. These are based on the presence and the nature of pyrrhotite (Po) content as well as other lithological criteria. Intercalations within the slates of fine-grained hornfelsic siltstone, finely-banded slate and occasionally an intercalated slate and quartz lithological unit with a zebra-like texture are also encountered in association with the mineralized zones. Lithological slate “types” are:

- Pz 1: Slate without disseminated Po
- Pz 2: Slate with finely disseminated Po
- Pz 3: Slate with laminated, disseminated acicular Po.
- Pz 4: Slate with coarse dissemination of Po.
- Pz 5: intercalated fine laminated slate and hornfelsic siltstone.

- Pz 6: intercalated slate and quartz banding (zebra-type texture).

The Pz 1 and Pz4 slates are the most common hosts of gold mineralization in the Minapampa and Minapampa East Zones.

Figure 7-3: Structural Interpretation of Minapampa and Minapampa East Zones



(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 and Minapampa East zones. The occurrence of coarse pyrite without other sulphides is often a counter-indicator of gold mineralization.

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

An orientated 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 / correlation of the mineralization relative to the foliation where favourable horizons continue. This information was also used to help interpret the mineralized zones.

8.0 Deposit Types

The deposit model guiding exploration targeting is mesothermal quartz vein style gold mineralization. Coffey Mining (2010, 2011a) and other workers (e.g. Trellius, 2006) 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. References to mesothermal, orogenic or lode gold deposits can be found in Kerrich (2000).

MKK is using 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 of 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.

This model is supported by field evidence at Ollachea and is suitable to guide continued exploration efforts.

9.0 Exploration

Core drilling has been the dominant exploration tool of MKK. Geological mapping and geochemical sampling, together with an Aster satellite image and structural geology targeting exercise completed by Telluris Consulting in September 2009, have additionally contributed to Project understanding.

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.

Coffey Mining considers the exploration targets justify further follow-up. The deeper down dip potential of Ollachea may be better targeted from any future underground development as diamond drilling from surface will require >1 km holes due to the high topography north of the main northward-dipping mineralization.

New discoveries such as the Concurayoc Zone, displaced by some 300 m from the main Minapampa and Minapampa East Zone, have exploration potential. All mineralization discovered to date at Ollachea remains open-ended along strike as well down-dip.

10.0 Drilling

At the database closure date on 28 April, 2011 153 drill holes totalling 60,846 m had been drilled on the Ollachea property. A total of 140 drill holes totalling 54,061 of these holes are around the Minapampa and Minapampa East zones and were used to construct the Mineral Resource Model used in this study. There are 120 drill holes totalling 46,404 m Within the Minapampa and Minapampa East Zones.

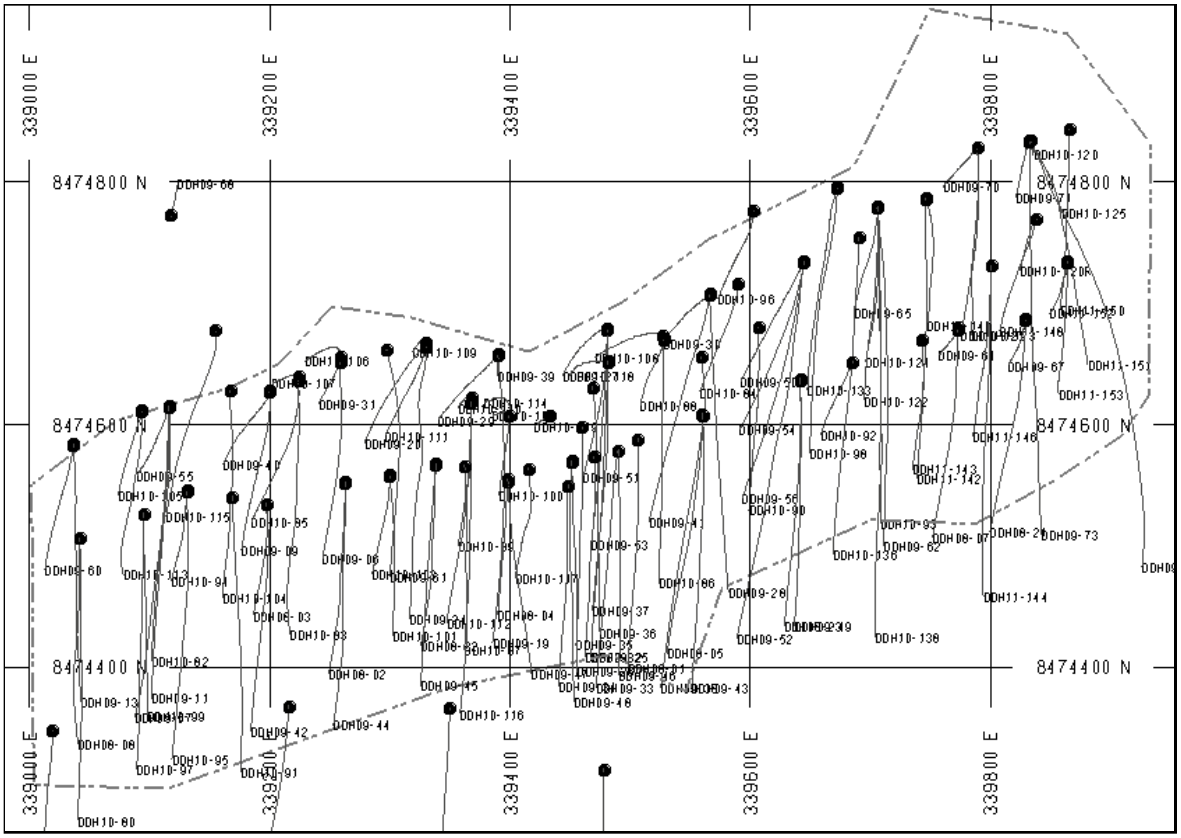
The principal methods used for exploration drilling at Ollachea have been diamond core drilling (DC) by MDH SAC (drilling company), using standard wireline diamond drilling of HQ diameter then reducing to NQ then BQ as ground conditions dictate. Core recovery was very good (greater than 99%); except 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.

All surveying, plotting and mineral resource modelling, utilises the UTM grid in the WGS 84 coordinate system (Zone 19S).

Limited information is known of the 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 drill holes in the Minapamapa and Minapampa East Zones.

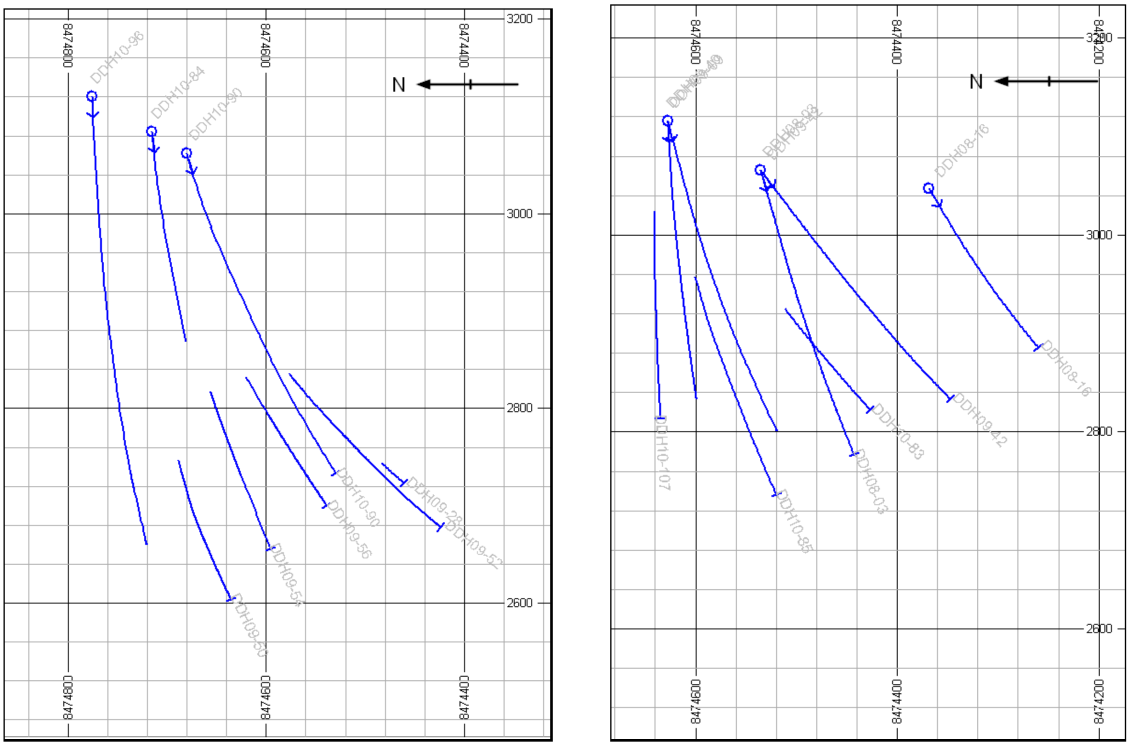
Figure 10-1: Plan View Exploration Drill Hole Location Map



Note: Drill collars are marked with black 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 Minapamapa and Minapampa East Zones.

Figure 10-2: Representative Sectional View - Exploration Drill Holes



Note: left plot is at 339600mE (+/- 20m) and the right plot is at 339200mE (+/- 20m). Grid shown on plot is 40 m by 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 and Minapampa East zones have been drilled at a nominal spacing of 40 m by 40 m.

Figure 10-3: Drill Hole Location Map

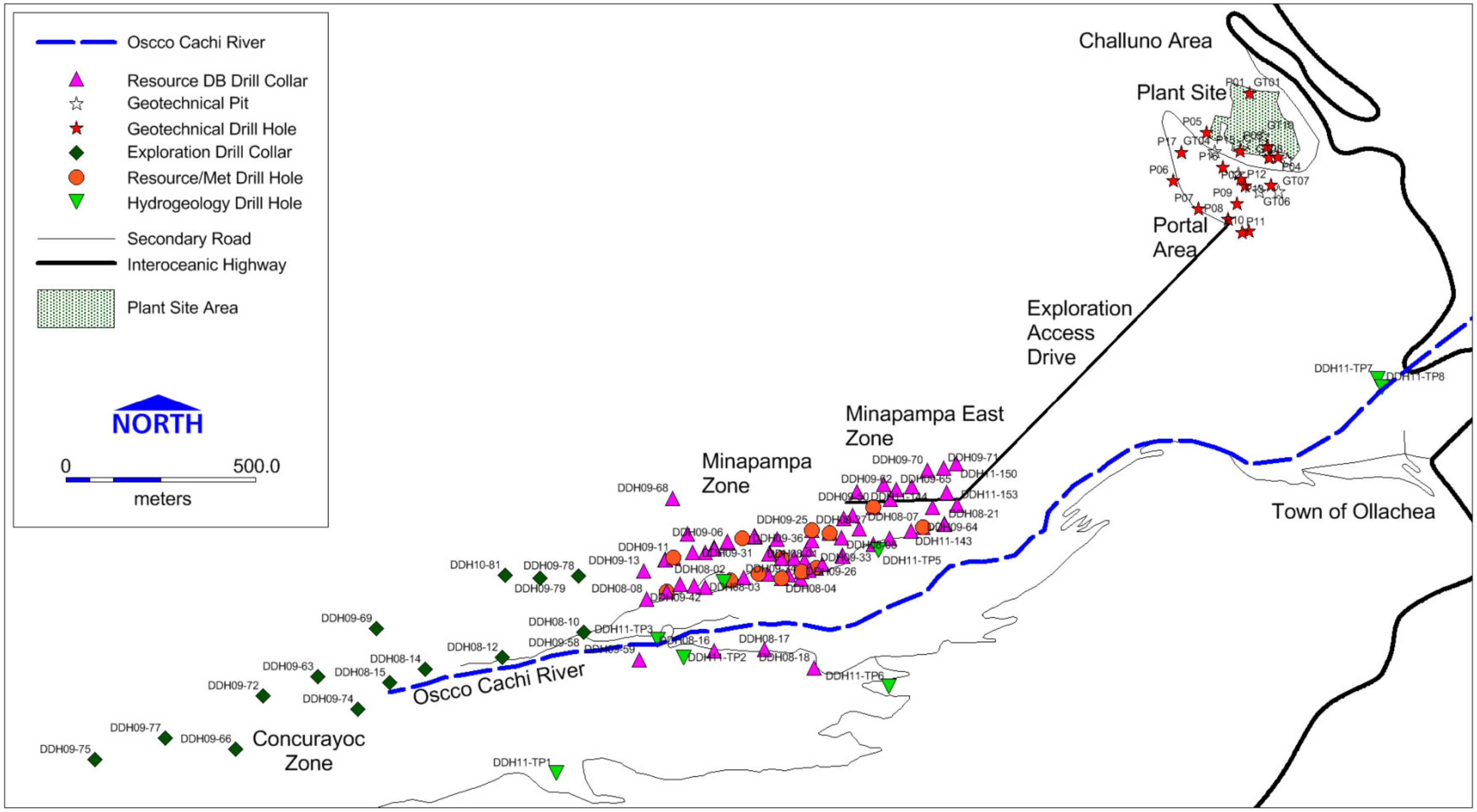


Table 10-4: Drilling and Sampling Campaigns at Ollachea

	Peruvian Gold Ltd.	MKK 2008	MKK 2009a	MKK 2009b	MKK 2010	MKK 2011
Period	1998-1999	September 2008 - May 2009	May 2009 to June 2009	June 2009 to October 2009	October 2009 to November 2010	November 2010 to May 2011
		DDH08-01 to DDH09-33	DDH09-34 to DDH 09-43	DDH09-44 to DDH09-73	DDH09-74 to DDH 09-125	DDH09-120R, DDH10-133 to DDH11-153
Drill Holes	5	35	10	30	51	14
Total Length (m)	501	Total of 48,111 m MKK to November 2010				5,949.6 m
Resource DB	No	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
Blanks	Unknown	8005	8005	8005	8005, commercial blank	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
Preparation Duplicate	Unknown	None	From DDH09-42	Yes	To DDH09-80	No
Pulp Duplicate	Unknown	Yes	Yes	No	No	No
Referee Lab Analysis	Unknown	Inspectorate	Inspectorate	ALS	ALS	ALS
Re-sampling	Unknown	Samples greater than 1 m length in mineralized zones were re-sampled at 1 m lengths to hole DH09-119				No
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			

10.1 Drilling Methods

All diamond drilling used in the May 2011 resource estimate was completed by the MKK contractor. Most diamond core holes were drilled using HQ and reducing to NQ diameter. There were some BQ diameter holes drilled, only one (2 m interval) sample was located within the Minapampa and Minapampa East defined mineralized zones.

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 has been conventional and appropriate.

10.3 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 (2010), Coffey Mining chose several drill collars and verified their location using a hand-held GPS unit. All drill holes checked where within +/- 5 m of the reported location (within the accuracy limits of the device).

The WGS84 datum for UTM Zone 19S is used to plot drill holes and other information in the Mineral Resource Estimate and PFS except where noted.

10.4 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 20m downhole depth.

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, and where therefore used for the resource estimation.

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

10.5 Geotechnical and Hydrological Drilling

10.5.1 Geotechnical Drilling

Two geotechnical diamond drill holes of less than 45 m depth were drilled in the vicinity of the proposed exploration access drive portal in 2010.

Twelve shallow geotechnical drill holes were drilled around the portal access road and plant site for soil assay and characterization purposes in 2011.

Twenty pits were excavated for geotechnical characterization of surface conditions in the vicinity of the plant site and portal access road in 2011.

10.5.2 Hydrogeological Drilling

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

Table 10-1: Hydrogeological 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.

10.6 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 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

10.7 Sample Length/True Thickness

Assay samples for the mineral resource database have been taken at 0.05 m to 2 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 1.3 m.

Exploration drill holes used in the mineral resource estimate were generally drilled to the south at between 40 degrees to 90 degrees dip. Holes were targeted to perpendicularly intersect the main trend of mineralization but given the access to deeper sections of mineralization the intersections are often oblique to mineralization. The deeper sections of Ollachea will need to be targeted from underground or via >1 km surface directional drilling The Minapampa and Minapampa East Zones have been drilled at a nominal spacing of 40 m by 40 m.

The relationship between exploration drilling used in the mineral resource estimate and mineralization is defined in further detail is Section 14. Drill holes typically intersect mineralization orthogonally, and the mineralized intercepts are typically 60% to 100% of the true mineralized thickness.

11.0 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 and 2010, re-sampling campaigns were undertaken such that all mineralized intervals were systematically sampled in intervals no longer than 2.0 m (2009), then intervals no longer than 1.0 m (2010).

The present procedure, in place since the MKK 2009b sampling campaign (refer to Table 10-1), 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. 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 is currently stored at temperatures that are maintained at below -5°C in refrigerated containers at MKK's Juliaca core storage facility.

11.2 Assay Sample Preparation and Analysis

MKK has used the CIMM Peru laboratories as its primary laboratory for preparation and assaying of drill core samples from Ollachea since the MKK 2008 drill campaign. CIMM 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 CIMM 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 crushers followed by a roll crusher to 2 mm. The full sample is riffle split to 500g.
- A 500 g pulp is prepared in LM2 pulveriser bowls to 85% < 75 µm (200 mesh). 50 g pulps were submitted for chemical analysis.

Chemical analysis is conducted at the CIMM 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 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.

The issues identified by Mr. Smee have since been rectified.

BSI Inspectorate laboratories, certified under ISO65 and certAll, and ALS Chemex Lima, certified under ISO 9001:2008, ISO 17025:2005, IQNet, were used as secondary laboratories during the assaying campaigns from 2008 to 2011.

11.2.1 Adequacy of Procedures

Coffey Mining has been advised the main issues identified by Smee (2009), have been rectified and this includes:

- Upgrading the pulverising unit to a COSAN TM, LM2 model
- Pulveriser bowls have been upgraded to B2000 type, so they can handle the 500g pulverisation in one pass
- In regards to the dust extraction unit, the pipe is no longer attached directly to the crusher as before, and the extraction power of the exhaust fan has been reduced.

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.

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

11.3 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.

All of the MKK samples in the Mineral Resource database has been submitted with standard reference materials to control assay accuracy, and depending on the program, has included field duplicate samples, coarse crush duplicates, pulp duplicates to control sampling, sub-sampling and analytical precision. Not all programs have included preparation or pulp duplicates.

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.

11.4 Metallurgical Sampling

11.4.1 Sample Storage

Early in the 2008 MKK drilling program it was noted that the pyrrhotite present in the ore was reactive. Given the anticipated gold associations with the mineral as well as the potential influence oxidation could have on possible flowsheets such as flotation, it was decided the core should be stored in freezers. Refrigerated sea containers were purchased and core stored at sub-zero temperatures.

As the exploration program developed and freezer capacity was exhausted, only the mineralized/gold-bearing intercepts were stored frozen whilst non-mineralized/gold-bearing intercepts were stored under cover but in non-sealed core trays.

Metallurgical samples have been sourced only from the intercepts retained in the freezers and therefore the level of oxidation is very low.

11.4.2 First Metallurgical Sampling Program - 2009

The first sampling program for metallurgical samples occurred in 2009. At this time the detail of the gold-bearing structures was still being developed. A number of composite recipes were developed and were based on continuous or semi-continuous gold-bearing intervals from the Minapampa Zone being combined to provide a number of "down-hole" composites. As a consequence, these first composites were made up to represent mineralized structure by hole and not necessarily "structure" or "lens". Some dilution material from each side of the continuous zones was included to represent a practical mining scenario.

Composite locations varied in depth as well as location along strike.

Such a compositing philosophy allows a partial understanding of metallurgical variance but does not allow this knowledge to be applied to different ore lenses or structures unless it is found later on in the program that the composites do match up with recognised lens/structures. As it turned out, the composites did align with lenses and so the results can be applied to lens metallurgy.

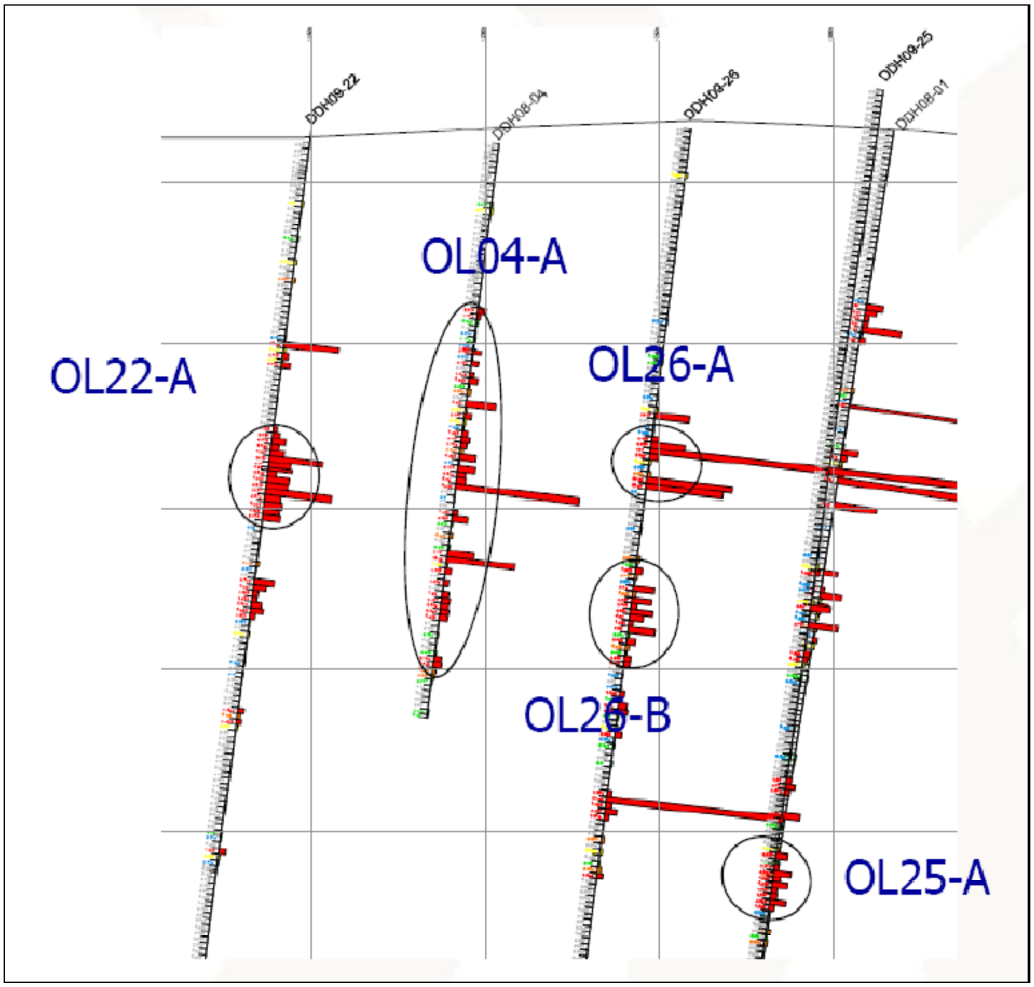
The selected intervals were split (diamond saw) and ¼ core samples were packed in plastic bags and sparged with nitrogen to displace air/oxygen. The sealed bags were then packed in drums, purged with nitrogen and sealed prior to air freighting. At the laboratory the various intervals and composites were stored in freezers to inhibit oxidation. For the first programs, KCA, based in Reno, Nevada was used, and subsequently Ammtec , Perth, Australia, was the primary metallurgical laboratory. Metallurgical laboratories typically do not hold accreditations.

Five composites were made up from the continuous intervals as well as one composite to represent a global (average) composite was also made up. The composites were named OL04-A, OL22-A, OL25-A, OL26-A and OL26-B (two composites from hole DDH09-26) and the global composites OLOGC-A:

- OL-04A was a large composite - considered to be possibly crossing multiple structures given a number of low/zero grade intervals. It was used for large mass tests like work index determination for grinding as well as extraction work.
- OL-22A was a low mass composite - considered to be possibly representing one large lens intersection. It was used predominantly for extraction work.
- OL-25A was a low mass composite - possibly representing one small lens intersection at depth. Used predominantly for extraction work and to see if depth influences metallurgy.
- OL-26A and OL-26B were both low mass composites from the same hole – considered to possibly represent two lenses. They were used predominantly for extraction work and to see if depth influences metallurgy as from the same hole.

Figure 11-1 shows the five down-hole composite sources and relative locations along strike and down dip.

Figure 11-1: Relative Composite Locations and Intervals



The composite head assays were found to vary compared to the theoretical values estimated from the mathematical weighting of the intervals.

- OL04-A. Weighted 2.90 g/t. Head assay 2.09 g/t
- OL22-A. Weighted 4.58 g/t. Head assay 3.05 g/t
- OL25-A. Weighted 2.21 g/t. Head assay 1.97 g/t
- OL26-A. Weighted 15.17 g/t. Head assay 18.74 g/t (good for high grade/coarse gold test work).
- OL26-B. Weighted 2.41 g/t. Head assay 1.27 g/t

Analysis conducted by MKK QA/QC personnel have not been able to find any issues with regard to analysis accuracy, and have concluded that the variation shown does fall inside expected variability .

As the production grade was expected to be around 4 g/t Au and that the five composites noted above varied from this assay, the Ollachea ore grade composite (OLOGC-A) was compiled to have a head grade of around 4 g/t Au. This composite was made up from intersections from DDH08-04, DDH08-22 and DDH09-26 and was found to have an assayed head grade of 3.57 g/t gold.

These various composites from the first round of sampling were the only samples used by KCA for their test work. Once KCA completed their work the remnants were sent to Ammtec for some preliminary work prior to receipt of a second lot of samples.

11.4.3 Second Metallurgical Sampling Program - 2010

During the latter part of 2009, a PA was conducted by Coffey Mining Limited (Coffey). The PA work defined a number of lenses that continued along strike and at various depths.

In 2010 samples were collected from more recent diamond drill holes to represent lenses of the Minapampa zone. Intercepts from holes DDH09-44, DDH09-45, DDH09-46, DDH09-52, DDH09-53, DDH09-54, DDH09-57 and DDH09-61 were identified and prepared and packaged using the same methodology as the intervals from the first round of sampling.

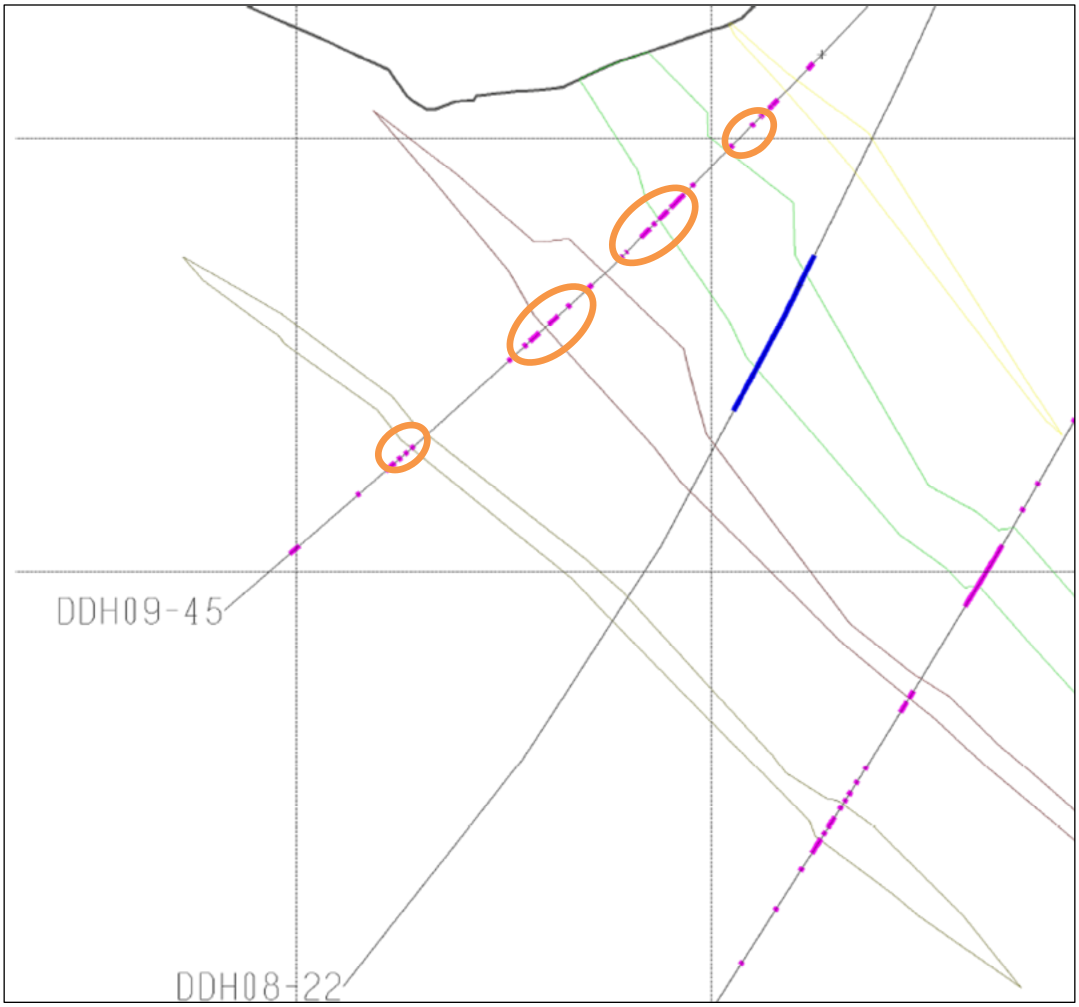
The mineralized intercepts selected were aligned with the lenses as best as could be determined based on a number of sections which presented the drill holes and the supposed location of the various lenses.

Figure 11-2 shows an extract from one of the drill cross-sections and presents drill holes DDH09-45 and DDH08-22. The various ore lenses identified as part of the PA work are defined in the figure by yellow, green, maroon and khaki polygons.

Drill hole DDH08-22 has a number of intervals shaded blue, and this represents the intervals that were used for composite OL22-A. It can be seen that these intervals approximate a representation of the “green” lens in Figure 11-2. Consequently this “down hole” composite is in fact a representation of this lens.

DDH09-45 can be seen to cut all four lenses. It appears from the grade-carrying intervals shown as pink intervals on the drill hole plot, that there is in fact some displacement down-hole in this section. The orange ovals represent the intervals considered to actually represent the lenses. If the drill holes selected for the second round of sampling were found to display this sort of displacement, the intervals outside the lens polygon but continuous with those in it were included and were considered to represent the lens.

Figure 11-2: DDH09-45 Intervals and Lens Associations



Twenty composites were derived from the drill holes selected for the second round of sampling. The composites were named such that the name provided information on the hole from which it was sourced (OL45). If the drill hole was representing multiple lenses (such as DDH09-45) a letter prefix was also added (A, B, C etc) for each consecutive group of intervals. Finally a three letter suffix was including which nominated which lens the composite was representing (GRN for green, MRN for maroon, KHI for khaki etc).

Table 11-1 summarises the 20 composites. The table provides the calculated grade of the composite based on the intervals used as well as the screen fire assay data and the resultant composite head grade calculated from the screen fire assay information.

Table 11-1: Metallurgical Sampling Composites from the Second Campaign

		Screen Fire Data		
Comp Name	Grade Au, g/t calcd based on intervals	-75 um Au average g/t	+75 um Au %	Composite calc'd head, g/t
OL44-A GRN	3.34	2.60	3%	2.66
OL44-B MRN	4.02	2.96	20%	3.66
OL44-C PNK	3.98	4.26	23%	5.48
OL44-D KHI	4.50	2.82	6%	2.98
OL45-A YLW	1.92	1.16	9%	1.25
OL45-B GRN	3.60	3.88	9%	4.23
OL45-C MRN	3.26	4.01	28%	5.51
OL45-D KHI	2.48	2.22	5%	2.31
OL46-A RED	1.50	0.33	12%	0.37
OL46-B MRN	5.32	1.67	25%	2.22
OL46-C PNK	2.36	1.08	9%	1.18
OL52-A GRN	5.11	2.54	32%	3.67
OL53-A GRN	13.73	2.06	5%	2.17
OL53-B MRN	4.12	3.17	10%	3.51
OL54-A GRN	4.49	1.69	9%	1.84
OL57-A RED	3.01	1.29	3%	1.32
OL57-B GRN	2.03	1.33	5%	1.39
OL61-A GRN	7.66	6.53	8%	7.04
OL61-B MRN	5.01	5.23	8%	5.61
OL61-C KHI	3.46	3.90	11%	4.35

As had been found in the first round on samples, there was considerable variation in the grades based on intervals and on the screen fire assay data. In this case however the scatter was both positive and negative. It was also apparent coarse gold is present and this can be expected to explain the variation, at least in part.

Referring to Table 11-1, it can be noted that there are seven composites representing the “Green” lens and five representing the “Maroon” lens. These are the two most significant lenses in the deposit and so it is expected there would be a bias with regard to representation of these lenses.

The major lenses, Green, Maroon and Khaki were also represented by combining the various down hole composites representing these lenses. This resulted in “lens” composites such as GRN-01, MRN-01 and KHI-01. This mix of composites allows testing to be conducted by lens, by intersection as well as down hole in some instances.

11.4.4 Third Metallurgical Sampling Program – 2011

In February of 2011 a third sampling program was undertaken. This program included collection of metallurgical and environmental samples from drill core and the samples represented areas from Minapampa, Minapampa East, and Cucuroyoc. The Minapampa samples were selected to represent "in-fill" regions of the Minapampa area that had not previously been represented in metallurgical testing.

The environmental samples were sub-sampled and subject to acid generation testing in Argentina prior to the rejects at a 3.35 mm crush being dispatched to Ammtec. These samples were composited by Ammtec and used for metallurgical test work where large sample masses are required, including thickening, filtration and paste backfill work.

The metallurgical samples collected at the same time were composited to provide a number of composites for variability testing and also to explore the metallurgical behaviour of the mineralization in Minapampa East and Cucuoyoc. These metallurgical programs were still underway at the Report filing date.

Some 22 composites were made up for "Bulk" samples tests (ex-environmental samples) and 15 "Metallurgical" composites. Designated "B" and "M" respectively.

11.5 Density Determinations

The Ollachea database contains 726 bulk density determinations. A total of 111 of these determinations are within the mineralized zones.

All bulk density determinations were completed used 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 was recorded for each sample, and was used to determine the spatial location.

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

11.6 Databases

MKK has no formal database for the collation of data. A series of Excel spreadsheets are used to store the data. MKK is in the process of implementing a commercial database for the storage of a 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 database when operational.

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

11.7 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 systems in place are of industry standard, and are adequate and appropriate for use in Mineral Resource estimation.

12.0 Data Verification

12.1 Independent Verification

In 2009 Barry Smee conducted and independent audit of MKK sampling procedures and of the preparation and analytical laboratories used (Smee 2009). There were a number of issues that required addressing. These issues were addressed by MKK subsequently, and have been documented; Coffey (2010, 2011a).

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), 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 (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 practise and are considered entirely appropriate and acceptable for use in the technical report.

The database currently held by MKK is a collection of Excel spreadsheets, which captures a whole variety of 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 complied as new data is collected. MKK is in the process of implementing a new commercially available database, which will use the database collated by Coffey Mining.

12.2 QA/QC

MKK’s QA/QCprogram 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 has made eight gold standards (8001 to 8009) of various grades. Coffey Mining (April 2010), identified issues with standards 8001 to 8004, and they are no longer used.

Coffey Mining considers that 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 (-2.6%). From the 4 May 2010 to the 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, though there is a slight negative bias, this has been exaggerated by a possible misallocated standard submitted towards the end of May 2010.
- 8008 Similar to 8007, generally expected values are returned, a possible misallocated sample was included in early November 2010.
- 8009 Overall good accuracy with expected value, with a very slight positive bias (+0.2%).

MKK used blanks to control sample contamination in the laboratory. Until August 2009 a pulverized blank 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.

12.2.2 MKK Duplicates

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 during the latest infill program, 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 (633 results) only 69% pass a 30% half absolute relative difference (HARD), whereas for the ½ versus ½ core samples (164 results) 70% pass a 30% HARD. The ¼ versus ¼ core samples (205 results) only 69% 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 no noticeable change due to the different sample volumes. There is a negative bias in the higher grade values (> 10 g/t Au), indicating the possible presence of coarse gold; although the mean of the field duplicate is higher for both data sets than for the original 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 recommends that this ½ core versus ¼ core duplicate be discontinued, in infill drill areas, as comparing different sample sizes does not produce conclusive results.

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.

Pulp Duplicate

During the 2008 and 2009a drill programs, CIMM 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 CIMM (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 were sent to ALS Chemex laboratories in Santigo, Chile from the 2010 drilling campaign. The pulps were analysed using the same method as used by CIMM 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.

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 was that there was no real nugget effect in the fine (-150 mesh) fraction. 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 12-2 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.

12.5 Comments on Item 12

Coffey Mining considers that the current drilling and sampling procedures undertaken by MKK are adequate for the purposed used in the technical report.

13.0 Mineral Processing and Metallurgical Testing

Three programs have been completed to this time and a fourth is underway at Report filing date. The first program was conducted by Kappes Cassiday and Associates (KCA) of Reno, Nevada, United States of America in 2009 and early 2010. The second and third programs have been completed by Ammtec Limited (Ammtec) of Perth, Western Australia in 2010 and Ammtec are currently undertaking the fourth program (August 2011).

Test work has focussed on the discovery zone known as “Minapampa”. Other zones known as “Minapampa East” and “Cucaroyoc” (western extension of Minapampa) have been sampled for the current metallurgical program.

A number of test work programs have been undertaken and completed:

- Program by Kappes Cassiday on the first sample set.
- Program by Ammtec on the first sample set and on the second sample set.
- Program by Ammtec predominantly on the second sample set.
- Thickening test work by Outotec.
- Filtration test work by Outotec.
- Acid Base accounting.

In addition there is work continuing:

- Program by Ammtec on the second and third sample set.
- Paste test work program being conducted by AMEC.

This section of this report summarises the outcomes and findings of the first five programs as well as provides comment on the current Ammtec program.

13.1 Kappes Cassiday Program

The KCA test work is summarised from the report “Ollachea Project Report of Metallurgical Test Work” (KCA, 2010).

The KCA work included some investigative work seeing as this was the first significant test work program conducted on the Ollachea ore types. The aim of the program was to basically gather data so as to understand the basic characteristics of the Ollachea ores and to identify a likely flow-sheet.

KCA undertook:

- Sample receipt and preparation of composites including head analyses.
- Comminution testing.
- Gravity testing.

- Agitated leaching and bottle roll leach testing.
- Magnetic evaluation.
- Flotation testing.

The program quickly identified a preg-robbing issue with the Ollachea ores that resulted in sub-economic cyanide leach extractions for direct leaching. Flotation testing showed a concentrate could be achieved however mass pulls to concentrate were high, significant tails losses were prevalent and leaching of the flotation concentrate was still problematic.

13.1.1 Comminution Test work

Composite OL04-A was subjected to Bond Rod Mill Work Index (RWI) testing at Phillips Enterprises whilst another sample was subjected to Bond Ball Mill Work Index (BWI) determination at KCA. The RWI was determined at 13.84 kWh/t and the BWI 16.25 kWh/t.

The fact the RWI is lower than the BWI is atypical as it is normal for the RWI to be greater. A possible reason for this is that the Ollachea ore is hosted in a graphitic shale/slate and this material easily breaks at the coarser sizes reflected by the RWI. Once the shale/slate is broken it is the harder quartz grains that are driving the BWI at the finer sizes.

13.1.2 Head Analyses

Detailed head analyses were determined for the various composites. Of note it can be seen that:

- Gold assays were repeatable and range from 1.3 g/t to 3.57 g/t Au except for the high grade OL26-A composite which had an average assay of 18.74 g/t Au.
- Silver assays tend to be around 4 g/t and not as variable as the gold grades nor proportional, further suggesting electrum is not the dominant silver mineral.
- Iron and sulphur assays are consistent suggesting not much variability in iron sulphide levels.
- There is some mercury present and this needs to be considered in process design.
- The typical problematical elements for cyanidation are low with the potential exception of arsenic.
- Organic carbon levels are low when compared with the organic carbon levels determined for other composites generated in follow up programs.

13.1.3 Agitated Leach Test work

Agitated leach test work on the OL04-A composite at various grind sizes gave gold extractions of only 15% to 20%. These tests were the first “scoping” level tests on the

Ollachea ores with regard to cyanidation. Sodium cyanide consumption was low to moderate at 0.63 to 0.80 kg/t and hydrated lime consumption was low at 0.5 kg/t. These results presented a potential issue to the applicability of cyanidation as a flowsheet option.

13.1.4 Magnetic Separation

Magnetic separation of a sample of OL04-A was successful in recovering 50% of the gold into a concentrate representing 11% of the feed mass. Given the dominant sulphide mineral pyrrhotite is magnetic, this work suggested an association of gold grades with this mineral.

Magnetic separation did not appear to be an effective concentration step; however, there may be application for magnetic separation with regard to managing acid generation issues from tailings repositories.

13.1.5 Gravity Concentration

Gravity concentration techniques proved effective in that tests on the OL04-A composite and the high grade (possibly high gravity gold component) OL26-A sample recorded high recoveries to a relatively low mass pull to concentrate by way of a 3" Knelson concentrator.

Knelson concentrator performance was such that the OL04-A sample presented 69% recovery to 1.6% of the feed mass based on a low head grade of 1.69 g/t Au. The OL26-A sample presented 78% recovery to 1.9% of the feed mass with a calculated head of 14.05 g/t Au.

The gravity tails losses were still too high to allow a gravity-only flowsheet and further losses would be expected on leaching the concentrates. The work did suggest that inclusion of a gravity circuit was mandatory, and that opportunities may exist to enhance extraction with gravity supplementation and possibly concentrate selective additional processing (fine grind, oxidation, intensive leaching).

13.1.6 Flotation

Flotation was evaluated to determine if pre-concentration could provide a high enough recovery to allow a reduced capacity leach plant and/or if the preg-robbing component could be isolated from the gold mineralization.

The results from the first flotation tests resulted in large mass pulls being required to achieve even mediocre recoveries, with values of +30% mass pull for +80% recoveries being typical.

13.1.7 Bottle Roll Leach Tests

Bottle roll leach tests were conducted on samples of all of the composites as well as flotation concentrates. A number of "direct" tests were run where the test was run without any activated carbon present. A comparative series of tests with activated carbon present to compete with potential preg-robbing species was also undertaken.

The following points can be made with regard to the results:

- There is significant scatter in the calculated head assays which suggests free gold can be an issue in evaluating results – gravity recovery steps are required in both ongoing test work and gravity is a likely full-scale flowsheet requirement.
- CIL leaching provides much improved leach extractions/recoveries than direct leaching. The extractions/recoveries achieved present CIL as a practical flowsheet option.
- Grind sensitivity results are inconclusive based on the results of this suite of tests.
- Leach extractions achieved on flotation concentrates coupled with the recovery to the flotation concentrate suggest flotation as practised in this round of tests would most likely be less economic than CIL. Leaching of the flotation tail would be required in addition to the concentrate leach. This further suggests the only advantage of a flotation step would be to isolate sulphides, which may be advantageous with regard to tailings management.
- It would appear the use of NaOH for pH control reduces the sodium cyanide consumption.
- Further development of the flotation flowsheet is justified however the recovery to concentrate would have to be greatly improved to be competitive with CIL.

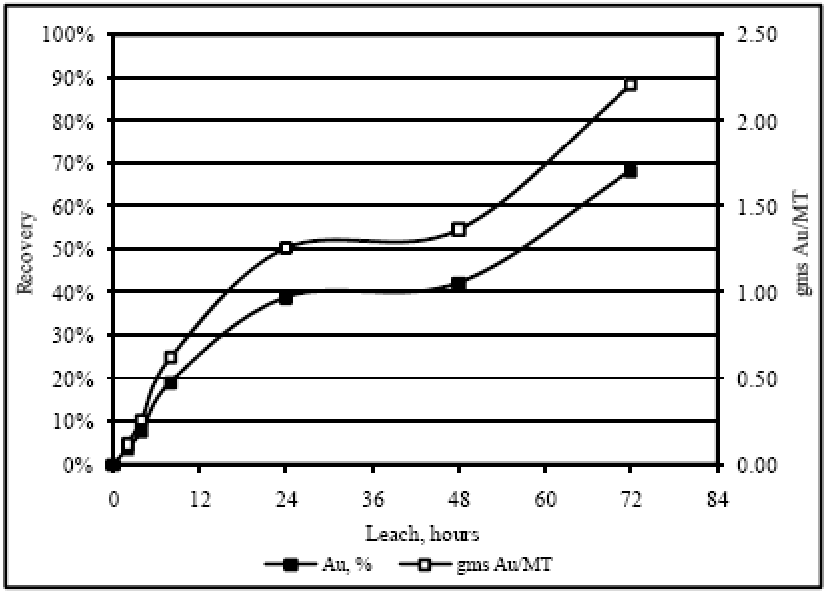
13.1.8 Leach Characteristics

A number of the leach tests conducted by KCA presented kinetics as is represented in the leach curve shown as

Figure 13-1. Whilst this graph is for gold, similar but less pronounced behaviour was similarly noted for silver. This curve is from a direct leach on sample OL22-A (test 42247A). The plateau in recovery from 24-48 hours was noted and discussed with KCA but could not be resolved. Subsequent leach test work did not have such a plateau.

Review of the detailed test work log sheets did not provide any obvious reason for this behaviour (cyanide level, pH, dilutions etc) and the cause was not identified. Such behaviour has not been seen in the programs conducted by Ammtec to date.

Figure 13-1: Gold Leaching Behaviour



13.1.9 Multi-stage Flowsheet

KCA undertook a series of tests to evaluate the CIL and flotation options on the OL04-A (low-grade) and the OL26-A (high-grade) composites. Three flowsheets were evaluated, each of which included a gravity step. The gravity tail was subjected to either a CIL bottle roll test (BRT), a CIL bottle roll test pre-sparged with oxygen or flotation followed by leaching of the flotation products and the flotation tails.

Table 13-4 provides a summary of the recoveries achieved by stage of the flowsheet. The gravity step extraction is that of the super-concentrate from the tabling of the rougher concentrate. The additional gravity stages recovery is assumed to have gone to the next stage of extraction for each option so as to simplify the summary table.

Table 13-1: Summary Extractions by Stage

Flowsheet	Cumulative Extraction OL04-A	Cumulative Extraction OL26-A
Gravity step (super concentrate).	39%	58%
Option A: CIL bottle roll.	93%	99%
Option B: Pre-sparged CIL bottle roll.	94%	99%
Option C: Flotation with leach of Conc 1.	(8.1% of mass) 87%	(6.6% of mass) 92%
Option C: Flotation with leach of Conc 2.	(5.8% of mass) 88%	(5.2% of mass) 94%
Option C: Flotation with leach of Conc 3	(10.9% of the mass) 89%	(9.3% of the mass) 94%
Option C: Flotation with leach of Conc 4.	(11.1% of the mass) 90%	(11.6% of the mass) 95%
Option C including leach of the flotation tails.	(remnant 64.2% of the mass) 93%	(remnant 67.3% of the mass) 99%

This program provided the following salient points:

- A gravity step in the flowsheet is supported for a number of reasons including removal of coarse gold but importantly, reduction in the solution concentration in the leach tanks to reduce losses to preg-robbing components.
- Use of CIL processing can provide acceptable extractions confirming this is a valid flowsheet.
- Use of flotation requires large mass pulls to concentrate (+30% in these two examples) and leaching of the flotation tails to achieve extractions equivalent to a CIL flowsheet. Use of a flotation flowsheet could therefore only be supported if flotation selectivity and recoveries can be significantly improved upon.
- Such high mass pulls to a flotation concentrate negate benefit from isolating sulphides.

13.2 Ammtec Test work Program – Part A.

The Ammtec Part A program (Ammtec, 2010) included further evaluation of flotation as well as CIL processing. The composites used for this campaign are listed in Table 11-1.

In addition to some 20 variability composites compiled, representing various areas of the various lenses, a number of lens composites were made up which were used for the Part A program.

13.2.1 Bond Ball Mill Work Index

A BWI determination was undertaken on composites OL44-A GRN and OL44-B-MRN. These composites represent the "Green" and "Maroon" lens down the same drill hole, DDH09-44. Both samples returned a BWI of 18.0 kWh/t (closing size 106 µm) suggesting similar physical characteristics in at least this region of the two most significant lenses.

This value is some 11% higher than the value KCA returned for composite OL04-A at 16.25 kWh/t at a closing size of only 75 µm. This suggests some variability in work index can be expected.

13.2.2 Test work on OL04-A Composite (ex KCA)

Ammtec undertook a series of tests on the OL04-A composite. The aim of this program was in part to confirm the KCA test results and the conclusions drawn from the earlier KCA work. In addition, use of the OL04-A composite allowed direct comparison of results from some new techniques (such as nitrogen flotation) to be directly compared with the existing testwork database.

Grind sensitivity work on sample OL04-A investigated the P80 grinds of 180, 125 and 75 µm under CIL conditions. The extractions were all within 3% of each other (84.1%, 85.9%, 86.9% respectively) and whilst suggesting a trend, the work was considered inconclusive due to the closeness of the results. Of note was increasing sodium cyanide consumption with finer grind, increasing some 0.8 kg/t between the three grinds.

A series of gravity, carbon pre-flotation tests to remove preg-robbing components and cyanidation of the concentrate was conducted at various grind sizes. Results were lower than whole of ore CIL leaching and the grind sensitivity was again inconclusive.

Work was also conducted to suppress carbon using D723 carbon depressant. Some 94% of the gold was recovered to a high mass pull concentrate of 15% and 17% of the organic carbon was recovered. Again the results were not as effective as whole-of-ore CIL leaching suggesting this option was not advantageous.

The fact the organic carbon recovery was the same as the mass pull within two percent suggested no success in depressing this component.

Gravity separation, carbon pre-float and then flotation of a concentrate was conducted using nitrogen as the sparging gas. The results were not as good as previous “conventional” flotation tests as consequently nitrogen tests were taken no further.

In summary, the Ammtec work confirmed the level of results achieved by KCA on this sample, and suggested whole of ore CIL was still a preferred flowsheet with regards to overall extraction.

13.2.3 Lens Composites

Three lens composites representing the major ore sources were subjected to whole-of-ore cyanidation and flotation testing to establish the most suitable flowsheet by lens type. The three lens composites GRN-01, MRN-01 and KHI-01 were considered representative of the major ore types.

Table 13-9 summarises a series of Gravity/CIL tests at various grinds. The extraction at $t = x$ represents the overall recovery including the gravity component.

These results indicate a level of grind sensitivity, most significant for the MRN-01 and KHI-01 composites. There would also appear to be an increase in sodium cyanide consumption at the finer grinds. The difference in extraction between the P80 of 90 µm grind and the 53 µm grind is minor, suggesting the optimum grind size is likely to be in this range.

Also of note is the incremental extraction achieved between 24 and 48 hours which is significant and suggesting a +24 hour leach time is required. Given the gravity step prior to leaching, this long leach time can be expected to be due to some other cause apart from the presence of coarse gold in the leached sample.

Table 13-2: Summary Lens Composite Grind Sensitivity to Whole-of-Ore CIL

Sample Identity	Test No.	Grind Size P_{80} (µm)	% Au Extraction @ Hours					Consumption (kg/t)	
			Gravity	2	8	24	48	Lime	NaCN
Composite #21 (GRN-01)	HS22571	125	10.13	64.96	84.65	89.47	91.78	0.31	2.61
	HS23273	90	19.89	67.25	77.51	87.04	91.81	0.36	2.69
	HS23274	53	17.04	61.95	78.61	88.06	92.78	0.39	3.01
Composite #22 (MRN-01)	HS22572	125	28.99	62.42	79.87	84.67	86.98	0.30	1.84
	HS23275	90	28.59	62.77	73.63	84.68	90.20	0.36	2.09
	HS23276	53	31.33	67.59	77.02	87.02	92.02	0.35	2.16
Composite #23 (KHI-01)	HS22573	125	31.96	60.30	71.09	79.04	81.58	0.30	1.85
	HS23277	90	28.57	57.63	70.17	82.71	88.98	0.21	1.73
	HS23278	53	30.34	58.81	71.10	83.39	89.54	0.14	1.72

A series of flotation tests were conducted. The first set investigated a pre-float of carbon to remove the problematical preg-robbing component and the results were quite consistent for each lens composite with some 14% to 16% of the feed mass reporting to the concentrate along with around 30% of the organic carbon. Some selectivity was therefore achieved however a substantial amount of gold also reported to the pre-float which would be a loss for the flowsheet if practically applied.

The results show that to achieve a high gold recovery to a flotation concentrate requires a high mass pull however with some 7.9% of the gold still reporting to tails and a further 6.8% lost to the pre-float, the extraction will not be as high as the whole of ore CIL option even if the concentrate leach were 100% efficient.

The deportment of total carbon and organic carbon are effectively the same which suggests there is no preferential separation for the carbon species.

Carbon depression flotation tests were conducted on the GRN-01 and MRN-01 composites. The results show even with very high mass pulls to concentrate there are still tail losses exceeding 10%. Again, the leach extractions on the concentrates will be less than 100% resulting in a final tail much greater than the equivalent whole of ore CIL option.

The results of the flotation tests are not as positive as the whole-of-ore CIL tests and so it was decided that the CIL flowsheet would be taken forward as the basis of the Pre-feasibility Study and further test work.

13.2.4 Cyanidation Flowsheet Development

A series of leaching tests were undertaken on the GRN-01 and MRN-01 composites to compare with the whole-of-ore CIL tests. A grind size of P80 = 125 µm was selected for the work.

Direct cyanidation leaching (not CIL) was compared with resins, use of kerosene "blanking" and direct leaching, 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. All of these methods being shown in the treatment of preg-robbing ores as providing some sort of benefit.

The baseline direct cyanidation provided a 48 hour extraction of 49% compared with 92% for the GRN-01 composite subjected to CIL conditions at 50 g/L carbon population. Similarly the direct cyanidation of the MRN-01 composite gave a 48 hour extraction of 34% compared to 82% for CIL leaching. Use of these other techniques could be directly compared with the direct leach and the CIL results already achieved.

Resin-in-pulp (RIP) tests gave improved results over the 50 g/L carbon concentration CIL tests when tested at a resin population of 25 g/L. However, RIP at 12.5 g/L gave slightly poorer results. RIP would therefore be considered a potential flowsheet based on these results. The issue with RIP is that it is still quite a young technology in the gold industry and whilst practical and under industrial use, it was discounted from further evaluation at this time.

The use of kerosene to blank the preg-robbing species was found to be effective with around a 20% jump in extraction.

Use of sodium hydroxide similarly provide beneficial although not as effective as the kerosene. Sodium hydroxide has some disadvantages apart from a higher cost than lime based pH modifiers. It can adversely influence settling and filtration characteristics as well as carbon kinetics. These influences need to be considered if sodium hydroxide is to be considered for the flowsheet.

Pre-leach oxygenation provided some extraction benefit, albeit minor. Pre-aeration is a relatively low cost flowsheet addition and it can reduce cyanide consumption if partial oxidation of the ore is experienced. For ores with reactive sulphides such as pyrrhotite, it is often advantageous in that it retains higher dissolved oxygen levels once leaching starts and assists in the initial leach rate.

The use of a low pulp density gave a result contrary to what was expected. Use of a low pulp density reduces the concentration of the gold in solution and should therefore reduce the influence of the preg-robbing species. During this test the sodium cyanide concentration was maintained at levels similar to those of the other tests, which is partially why the cyanide consumption was so high, so cyanide should not have been restrictive. All other conditions were as expected.

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

13.2.5 Filtration Tests

As it was considered likely filtration of the tailings may be required, Ammtec undertook some scoping level filtration tests.

The filtration tests show some variability however this apparent variability is complicated by different cake thicknesses for the various tests. There is a general improvement using lime based pH modifiers over sodium hydroxide. There are conflicting results for the various grind sizes. What is apparent is that there would be a large filter area required if vacuum filtration were to be employed based on the form plus dry filtration rates for all cases tested.

13.2.6 Diagnostic Analyses of Select Leach Residues

CIL residues from GRN-01 and KHI-01 (Comp #21 and Comp #23) were subjected to diagnostic leaching to identify where the residual gold was reporting. Diagnostic leaching of preg-robbing ores is difficult as each re-cyanidation leaching stage up until the point of undertaking a roasting step must be undertaken as a CIL test, which means continued competition with the preg-robbing components themselves. This must be considered when evaluating the diagnostic results.

The results are summarised in Table 13-3 and is taken from Ammtec (2010).

Table 13-3: Diagnostic Leach Results

Diagnostic Stage (sequentially)	Description	Leach Residue HS22571: Composite #21 Mode of Au		Leach Residue HS22573: Composite #23 Mode of Au	
		g/t	Dist'n (%)	g/t	Dist'n (%)
CIL Cyanidation Leach	Free/Cyanidable Gold	0.240	43.53	0.360	48.66
Dilute HCl Digest/Cyanidation	Carbonate Locked Gold	0.065	11.86	0.064	8.66
Dilute HNO_3 Digest/Cyanidation	Arsenical Mineral (Arsenopyrite) Gold Content	0.119	21.62	0.115	15.60
Desorption/ Roast/Cyanidation	Graphitic/Organic Carbon Gold	0.046	8.32	0.089	12.00
Aqua Regia Digest	Pyritic Sulphide Mineral Locked Gold Content	0.045	8.10	0.056	7.59
Total Fire Assay Smelt	Silicate (Gangue) Encapsulated Gold Content	0.036	6.58	0.055	7.50

The diagnostic leaching showed similar results for both composites, again supporting previous indicators that the metallurgical responses of the various lenses are very similar.

Most of the residual gold reported as free/cyanidable. Some caution is warranted here as it is possible, and likely, that much of this is actually gold that has leached in the original CIL test and adsorbed onto the preg-robbing species. Following removal of the high ionic strength leach solution and washing of the CIL residue, then re-leaching at high cyanide concentration in the diagnostic stage has allowed desorption. This result can be considered indicative of the losses still occurring due to the preg-robbing species even under CIL conditions and indicates the values that can at least in part be recovered if the influence of the preg-robbing species is further reduced.

The diagnostic assessment suggests on the whole that most of the gold is leachable, there is little tie up in sulphides and that losses to silicates are low which in turn suggests even at a 125 µm grind size, liberation is good.

13.3 Ammtec Test work Program – Part B.

The Ammtec Part B program included further evaluation of CIL processing. In addition, CIL processing was conducted on a number of variability composites that were originally put together and used to generate the lens composites.

13.3.1 Variability

To assess the variability of the ore, each of the variability composites was subjected to CIL leaching at a carbon population of 50 g/L for 72 hours. Test results are summarised by Table 13-4 and are from Ammtec (2010).

Table 13-4: Variability CIL Results

Sample Identity	Test No.	Grind Size P_{80} (µm)	Calc'd Head (g/t)	% Au Extraction		Leach Residue (ppm)	Consumption (kg/t)	
				Gravity	72		Lime	NaCN
Comp #1: OL44-A GRN	HS23778	75	3.74	9.74	95.98	0.15	0.45	5.26
Comp #2: OL44-B MRN	HS23779	75	5.41	39.80	95.38	0.25	1.16	2.64
Comp #3: OL44-C PINK	HS23780	75	4.23	42.10	93.86	0.26	0.43	2.89
Comp #5: OL45-A YLW	HS23781	75	1.17	38.11	89.75	0.12	0.64	2.76
Comp #6: OL45-B GRN	HS23782	75	4.61	40.46	92.85	0.33	0.34	2.85
Comp #7: OL45-C MRN	HS23783	75	6.13	59.99	95.92	0.25	0.33	2.85
Comp #8: OL45-D KHI	HS23784	75	2.11	31.53	80.12	0.42	0.30	2.56
Comp #9: OL46-A RED	HS23785	75	0.58	29.39	62.24	0.22	0.41	2.83
Comp #12: OL52-A GRN	HS23786	75	5.54	70.70	94.95	0.28	0.53	3.61
Comp #13: OL53-A GRN	HS23787	75	2.16	26.34	65.26	0.75	0.26	2.81
Comp #14: OL53-B MRN	HS23788	75	2.90	50.49	87.57	0.36	0.43	2.93
Comp #15: OL54-A GRN	HS23789	75	1.77	38.55	84.75	0.27	0.41	2.67
Comp #16: OL57-A RED	HS23790	75	1.69	39.79	89.35	0.18	0.36	2.67
Comp #19: OL61-B MRN	HS23791	75	4.57	31.40	85.34	0.67	0.35	2.72
Comp #20: OL61-C KHI	HS23792	75	3.22	33.96	85.73	0.46	0.25	2.70

The results present some variation; however, they do present the case that a CIL flowsheet is capable of providing high extractions. Cyanide consumption is high by industry standards and this is expected to be exacerbated by the high population of activated carbon.

13.3.2 Adjusting Leach Conditions

A second lens composite was generated called GRN-02 because the original lens composites were near exhausted. This composite used the same recipe as GRN-01.

A number of leach tests were conducted on GRN-02 at various carbon populations under CIL conditions to assess sensitivity to population. In addition, tests were undertaken with carbon loaded to approximately 1,500 g/t gold to investigate the influence this would have on extraction.

Results show that the relationship between carbon population and loading will have to be understood to effect final process design given the sensitivity of the extraction to both parameters. Whilst the extraction is reduced at lower carbon populations, there is a reduction in the cyanide consumption which supports the concept that the carbon is having a significant impact on cyanide consumption.

The shape of the leach time versus gold extraction curves with fresh and loaded carbon in 10 g/L, 25 g/L and 50 g/L concentrations suggest that there may be some adsorption-desorption occurring. The curves for the loaded carbon in particular suggest this may be happening. This would explain why such a long leach time is required for many of the leach tests when the diagnostic work had suggested liberation was good and gravity recovery before CIL leaching had removed coarse (slow leaching) gold.

CIL tests at 50 g/L carbon population were conducted at a pH of 12.5, one with hydrated lime and the other with sodium hydroxide suggesting that there is an opportunity to reduce cyanide consumption at elevated pH; however, the cost of the pH modifier will need to be considered as the doses here were very high.

There seems to be a benefit of some 0.05 g/t Au extraction from these two tests over the baseline CIL test on the same sample. This is supported by the knowledge that high pH often suppresses preg-robbing species, a fundamental parameter used to accelerate desorption in elution circuits Further investigation is warranted with regards to pH and the type of modifier used.

Manipulation of sodium cyanide concentration was undertaken where the same initial dose was applied and the same background maintained, but at *t =24* hours a spike was added. The results suggest a subtle improvement in extraction but at a higher cyanide consumption. Given the variability in assay results the outcome is arguable.

Composite #27 was compiled as a ratio of 3 parts GRN-02 to 1 part MRN-02. This was considered appropriate to reflect the type of blend of lenses likely to be experienced during operations, especially seeing as the Green and Maroon lenses are dominant and make up the bulk of the ore. This composite was then used to review a number of leach parameters as well as determine the influence of other new parameters that had not been explored up until this time.

A series of tests were conducted all at a grind P80 of 75 µm. Table 13-20 provides a summary of the results of this program that investigated. Variables explored were:

- Carbon population
- Kerosene blanking with and without carbon (CIL).
- Pulp density.
- Cyanide concentration.
- Pre-oxygenation.

- Leach temperature.

The results are summarised as follows:

- The results confirmed the sensitivity to carbon population (tests 24515 to 24517).
- Kerosene alone at a dose of 1 kg/t was found to provide excellent extraction (residue grade of 0.23 g/t) without any carbon present – test 24518.
- Low pulp density also provided an excellent extraction (residue grade 0.23 g/t) but at elevated sodium cyanide consumption - test 24519.
- Elevated sodium cyanide was found beneficial but with elevated consumption - test 24521. Whilst starvation cyanidation techniques gave a poor gold extraction of only 30% but at a very low consumption – test 24522.
- Pre-oxygenation (test 24523) did not appear to provide any appreciable advantage in either sodium cyanide consumption or extraction.
- Elevated temperatures of 60º and 90ºC provided the best extractions noted to date in the presence of carbon with residues of only 0.18 g/t gold. Elevated temperatures without carbon present still gave a 25% extraction benefit over ambient temperature direct leaching. Refer to tests 24524 to 24527 and the baseline test 24515.
- Leaching with a 0.5 kg/t kerosene dose at 10 g/L carbon population was found to be as effective as a direct leach with 1.0 kg/t kerosene dose or CIL with 50 g/L and no kerosene. Refer to tests 24517, and 24720 to 24722. These results revealed the options available to negate the influences of preg-robbing components.
- Leaching at elevated temperatures of 30 and 50 degrees Celsius in the presence of 10 g/L carbon gave low residues of 0.24 and 0.22 g/t gold respectively. Refer to tests 24723 and 24724. Typically the grinding circuit could be expected to operate in this range of temperatures and so this type of behaviour could be expected in the full-scale plant.
- A low carbon population of 5 g/L was found to result in elevated residue grades however the addition of 0.1 kg/t of kerosene lifted the extraction and reduced the residue grade by more than 0.05 g/t gold. Refer to tests 24790 and 24791.

Table 13-5: Composite #27 Leach Variables (P_{80} 75um)

Test AMMTEC	Assay head g/t	Calc head g/t	Residue g/t	PD %w/w	Extn % Gravity	Extn % 48	Extn % 72	NaCN kg/t 72 h	CaO kg/t 72h	Carbon g/L	Kero kg/t	Temp C
24515	3.04	2.93	1.83	40%	9.72	36.8	37.6	2.26	2.55			Ambient
24516	3.04	2.91	0.28	40%	9.81	89.1	90.4	2.98	2.67	10		Ambient
24517	3.04	2.97	0.24	40%	9.59	85.6	91.9	3.95	2.96	50		Ambient
24518	3.04	2.97	0.23	40%	9.60	91.0	92.3	2.67	1.32		1.00	Ambient
24519	3.04	3.05	0.24	10%	9.35	89.9	92.1	4.13	2.63	10		Ambient
24520	3.04	3.08	0.31	50%	9.27	87.2	89.9	2.35	2.55	10		Ambient
24521	3.04	3.03	0.25	40%	9.42	90.5	91.7	3.52	2.40	10		Ambient
24522	3.04	3.02	2.12	40%	9.45	25.9	29.7	0.32	3.25	10		Ambient
24523	3.04	3.10	0.30	40%	9.21	88.5	90.3	2.82	3.39	10		Ambient
24524	3.04	3.28	1.18	40%	9.02	61.2	62.7	2.99	7.70			60
24525	3.04	3.28	1.16	40%	9.16	56.9	62.7	3.30	6.38			90
24526	3.04	3.11	0.18	40%	9.17	93.0	94.2	3.70	8.78	10		60
24527	3.04	3.15	0.18	40%	9.06	93.1	94.3	3.68	9.29	10		90
24720	3.28	2.93	1.36	40%	9.74	53.6	53.5	1.26	3.11		0.10	Ambient
24721	3.28	2.99	1.05	40%	9.55	64.8	64.8	2.12	3.31		0.50	Ambient
24722	3.28	3.09	0.27	40%	9.24	90.0	91.2	2.68	3.85	10	0.50	Ambient
24723	3.28	3.09	0.22	40%	9.22	91.7	92.9	2.97	3.30	10		50
24724	3.28	3.04	0.24	40%	9.39	90.8	92.1	2.89	3.30	10		30
24790	3.28	2.85	0.33	40%	10.02	84.8	88.4	2.69	3.54	5		Ambient
24791	3.28	2.88	0.26	40%	9.95	89.5	91.0	2.68	3.82	5	0.10	Ambient

Figure 13-2 and Figure 13-3 summarise a number of these results graphically.

Figure 13-2: Effects of Carbon Population and Kerosene Dose

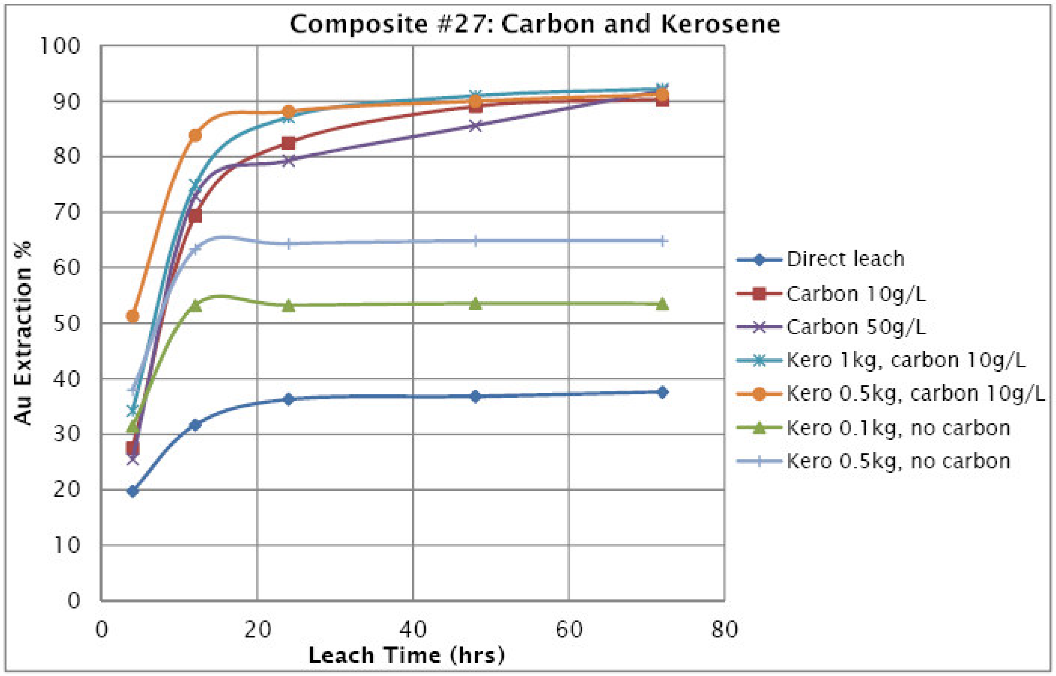
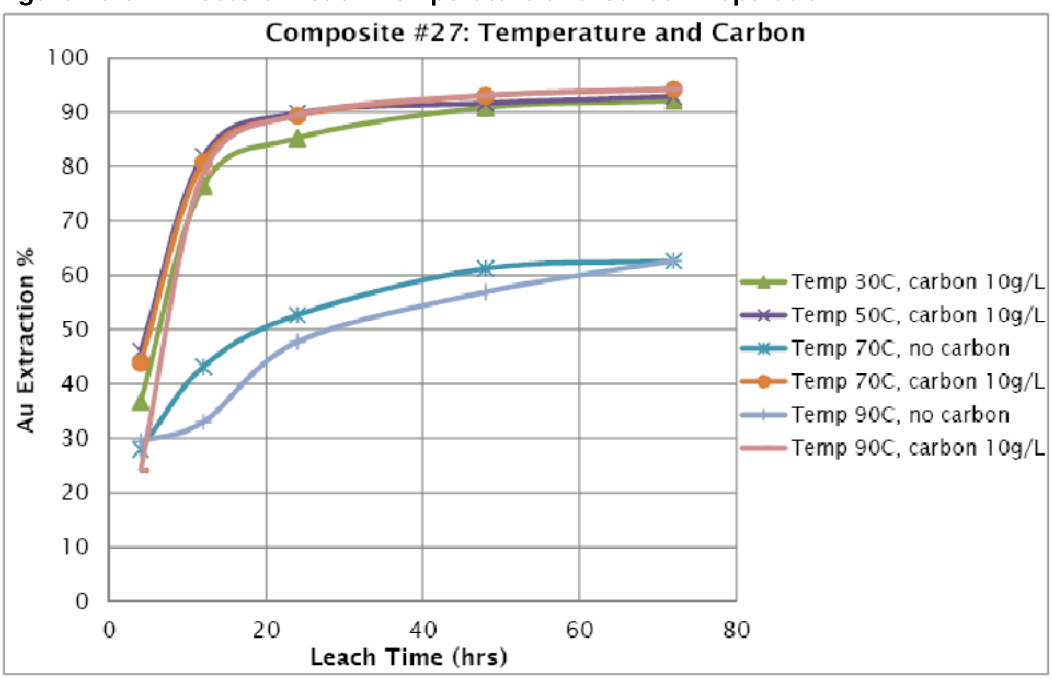


Figure 13-3: Effects of Leach Temperature and Carbon Population



A number of viscosity determinations at different shear rates were undertaken on residues from Composite #27 leach tests. Generally the viscosity of the slurry is low.

13.4 Outotec Test work

Outotec conducted a thickening test work program and a filtration test work program on a sub-sample of the "bulk" composite. The sample exhibited poor settling and filtration characteristics resulting in a thickener and filter sizing larger than might have been expected for a plant of the capacity presented by this study. The thickening rate used for design being 0.25 $t/m^2/h$ and the filtration rate 152 $kg/m^2/h$.

13.5 Recovery Estimates

Metallurgical recovery has been estimated from the testwork carried out as part of the Ammtec leach variability program (Ammtec, 2011b). Metallurgical recovery is expressed as a function of gold head grade and tail grades according to the relationship:

$$\text{Metallurgical Recovery (\%)} = (Au_H - Au_R)/Au_H * 100$$

$$= (Au_H - (0.0209Au_H+0.2401))/Au_H * 100$$

Where Au_H is the head gold grade and Au_R is the tailings residual gold grade.

14.0 Mineral Resource Estimates

14.1 Basis of Estimate

Coffey Mining has estimated an Indicated and Inferred Mineral Resource for the Minapampa and Minapampa East Zones of the Project as at 31st May 2011. 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.

The interpretation of the mineralized zone and block model extend into the gap between the Oyeachea 3 and Oyeachea 4 concession boundaries shown on Figure 4-2. Mineralization in this gap was removed from the Mineral Resource estimate.

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 and Minapampa East area.

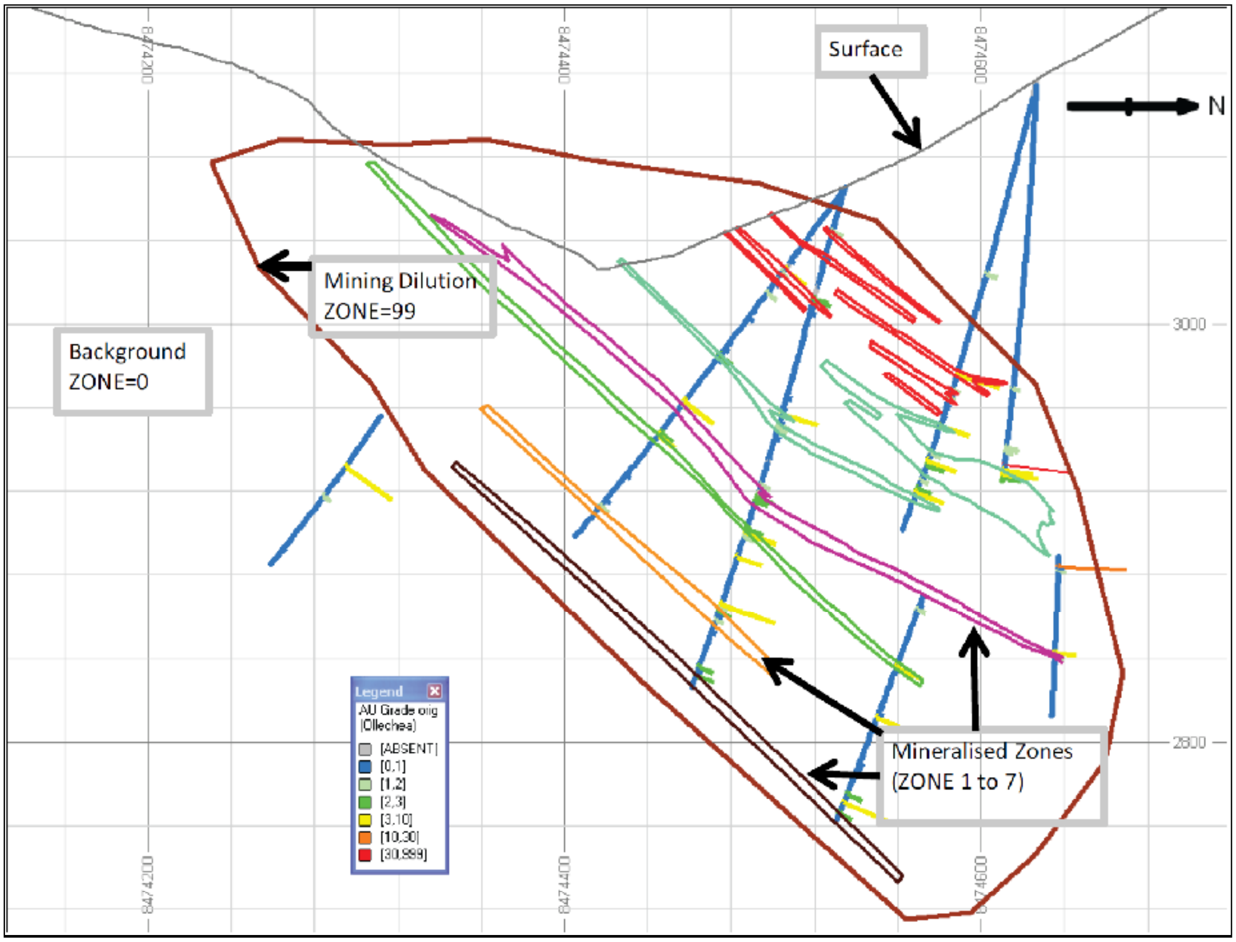
The topographic surface is based on a total station survey provided to Coffey Mining by MKK.

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). 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 – 339,200 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 wireframe defining the three-dimensional geometry of each interpreted feature. The interpretation and wireframe models were developed using the commercially-available 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 and DOMAIN: at Minapampa or Minapampa East.

14.3 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 are treated as barren (i.e. grade=0.0025 g/t Au).

14.4 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.

14.4.1 Summary Statistics – Raw Data

Table 14-1 presents the summary table of the raw statistics, grouped by mineralized zone for the combined Minapampa and Minapampa East domains.

Table 14-1: Summary Statistics of Raw Gold Assays by Zone

Zone	Description	Count	Min	Max	Mean	Std. Dev.	Variance	CV
0	Background	13052	0.003	82.54	**0.16**	1.25	1.57	7.94
99	Dilution Zone	12654	0.003	56.08	**0.20**	1.03	1.06	5.10
1	Min. Lens 1	199	0.030	42.55	**3.11**	4.10	16.78	1.32
2	Min. Lens 2	690	0.046	153.00	**5.56**	12.84	164.79	2.31
3	Min. Lens 3	323	0.026	29.31	**3.68**	4.36	19.01	1.18
4	Min. Lens 4	89	0.111	23.84	**2.91**	3.61	13.04	1.24
5	Min. Lens 5	522	0.003	121.45	**3.40**	7.61	57.98	2.24
6	Min. Lens 6	150	0.017	51.29	**2.89**	5.90	34.85	2.04
7	Min. Lens 7	64	0.031	17.04	**2.45**	2.38	5.68	0.97

14.4.2 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

Zone	Description	Count	Min	Max	Mean	Std. Dev.	Variance	CV
0	Background	16013	0.003	82.54	**0.11**	0.88	0.78	8.33
99	Dilution Zone	9825	0.003	28.11	**0.18**	0.56	0.31	3.07
1	Min. Lens 1	147	0.119	42.55	**3.27**	4.33	18.73	1.32
2	Min. Lens 2	467	0.137	153.00	**5.57**	11.74	137.94	2.11
3	Min. Lens 3	227	0.057	29.31	**3.63**	4.19	17.57	1.15
4	Min. Lens 4	63	0.111	23.84	**3.07**	3.80	14.41	1.24
5	Min. Lens 5	356	0.003	66.43	**3.15**	4.96	24.62	1.58
6	Min. Lens 6	126	0.017	51.29	**3.00**	6.27	39.28	2.09
7	Min. Lens 7	63	0.031	17.04	**2.49**	2.42	5.85	0.97
Total 1-7	MINZONE=1	1449	0.003	153.00	**3.97**	7.79	60.72	1.96

14.5 Density Assignment

The Ollachea database contains 726 bulk density measurements; there have been 47 new bulk density measurements collected within the mineralized material as compared to the 64 samples used previously (Coffey, 2011). Table 14-3 summarises bulk density determinations by ZONE.

Table 14-3: Summary Statistics of Density Determinations by Zone

Zone	Count	Min	Max	Mean	Median	Std. Dev.	Variance	CV
0	325	2.63	3.12	**2.81**	2.82	0.058	0.003	0.021
99	290	2.60	2.99	**2.79**	2.80	0.069	0.005	0.025
Total 0,99	615	2.60	3.12	**2.80**	2.81	0.064	0.004	0.023
1	10	2.71	2.89	**2.82**	2.83	0.052	0.003	0.018
2	33	2.61	2.93	**2.82**	2.83	0.080	0.006	0.028
3	25	2.72	3.11	**2.83**	2.83	0.073	0.005	0.026
4	2	2.66	2.83	**2.75**	2.66	0.118	0.014	0.043
5	31	2.75	2.96	**2.86**	2.86	0.052	0.003	0.018
6	5	2.66	2.86	**2.76**	2.73	0.085	0.007	0.031
7	5	2.66	2.87	**2.75**	2.68	0.102	0.010	0.037
Total 1-7	111	2.61	3.11	**2.83**	2.84	0.075	0.006	0.027

The data shows that the 2.80 g/cm³ dry in-situ bulk density value used for the previous resource estimate (Coffey, 2011) should be adjusted within the Mineralized zones (MINZONE=1).

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.

14.6 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 three outlier values resulted in a 15.6% 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 for each element, are presented in Table 14-4.

Table 14-4: Cut and Un-cut Composite Statistics

ZONE	Element	Uncut Number Data	Uncut Mean	Uncut Std. Dev.	Uncut CV	Upper Cut	Cut Mean	Cut Std. Dev.	Cut CV	Number Data Cut	% Change in Mean
1	Au(g/t)	147	3.27	4.33	1.32	**20**	3.12	3.18	1.02	1	-4.7
2		467	5.57	11.74	2.11	**40**	5.00	6.76	1.35	4	-10.3
3		227	3.63	4.19	1.15	**22**	3.55	3.77	1.06	4	-2.1
4		63	3.07	3.80	1.24	**18**	2.98	3.33	1.12	1	-3.0
5		356	3.15	4.96	1.58	**25**	2.96	3.28	1.11	3	-5.9
6		126	3.00	6.27	2.09	**20**	2.53	3.12	1.23	3	-15.6
7		63	2.49	2.42	0.97	**NC**	2.49	2.42	0.97	0	0.0
99		9825	0.18	0.56	3.08	**0.9**	0.16	0.21	1.33	218	-13.5
0		16013	0.11	0.88	8.33	**0.9**	0.07	0.15	2.31	289	-38.1

14.7 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 to 2 m sample intervals.

14.8 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 140 m for Zone 2 to 190 m for the combined variogram model. Ranges are greater than the average drill hole spacing which is a nominal 40 m x 40 m grid. For the low-grade zones, the total range in the major direction varied from 190 m for Zone 99 to 450 m for Zone 0.
- The relative nugget effect or short-scale variability in the mineralized zones ranges between 61% to 63%, displaying a high degree of short-spaced variability; this is common in narrow-veined gold deposits. For the lower-grade zones the nugget effect ranges between 38% for Zone 0 and 54% for Zone 99.

14.9 Block Model

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. Estimation/Interpolation Methods

The sample search strategy was based upon analysis of the variogram model anisotropy, mineralization geometry and data distribution.

Table 14-5: Interpolation Parameters

Zone	0	99	1	2	3	4	5	6	7
Major Range (m)	150	150	100	120	100	100	100	100	100
Semi-major Range (m)	90	100	80	100	80	80	80	80	80
Minor Range (m)	60	35	25	25	25	25	25	25	25
Major Dip (°)	0	45	45	0	45	45	45	45	45
Major Az (°)	100	20	20	110	20	20	20	20	20
Semi-major Dip (°)	45	0	0	45	0	0	0	0	0
Semi-major Az (°)	10	110	110	20	110	110	110	110	110
Minor Dip (°)	45	45	45	45	45	45	45	45	45
Minor Az (°)	190	200	200	200	200	200	200	200	200
Minimum Composites	8	8	8	8	8	8	8	8	8
Maximum Composites	20	20	20	20	20	20	20	20	20
Search Volume Factor	2	2	2	2	2	2	2	2	2
Minimum Composites	4	4	4	4	4	4	4	4	4
Maximum Composites	20	20	20	20	20	20	20	20	20
Search Volume Factor	-	-	3	3	3	3	3	3	3
Minimum Composites	-	-	4	4	4	4	4	4	4
Maximum Composites	-	-	16	16	16	16	16	16	16
Maximum Comps/DH	5	5	4	4	4	4	4	4	4

In addition to the high-grade capping strategy discussed in Section 14.6, an outlier-restriction strategy was applied to control the distance of influence of composites above a high-grade threshold. Composites above the thresholds listed in Table 14-8 were restricted to an ellipsoid of influence of 40 m x 40 m x 12.5 m.

Table 14-6: Outlier Restriction

Zone	Au Grade Threshold (g/t)	Restriction Distance (Major, Semi-major, Minor)
1	10	40 m x 40 m x 12.5 m
2	25	40 m x 40 m x 12.5 m
3	10	40 m x 40 m x 12.5 m
4	10	40 m x 40 m x 12.5 m
5	9	40 m x 40 m x 12.5 m
6	10	40 m x 40 m x 12.5 m
7	9	40 m x 40 m x 12.5 m

Grade estimates were interpolated into parent cells and all sub-cells were assigned the parent cell grades. Any un-estimated blocks were assigned a value of 0.005 g/t Au.

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, Kriging Efficiency %, which were later used in the determination of the resource classification.

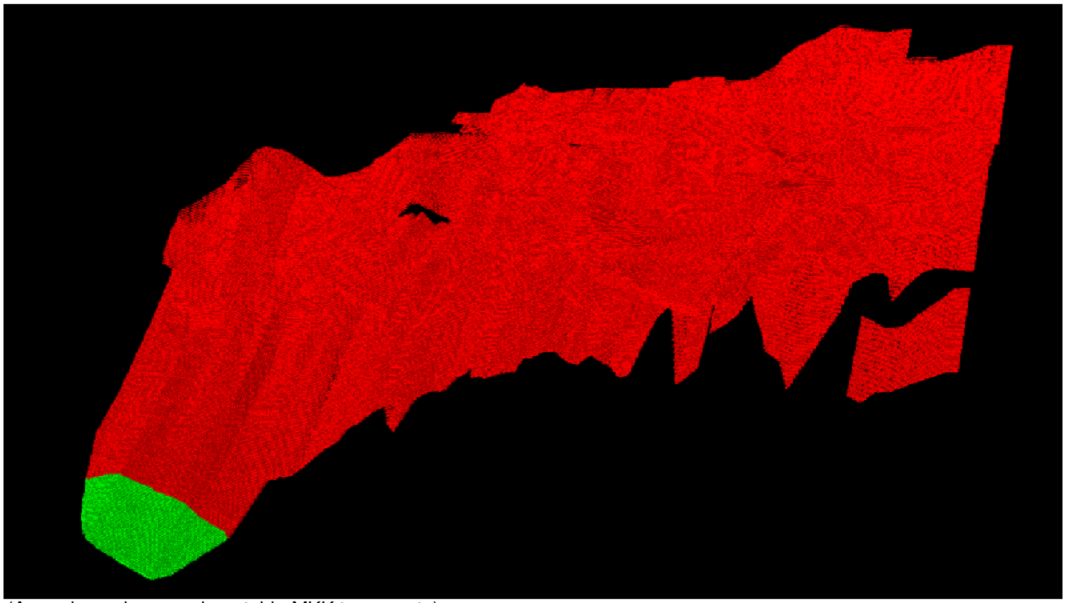
Depletion for Underground Workings

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.

Mining Lease Boundary

Within the mining lease tenements at Ollachea, there is a small lease area, not owned by MKK, as discussed in Section 4.2. Any part of the block model, which was not within the vertically projected property boundaries was flagged as MLEASE=0. The resource is reported only where MLEASE=1. Figure 14-2 shows the mineralized blocks that within the MKK mining concessions as red and those that are outside the MKK mining concessions coloured green.

Figure 14-2: Three Dimensional View of Mineralized Blocks with respect to Property Boundaries



(Area shown in green is outside MKK tenements)

14.10 Block Model Validation

14.10.1 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.

14.10.2 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 models' 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 models' 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.

14.11 Classification of Mineral Resources

The Mineral Resource estimates for the Ollachea Project (Minapampa and Minapampa East deposits) 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 14-10.

Table 14-7: 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 1m sampling method has been implemented, though there is a high amount of 2m samples from earlier campaigns. Within the new 14 drill holes, some areas within the mineralization have been 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 40m x 40m spaced drilling in central zone has provided adequate data for an inferred / Indicated resource. Infill to 20 x 20m will be required to increase the confidence of the current interpretation.	Moderate
Audits or Reviews	Audits have been routinely completed, last one by Smee (2009) on laboratory and QA/QC procedures. All issues identified have been rectified in a timely manner.	High
Database Integrity	Assay hard copy sheets were randomly 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 dominate 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 three pass estimation method for all blocks. High grade values were distance limited.	High
Cut off Grades	A threshold of 1g/t Au was used to define the high grade envelopes.	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 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.

14.12 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 this study.

14.13 Mineral Resource Statement

Mineral Resources for the Ollachea property above a 2.0 g/t Au cut off consist of 10.7 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 3.0 g/t Au. Mineral Resources were estimated by Doug Corley, MAIG, of Coffey Mining Perth, a Qualified Person under National Instrument 43-101, and have an effective date of 31 May, 2011 (Table 14-8).

The Mineral Resources replace those declared in the January 2011 Technical Report (Coffey, 2011) and previous estimates declared for the Property (Coffey, 2010). 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 14-8: Mineral Resources for the Ollachea Project

Mineral Resources above a 2.0 g/t Au Cut-off Grade	Tonnage (Mt)	Au Grade (g/t)	Contained Au (Moz)
Minapampa			
Indicated	9.3	4.0	1.2
Inferred	2.4	3.0	0.2
Minapampa East			
Indicated	1.4	3.9	0.2
Inferred	0.9	3.0	0.1
Total			
Indicated	10.7	4.0	1.4
Inferred	3.3	3.0	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, QP, of Coffey Mining and have an effective date of 31 May, 2011.

15.0 Mineral Reserve Estimates

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 cut-off grade for the Indicated Mineral Resource are shown in Figure 15-1 and Figure 15-2 and. Total ounces of gold are split into geologically-defined mineralised zones.

Points related to Figure 15-1**Error! Reference source not found.** are:

- The planned exploration incline is to be located around 2775 mRL. Approximately 86% of the in situ ounces are located above this RL.
- Zone 2 contains approximately 48% of the in situ ounces over approximately 200 vertical metres. This zone contains mineralised lodes that have the greatest width.
- Zone 5 contains approximately 24% of the in situ ounces however this is over approximately 450 vertical metres. This zone has mineralised lodes that are of significantly lesser width than the mineralised lodes in Zone 2.
- Zone 3 contains approximately 17% of the in situ ounces and this is over a similar vertical distance to that of Zone 2. Zone 3 mineralised lodes are of a similar width to those of Zone 5.
- The remaining 11% 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)

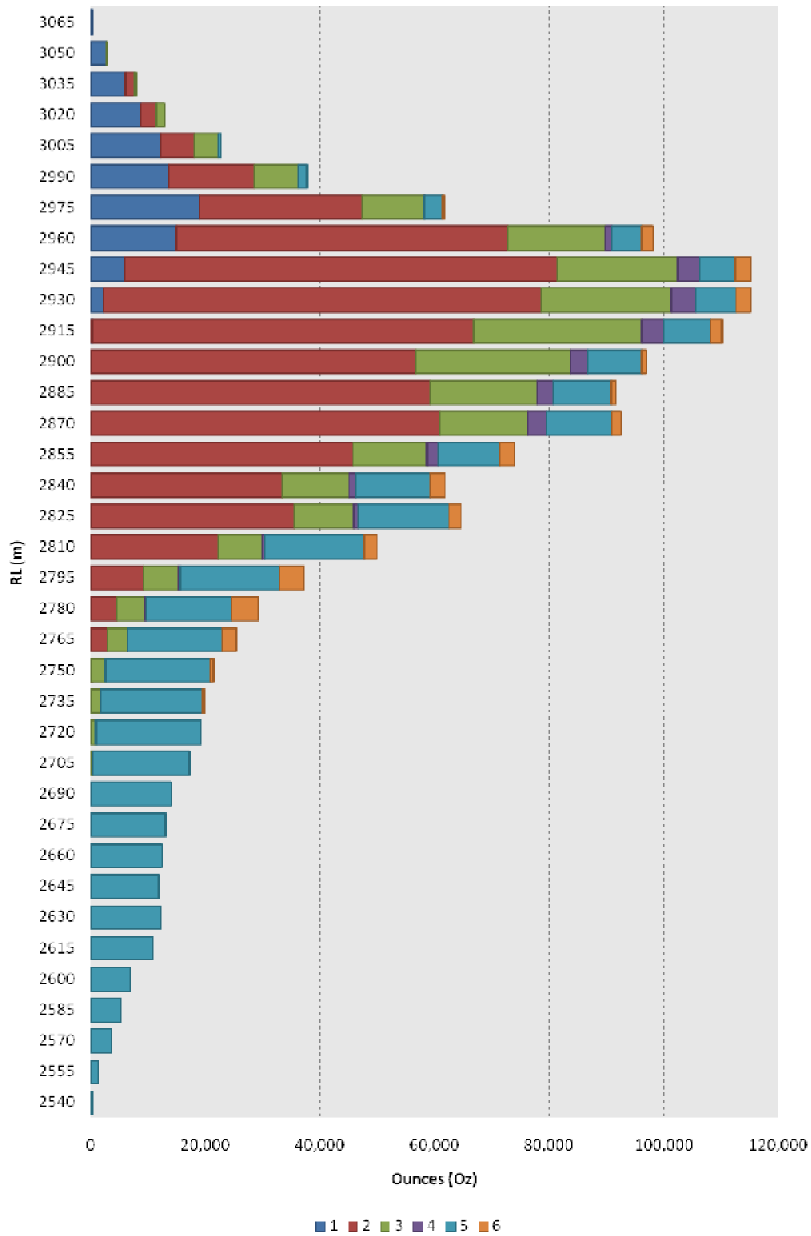
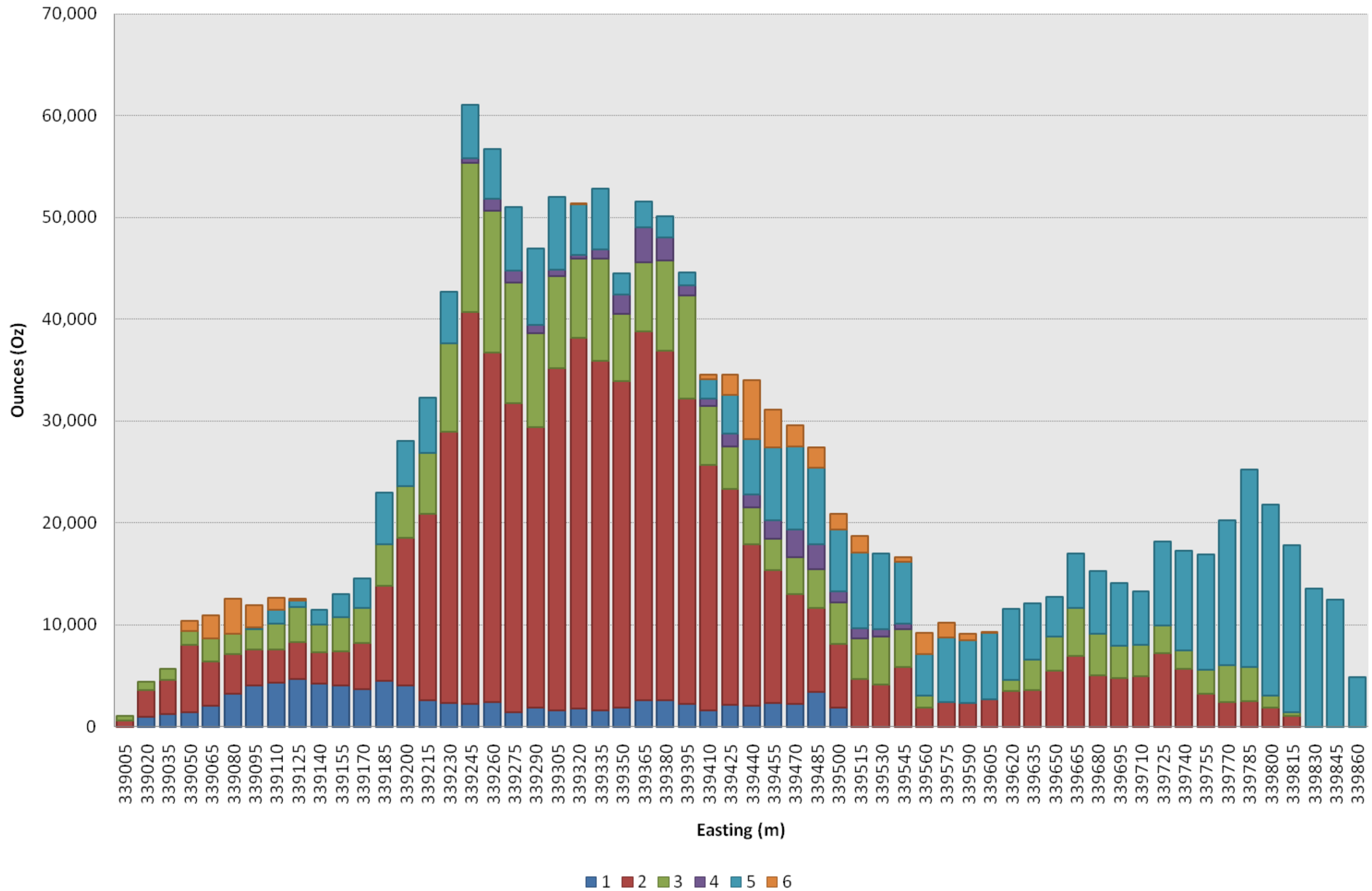


Figure 15-2: Ounces per 15 m Easting Step (Indicated Category Only)



Points related to Figure 15-2 are:

- Two domains have been flagged for the Mineral Resource, Minapampa and Minapampa East. The split occurs approximately at an Easting of 339680 m.
- Approximately 87% of the in situ ounces are located in Minapampa.

15.2 Cut-off Grade

A cut-off grade of 2.0 g/t Au was used for the Pre-feasibility Study and this was based on the work completed for the Preliminary Assessment (PA) and a base Project gold price of US$1,100/oz. Table 15-1 shows the Project in situ Au break even grade based on operating costs and recoveries estimated during the completion of the preliminary assessment for a range of gold prices.

Table 15-1: Break Even Grade Estimate

Parameter	**Gold Price (US$/oz)**					
Gold Price (US$/oz)	850	950	1,000	1,100*	1,200	1,400
Mill Recovery (%)	90	90	90	90	90	90
Mining Recovery (%)	85	85	85	85	85	85
Mining Cost (US$/t ore)	22.14	22.14	22.14	22.14	22.14	22.14
Mill Cost (US$/t ore)	19.63	19.63	19.63	19.63	19.63	19.63
G&A Cost (US$/t)	3.98	3.98	3.98	3.98	3.98	3.98
Realisation (US$/t)	0.64	0.64	0.64	0.64	0.64	0.64
Royalty (US$/t)	2.06	2.06	2.06	2.06	2.06	2.06
Total Cost (US$/t)	48.45	48.45	48.45	48.45	48.45	48.45
In Situ break even grade Au g/t	2.32	2.07	1.97	1.79	1.64	1.41

*Base case gold price.

15.3 Mining Limits

Determination of the PFS underground mining limits was completed using three primary software processes:

- Datamine Mineable Shape Optimiser (MSO): creation of stope shapes.
- Mine2-4D: mine design, sequence and geological resource model evaluation.
- Earthworks Production Scheduler (EPS): mine development and production schedule and final design element classification.

The second and third parts of the process were iterative.

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

Datamine Mineable Shape Optimiser Process

The initial mineable limits of the study were identified by using the Datamine process, MSO. Various parameters such as the cut-off grade, stope dimension, minimum mining width and dilution factors were inputted into MSO which then automatically generates three-dimensional rectangular stope shapes, based on the defined criteria.

A surface crown pillar of 20 m to the topography was also applied as a constraint during the MSO process. This limit was used to minimise the impact of subsidence and interaction with the artisanal mining near surface. Geotechnical and hydrogeological work will be required to optimise this limit criterion.

Stope shapes were defined by processing the geological resource model and flagging shapes which were equal to or greater than the selected Au cut-off grade of 2.0 g/t within a volume defined by minimum mining unit dimensions plus dilution. All Mineral Resources (Indicated and Inferred) were considered during this initial step.

Key criteria or data used in the MSO process were:

- geological resource wireframes that represent the location of the lode mineralization;
- 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 mining unit dimensions based on the mining method and its selectivity:
    - 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 information, lode geometry and the nature of the MSO process:

- The geotechnical review indicates poor ground conditions requiring support on small span openings. At 15 m high, the 45° dipping panel requires support and can be adequately supported from within the drilling and extraction drives. Larger stope dimensions would be more difficult to support adequately from the drives.
- The geological interpretation of the mineralised lodes suggests the location of the mineralization is variable over short distances in three dimensions. To be able to use the MSO process effectively and efficiently to create economic mineable shapes based on the selected criteria, smaller stope dimensions were required. This allowed ore loss and dilution from lode geometric changes to be minimised.

Mine2-4D Process

The process steps completed in Mine2-4D 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 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.

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.

15.4 Mineral Reserve

Table 15-2 shows the Mineral Reserve estimate, based on a cut-off grade of 2.0g/t Au. The Mineral Reserve is included within the declared Indicated Mineral Resource and is declared inclusive of approximately 1.4 Mt of dilution at an average grade of 0.4 g/t Au.

Table 15-2: Mineral Reserve Estimate (June 26, 2011)

Classification	Tonnes (Mt)	Au Grade (g/t)	Contained Gold (Moz)
Probable Mineral Reserves	9.5	3.6	1.1

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.

A summary of the main input parameters used in estimating the Mineral Reserve are shown in Table 15-3.

Table 15-3: Input Parameters used for the Mineral Reserve Estimate (June 26, 2011)

Description	Units	Value
Gold Price	US$/oz	1,100
Mine Design Au Cut-off Grade	g/t	2.0
Mining Method		SLOS
Minimum Mining Width (excluding dilution)	m	2.0
Annual Production Rate	Mt /a	1.1
Mining Operating Cost	US$/ t ore	18.5
Milling Operating Cost	US$/ t ore	24.3
G&A Operating Cost	US$/ t ore	3.9
Mining Dilution (Stopes - Transverse; Longitudinal)	%	9; 15
Mining Recovery (within mine design shape)	%	100
Mill Recovery	%	91.3
Project Capital Cost	US$M	169.5
Sustaining Capital Cost	US$M	47.0
Closure Cost	US$M	3.1
Royalty	US$M	28.8
Workers Profit Share	%	8.0
Tax Rate	%	30.0

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 15-4.

Table 15-4: Mineral Reserve Estimate Sources of Supporting Information (June 26, 2011)

Modifying Factor	Source
Mineral Resources	Coffey Mining
Geotechnical Engineering	Coffey Mining
Mine Design and Scheduling	Coffey Mining
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)
Social	MKK

The underground mine design includes a 20m crown pillar to maintain the stability of the upper valley floor and retain watercourse integrity. This area was excluded from the process to estimate a project Mineral Reserve.

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 20m crown pillar.

No formal Project risk assessment has been completed therefore quantitative impact has not been determined. 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.

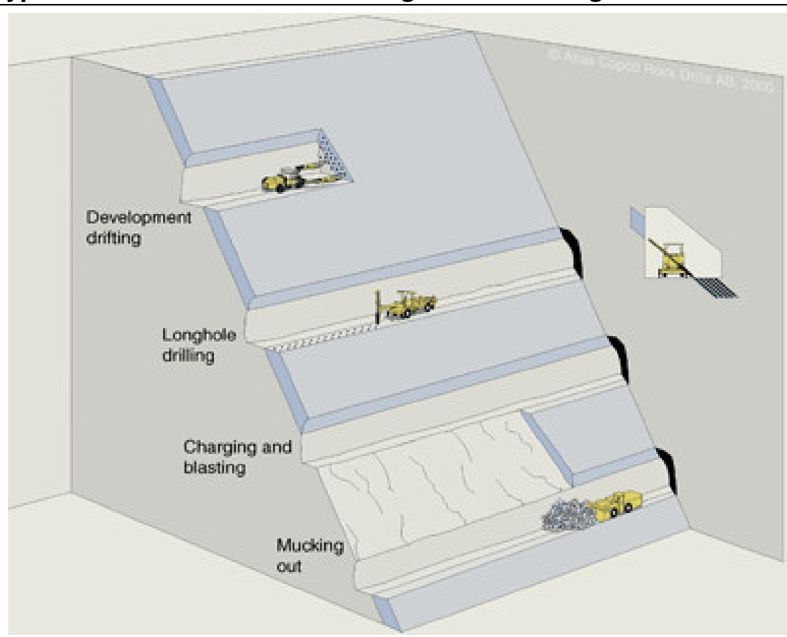
16.0 Mining Methods

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.

The mining method selected for the PFS was sublevel open stoping (SLOS) with fill, also referred to as bench stoping with fill when the mining occurs along the strike direction.

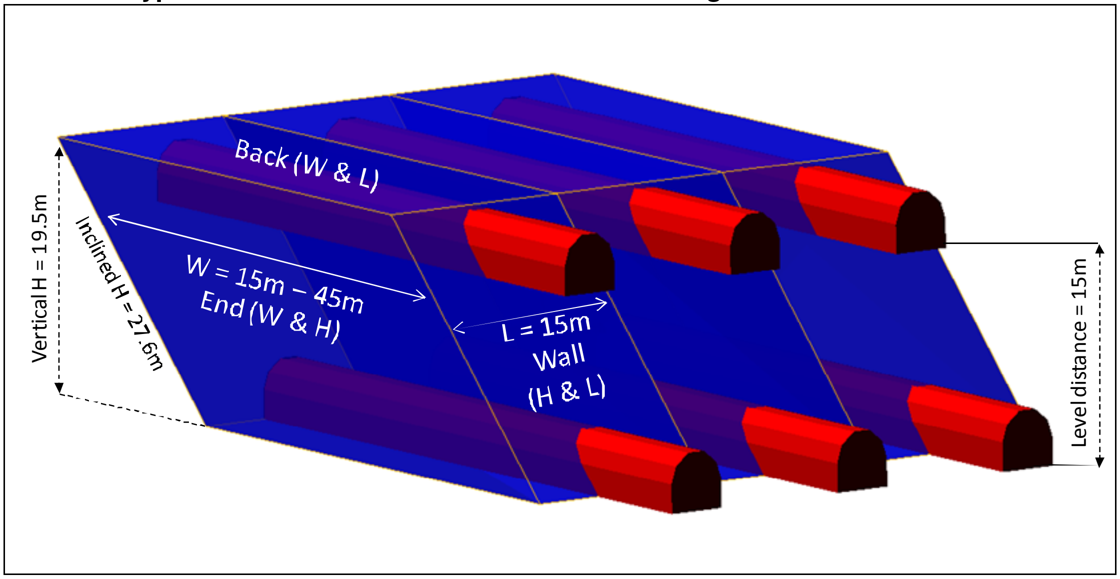
Stopes will be accessed either longitudinally (along strike) or transversally (perpendicular to strike) dependent on lode thickness. In general, when lode thickness is greater than 15 m, transverse stopes will be favoured. Figure 16-1 and Figure 16-2 present both longitudinal and transverse typical views of the mining method.

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



(Sourced from Atlas Copco)

Figure 16-2: Typical View of the Selected Transverse Mining Method



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

16.1 Sublevel Open Stoping with Fill

Key selection characteristics of the SLOS mining method are:

- Mining is fully mechanised.
- Provides an acceptable mix of productivity, low cost and selectivity.
- High ore recovery and controlled dilution is possible.
- Can be used longitudinally and transversally.

16.2 Mine Design Process

The mine design process consisted of the following steps:

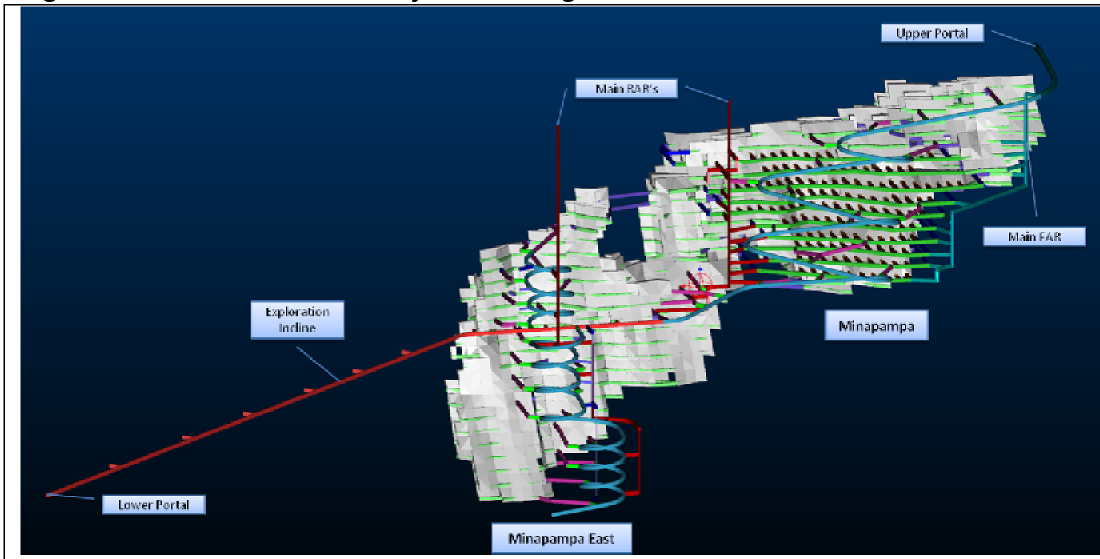
- Review of Preliminary Assessment (PA) outcomes and supporting data.
- Review of the PFS block model and supporting data.
- Review of the PFS geotechnical outcomes and supporting data.
- Determine mineable stope shapes using the PFS cut-off grade and mine design criteria.
- Design mine accesses and sequences based on the previous step, and evaluate development and stope shapes against the PFS block model.

- Apply resource constraints to allow a practical mine development and stoping schedule to be produced.

16.3 Final Mine Layout

The final PFS mine layout is shown in Figure 16-3.

Figure 16-3: Isometric Mine Layout Looking South



The main access to the mineralisation will be via a 1.3 km-long exploration access drive or incline which has its portal in a valley on the north-eastern side of Cerro Joropiña and the Occo Cachi River valley. The drive will be excavated in the second half of 2011 and will initially be used for exploration drilling and subsequently converted to an access and haulage drive for the planned mine. This portal will be the main mine portal.

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 2785 mRL. The incline drive will be developed to half way up the mineralisation, and the decline drive down to approximately 2550 mRL to provide access to the planned sublevels. A second portal (3060 mRL) and decline will be developed simultaneously from the top of the mineralisation to provide a second means of egress, access to the upper mineralisation, and early establishment of the primary ventilation system.

The selected mining method and sequence will be bottom up with production starting from 2790 mRL and 2835 mRL as soon as the development of the decline from surface is completed.

The geometry of the lodes, the average 45° dip, width variability in the strike direction and the geotechnical aspects limit stope size. Stope sublevel spacing will be 15 m vertically floor to floor. Stopes will be subdivided into 15 m panels along strike. Transverse stopes will be on average 15 m high by 15 m wide by 15 m or greater in length. Longitudinal stopes on average will be 15 m high by 15 m or less in width by 30 m in length. Two 15 m panel stopes will be mined before backfilling. To control the stability of the longitudinal stopes and minimise dilution, the length of open void can be altered based on local ground conditions.

The majority of production drilling will be medium diameter (89 mm) down-holes with some requirement for up-holes 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 and maintain stability, stopes will require cable bolts to be installed. Cable bolt holes will be drilled using the same type of drill as used for production drilling but at a smaller diameter. Drill rigs will therefore be standardised to optimise fleet maintenance requirements.

All ground support installation will be completed manually. This is based on the current relatively low labour cost.

Stopes will be backfilled primarily for stability purposes but also to reduce the surface area required for waste and tailings disposal. Pastefill will be the principal backfill used with waste rock used in secondary transverse stopes, and as a capping for tramming purposes on all paste filled stopes.

The primary ventilation circuit comprises two centrally-located return air raises, one intake air raise and two decline/incline intakes.

16.4 Hydrogeology

No hydrogeological report or measurement information of the area was available during the execution of the mine design component of the PFS. Water inflow to the mine was expected. The proximity of the mineralised zone to the stream and the fact that potable water for the Ollachea village is being sourced from underground near the mineralised area indicated high potential for ground water inflow to the mining area. It was also noted that some diamond drill holes encountered artesian water, albeit at low pressure. Water is also seen flowing out from several of the artisanal miners' openings. No measurements of flow were available from either the drill holes or the artisanal drives at the time of execution of the PFS or the filing date of this Report.

For design purposes, it was assumed that water would need to be drained from the upper levels and pumped from the lower levels to 2775 mRL and then be pumped or gravity drained to the plant through the planned exploration incline. At the mine portal, drainage water would be collected and transferred to the water treatment facility at the plant site.

Interpretation of hydrogeological data from the area support these design criteria.

16.5 Geotechnical

16.5.1 Rock Mass Conditions

An assessment of rock mass conditions was based on core logging carried out by MKK from 117 exploration drill holes. Of these, 18 exploration drill holes were oriented and formed a basis to determine the defect orientations and characteristics.

Rock Mass Classification (*Q’*)

Coffey Mining has utilised the Modified Rock Quality Index (Q’) to classify the *in situ* rock mass at Ollachea. Based on the information available, Q’ is calculated to be 3.5, which is considered a poor ground rating. This rating is based on a RQD value of 21, which is the 25th percentile of the RQD distribution. The mean and median RQD values are 52 and 56 respectively. This is therefore a more conservative value and reflects the confidence level of the geotechnical data. This rating has impacted on the costing of ground support for both the development and production areas and as such, provides a practical basis for the study.

Major Structures

Oriented core logs indicated that the foliation is the major structure followed by the sub-vertical faults striking parallel to the orebody.

Intact Rock Properties

Coffey Mining was supplied with 14 UCS test results by MKK. UCS testing was carried out by Universidad Nacional de Ingenira’s Laboratorio de Mechánica de Rocas. The test results yielded in UCS values ranging 27 MPa to 57 MPa with an average UCS value of 35 MPa.

In-situ Stress Field

No stress measurement was undertaken at Ollachea. For the purpose of the Pre-feasibility Study, a stress ratio of one has been considered.

16.5.2 Stable Span Analysis

Stable Span Methodology

Coffey Mining used the stability graph method, after Potvin (1988) and Nickson (1992), to assess the maximum stable spans for the stoping geometry at Ollachea. This is undertaken by calculating the modified stability number (N') for the respective areas within the stope, i.e. the backs, ends, and walls, and by correlating it to empirical stability curves, which are based on an extensive experience based data set of unstable and stable underground mines.

Maximum Unsupported and Supported Hydraulic Radius for Open Stoping

Based on the calculated hydraulic radii of the different surfaces, the dimensions for supported stopes are given in Table 16-1

Table 16-1: Supported Stope Dimensions for Ollachea Deposit

Stope Face	HR	Stope Dimension (m)									
Near Surface											
Back	7.80	Width (W)	5.00	10.00	15.00	20.00	25.00	30.00	35.00	40.00	45.00
		Length (L)	∞	∞	∞	70.91	41.49	32.5	28.14	25.57	23.88
Wall	9.10	Height (H)	27.60								
		Length(L)	53.44								
Ends	13.20	Maximum orebody width to satisfy the Hydraulic Radius is infinite									
At Depth											
Back	7.40	Width (W)	5.00	10.00	15.00	20.00	25.00	30.00	35.00	40.00	45.00
		Length (L)	∞	∞	1110.0	56.92	36.27	29.21	25.64	23.49	22.05
Wall	8.90	Height (H)	27.60								
		Length(L)	50.13								
Ends	12.90	Maximum orebody width to satisfy the Hydraulic Radius is infinite									

Conclusions on Stope Dimensions

Based on the stable span analysis, the following conclusions can be made:

- For the ends (east and west walls), the inclined height (H) is set to be 27.6 m. Based on this height, the maximum unsupported width (W) of the stope ends is calculated to be 157.99 m. As the maximum stope width will be limited to the orebody thickness (45m), the stope ends will not be the limiting factor for the stope dimensions. It is considered that the stability of the ends will not be problematic for both mining directions (longitudinal and transverse).

- For the stope walls (north and south walls), the inclined height (H) is set to be 27.6 m and the unsupported hydraulic radius of 3.90 m for the walls results in maximum strike length (L) of 10.87 m. For longitudinal mining, maximum strike length will be limited to 10.87 m (near surface) and 10.11 m (at depth), if left unsupported. For transverse mining, the maximum unsupported strike length is lower than the mine design length of 15 m. Cable bolt support will be required for the hanging wall. The supported stope dimension calculations showed that the strike length (L) can be just over 50 m, which satisfies both mining directions (longitudinal and transverse).
- For the stope backs, stope back is controlled by the orebody width and strike length. Unsupported strike length (L) of 10.87 m (near surface) and 10.11 m (at depth) will be a limiting factor for the unsupported back design. Similar to stope walls, installation of support will also be required for the backs. Supported backs will satisfy the required stable hydraulic radii of the backs for both longitudinal and transverse mining directions.

16.5.3 Mine Development

Support Requirements Recommendations

The support recommendation for access tunnels is systematic bolts 2.1 m long, spaced 1.5m in the crown and walls. Where required, fibre reinforced shotcrete (FRS) should be installed with a thickness of 50 mm on the crown and on the sides.

For costing purposes, 50% of the ramp access with a 5 m by 5 m cross section and 100% of the ventilation decline with a 6.5 m by 6.5 m cross section will have systematic bolts and FRS as they are of a more permanent nature. All other drives are assumed to have systematic bolts and mesh with 25% requiring FRS.

Large Excavation Support Requirements

For excavations larger than the regular drives, such as intersections, in addition to the systematic bolts and FRS support, 6 m-long cable bolts fully grouted spaced at 3 m are included for cost estimate purposes.

16.6 Mine Development Design

The location of the main mine accesses for the purposes of the PFS is in the orebody hanging wall. This was selected primarily based on the location of the planned exploration incline and no discernible difference in the rockmass between hanging wall and footwall. Orebody access development stand-off distances have been assumed for the PFS as no geotechnical analysis work has been completed. This will be undertaken during the completion of a planned feasibility study.

Key design parameters for lateral development are shown in Table 16-2. Other parameters are:

- maximum grade for access development is 1 in 7.
- minimum grade target is 1 in 50.
- minimum turning radius is 30m.

Subsequent to the completion of the mine design, ventilation modelling has identified the requirement for a development connection between the two primary surface return air raises (RARs) to enable load sharing. This is not included in the PFS mine design.

Table 16-2: Lateral Development Design Parameters

Description	Design Criteria	Units	Value
	Height	m	4.5
In ore drive	Width	m	4.5
Access transverse draw point	Profile type (Radius)	m	Square
	Profile area	m²	20.25
Incline	Height	m	4.5
Incline Stockpile	Width	m	4.5
Access Footwall Drive	Profile type (Radius)	m	Arch (2.0)
Access Main Cross-Cut	Profile area	m²	23.18
Access In Ore Cross-Cut			
Fresh Air Way Cross-Cut			
Access Incline Cross-Cut			
Access To Ore Cross-Cut			
Return Air Way Cross-Cut			
Access Main Cross-Cut			
Access In Ore Cross-Cut			
Access Inc. Cross-Cut			
Access To Ore Cross-Cut			
Escapeway Main Access			
	Height	m	6.5
Incline Fresh Air	Width	m	6.5
	Profile type (Radius)	m	Arch (2.0)
	Profile area	m²	40.52

Key design parameters for vertical development are shown in Table 16-3. The profile for vertical development is either square or circular and depends on the excavation technique used. The longer primary ventilation raises are excavated by raise boring machines whereas the secondary shorter raises are blasted drop raises.

Table 16-3: Vertical Development Design Parameters

Description	Design Criteria	Units	Value
FAR_Primary RAR_Primary	Height	m	4.5
	Width	m	4.5
	Profile type (Radius)	m	Circular (4.5)
	Profile area	m²	15.9
FAR_Secondary RAR_Secondary	Height	m	4.0
	Width	m	4.0
	Profile type (Radius)	m	Square
	Profile area	m²	16.0
Escape_raise	Height	m	1.5
	Width	m	1.5
	Profile type (Radius)	m	Circle (1.5)
	Profile area	m²	1.77

Infrastructure development includes lunch room/refuge, pump stations for water and paste, stockrooms, explosives and detonator magazines, temporary workshops and loading bays. These are not included in the PFS design.

An allowance of 3% of the total waste development has been added to account for all PFS development design exclusions.

16.7 Mine Production Design

The key elements of the stope design are shown for transversal stopes in Figure 16-2 and for longitudinal stopes in Figure 16-1. 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 are drilled using down-holes except for stopes located at the top of a lode. These will use up holes to eliminate the requirement for specific drill drive development. Stope design sizes are shown in Table 16-4.

Table 16-4: Stope Design Size

Area	Criteria
Stope Size	Longitudinal stope – 15 m H x 30 m L (strike length) x 2 m to 15m W Transverse stope – 15 m H x 15 m W (strike length) x 15 m to 45m L

For both the longitudinal and the transverse stopes, the slot raise will be drilled and blasted using a drop raise technique. This requires holes to be drilled from the top drill drive in a similar pattern to a development drive drill pattern.

For stopes that will not have a top drill drive, it has been assumed the slot will be manually excavated using airleg techniques. This is expected to occur only in longitudinal stopes.

The ground support for both types of stope will be mainly oriented at supporting the backs (roofs) and the hanging walls. The support will be fully grouted cable bolts as described below:

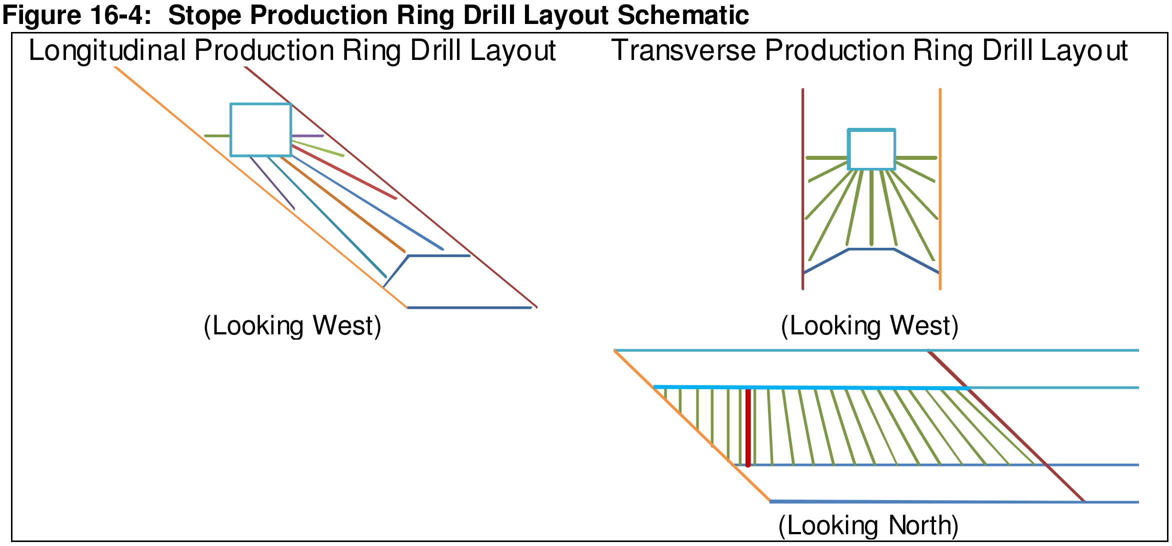
- Holes will be pre-drilled as up holes using the same drilling equipment that is used for production drilling but using a smaller diameter.
- Cables will be installed manually from scissor-lift equipment.

Stope drill and blast parameters assumed for the PFS are:

- Drill hole size is 89 mm.
- Drill factor for longitudinal stopes is approximately 6.9 tonnes per drill metre, including slot raise metres.
- Drill factor for transverse stopes is approximately 9.3 tonnes per drill metre, including slot raise metres.
- ANFO type explosives have been assumed and these would be loaded using a specific charging vehicle.

A schematic of a typical drill pattern for a longitudinal stope and a transverse stope is shown in Figure 16-4.

Figure 16-4: Stope Production Ring Drill Layout Schematic



16.8 Materials Handling

The strategy adopted for the PFS is for all ore and waste material to be loaded using 10 t-capacity load-haul-dumps (LHDs) and transported to dumping areas located outside the two mine portals or internally as waste rockfill by dedicated 28t capacity underground mining 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.

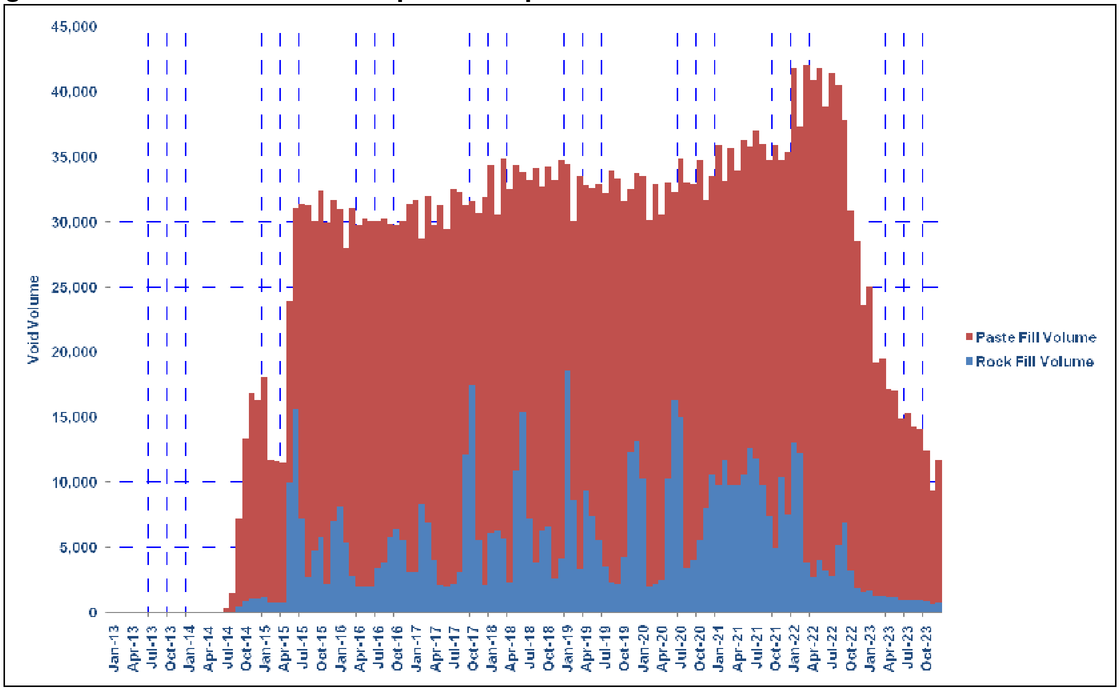
16.9 Backfill

The SLOS mining method and extraction sequence adopted for the Project is reliant on the use of backfill. Two primary types of backfill have been selected for the Project, rockfill and cemented pastefill. Total tailings will be used to produce the pastefill.

Pastefill is to be used as the primary fill type with waste rock being used in transverse secondary stopes and as a floor cap to paste filled stopes.

Figure 16-5 shows the monthly underground backfill volume requirement profile for the Project. This data should be viewed in conjunction with the data shown in Figure 16-9, Project waste balance.

Figure 16-5: Backfill Volume Requirement per Month



The overall backfill volume requirement split between pastefill and rockfill is 81% and 19% respectively.

The backfill strength demand is controlled by two significant mining factors: the stope geometry and the mining sequence. At Ollachea, where the mining sequence is predominately bottom up and backfill will generally not be undercut, the stability of the vertical exposure and the bearing capacity for re-entry and working on top of the pastefill is considered most relevant. Minapampa East requires some stopes to be filled that will be undercut. A review of the geometry of the proposed undercut stopes revealed that they are relatively small and that commonly-used mining and backfill techniques can be implemented to successfully undercut the stopes.

A vertical wall stability assessment was carried out using the methods outlined in Mitchell, Olsen and Smith (1982) that are considered suitable for a single exposure.

The assessment indicates that an unconfined compressive strength of between 300 kPa and 500 kPa will be adequate to ensure stability in a 30 m-high vertical stope for the range of exposure width considered for this material.

Time available for the pastefill to cure before being exposed was assessed. The analysis indicated that the stope cycle time is not critical with enough time for the binder to gain strength before the pastefill is required to perform.

The backfill assessment undertaken during the PFS was completed using limited data. No information was available for tailings particle size distribution, moisture content and mineralogy, rheological properties and strength testing of the pastefill. Because of this, pastefill characteristics were assumed. The assumptions used are considered to be on the conservative side of the range of possible values. The assumptions were:

- Ollachea plant tailings have a particle size distribution that is suitable for use as pastefill.
- The mineralogy is not unusual and there are no minerals present that act to retard or enhance strength development.
- Ollachea rheology is typical of pastefill in the normal range of operating parameters.

The pastefill mix design presented in Table 16-5 is based on the previous assumptions and considers a likely range of mix designs applicable for the Ollachea pastefill.

Table 16-5: Pastefill Mix Design

Water Content %	Cement Content (%)	Dry Density (t/m^3)	Dry Density (t/m^3)	Bulk Density (t/m^3)
25%-30%	3%-5%	1.4	1.4	1.8

Three pastefill reticulation concepts were considered with the selected option requiring the paste plant to be located close to the process plant area and paste to be pumped to the proposed mine.

A reticulation assessment was undertaken to determine Project piping and pumping requirements. This assessment was based on previous project and operating mine experience because no rheological testing had been undertaken at the time of completion of the mining component of the PFS.

The assessment showed that the most suitable standard pipe dimension is a 150 mm nominal bore pipe and that three suitably sized positive displacement pumps will be required to reticulate the Ollachea pastefill. The first pump is to be located at the paste plant, the second at the lower portal, and the third pump placed at the base of the proposed vertical section of the reticulation close to where the exploration incline ends.

The relatively small stope sizes and relatively short cycle times mean that a simple and rapid pastefill barricade construction method would be suitable for the mining method. A waste rock bund/barricade design with shotcrete support would suffice.

Subsequent to the completion of the mining component of the PFS, AMEC reported that preliminary filtration and tailings test work confirm that the Ollachea tailings will be suitable for the production of paste backfill.

AMEC also reported that the tailings characterization test work has been conducted, and a draft test work report prepared by Outotec was delivered to AMEC in June 2011. Rheology and strength testing of pastefill samples was underway at the Report effective date.

16.10 Ventilation

16.10.1 Primary Ventilation

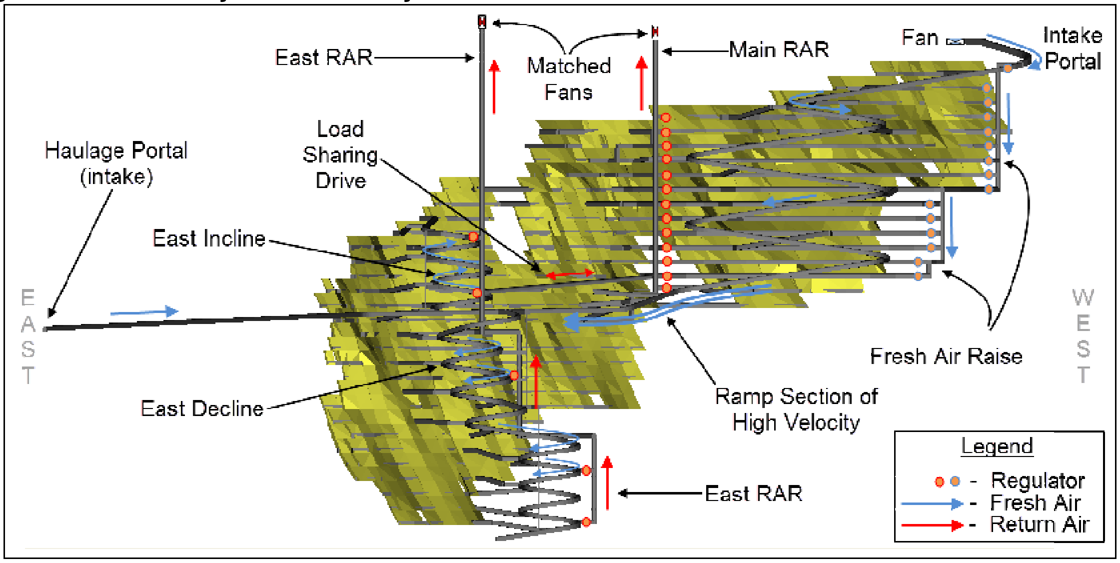
Ventilation milestone analysis was used to determine the primary ventilation requirements for the Project. Maximum ventilation demand for each milestone was estimated by analysing the mine development and production schedule to determine the number of active stopes and development headings in each month.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

Table 16-6 briefly describes each milestone that was identified and Figure 16-6 shows an isometric view of the Project primary ventilation system.

Table 16-6: Primary Ventilation Milestones

Stage	Description of Milestone
Stage 1 Start to 18 months	Breakthrough of the Main RAR to surface and installation of the primary exhaust fan. Q_{TOTAL}=45m^3/s
Stage 2 18 months to 29 months	Breakthrough of the Intake Ramp to the Haulage Ramp. Q_{TOTAL}=225m^3/s
Stage 3 29 months to 30 months	Breakthrough of the Main district FAR and installation of the primary intake fan. Q_{TOTAL}=260m^3/s
Stage 4 30 months to end of mine	Maximum primary ventilation airflow. Q_{TOTAL}=420m^3/s

Figure 16-6: Primary Ventilation System



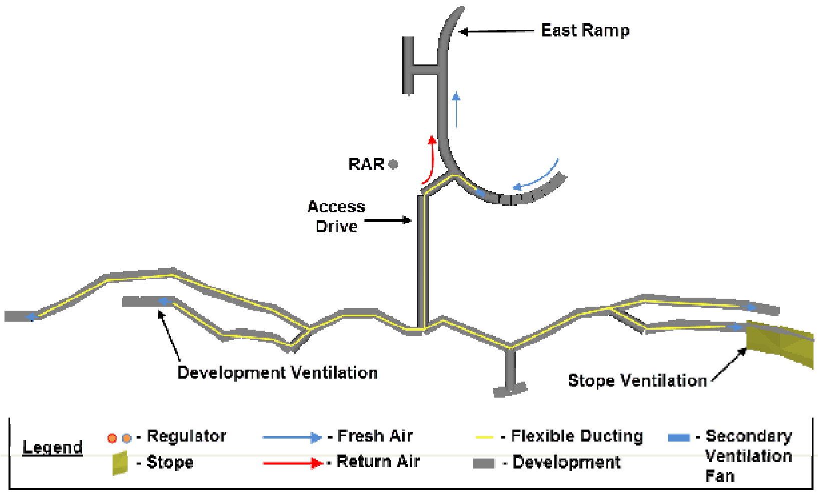
16.10.2 Secondary Ventilation

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

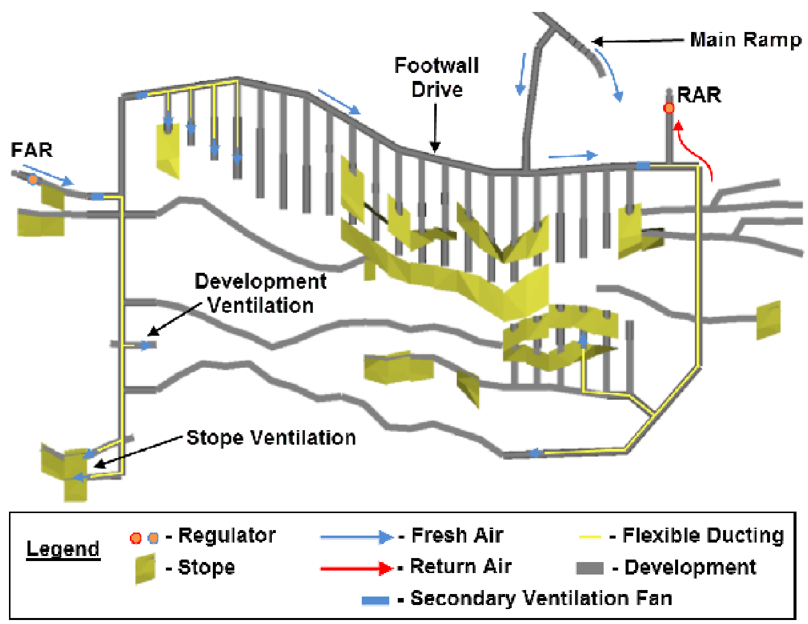
- A long-range configuration for development designed to establish or extend the primary ventilation circuit.
- The levels of the East district (Minapampa East) where the secondary fan is located in the fresh air decline and ducting is run into the level with branches to each heading or stoping area.
- The layout found in the Main district (Minapampa), whereby the fan is located in the fresh air footwall drive and ducting is run into drives and stoping areas branching off this development.

The second and third layouts are shown schematically in Figure 16-7.

Figure 16-7: Secondary Ventilation System – Minapampa East and Minapampa



Typical Mining Level - Minapampa East (above) and Minapampa (below)



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 as a primary ventilation intake.

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 PFS was as follows:

Expedite the development of the primary mine accesses and ventilation system to minimise the production ramp up period and provide a second means of egress.

Priority given to starting production on the upper levels of Minapampa East, which has been estimated to contain higher grade material, and the lower levels of Minapampa to establish the bottom up mining method and maximise ore extraction from the area.

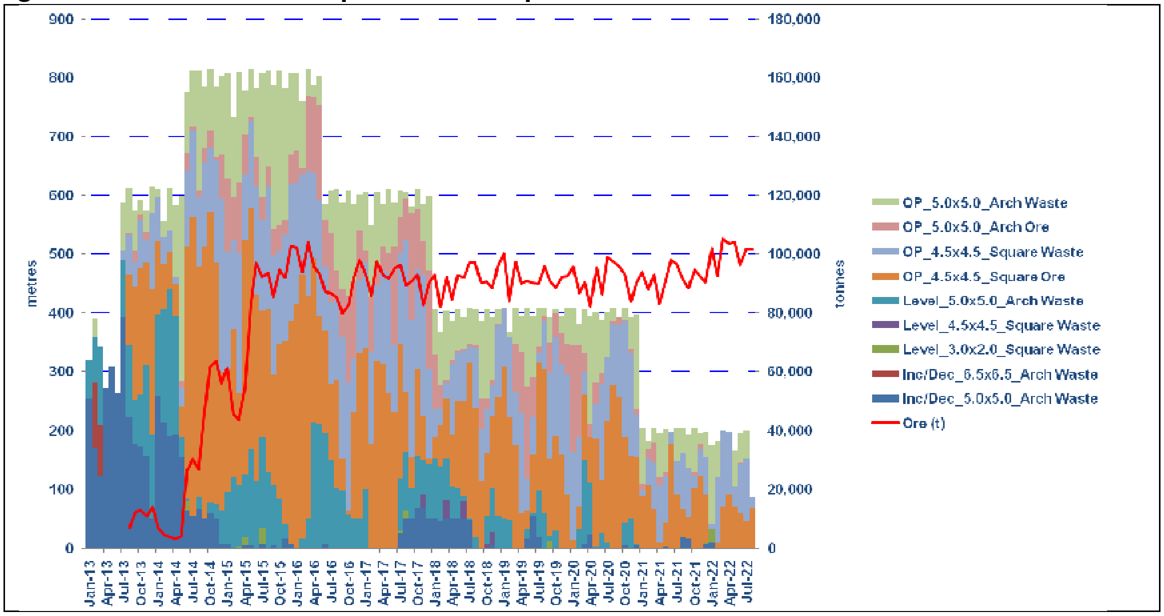
Table 16-7 shows the jumbo development rates that were used to schedule lateral development activities.

Table 16-7: Jumbo Development Advance Rates

Jumbo Development	Scheduled Work Rate (m/month)
Single heading maximum per jumbo	120
Multiple heading maximum per jumbo	200

Figure 16-8 shows the average lateral development metres per month split by development type. The maximum number of metres per month is 800, which is equivalent to employing four jumbo crews, and this occurs for a period of two years.

Figure 16-8: Lateral Development Metres per Month



Development is scheduled to be completed by mid-2022 with production scheduled to extend until 2023.

Table 16-8 shows the development rates that were used to schedule vertical development activities. Stope slot raises are not explicitly defined. The timing to develop these was incorporated in an all-inclusive stope turnaround time.

Table 16-8: Vertical Development Advance Rates

Development Type	Dimensions (m)	Scheduled Work Rate (m/day)
Raisebore	4.5 dia.	2.5
Raisebore	1.5 dia.	2.5
Longhole drop raise	4m by 4m	2.5

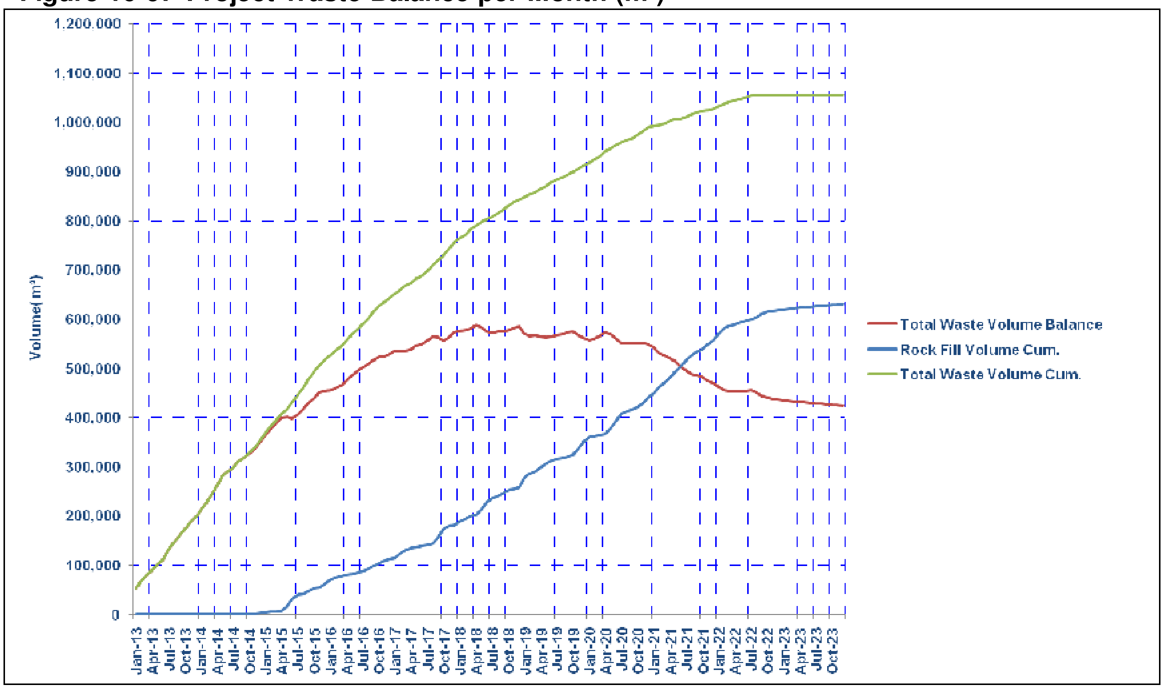
The majority of vertical development is associated with the primary ventilation system and this is completed pre full production.

Waste material generated from the development of the exploration incline and pre-production waste development will be hauled to the lower portal and used to build a platform that will contain the coarse ore bins, primary crusher and also be used for ore stockpiling for the life of the mine.

Waste rock from the development of the upper portal and decline will be deposited on an upper waste dump that will be rehabilitated in a timely manner. All other waste rock will be transported out of the lower portal and stockpiled until it is required to fill stope voids that are not filled with paste and to cap paste filled stopes or rehabilitated in a timely manner.

Figure 16-9 shows the Project waste balance per month in cubic metres for waste generated versus rock fill required (stope void to be filled). This includes approximately 33,000 m³ of in situ waste material generated from the completion of the Exploration incline and stockpiles (cuddies). All volumes of waste material have been expanded by applying a 30% swell factor.

Figure 16-9: Project Waste Balance per Month (m³)



The maximum volume of waste that has to be located on a surface waste dump during the life of the project is 231,000 m³. At the end of the mine life, approximately 70,000 m³ has to be contained in surface waste dumps.

16.12 Mine Production

The production strategy employed for the PFS was as follows:

The 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 Minapampa geological domain, the bottom up sequence is valid for the entire sequence;
- for the Minapampa East geological domain, which is significantly smaller than the main Minapampa geological domain, the bottom up method has been modified to improve productivity and allow consistent parallel production with Minapampa. Mining levels have been grouped together (three or four) with grouped levels mined with a bottom up sequence. This essentially allows multiple areas to be mined simultaneously if access development has been completed and ventilation is available. It does however create artificial crown 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. The width of the lodes to be extracted in Minapampa East should not present any significant issues with regard to providing stable artificial crown pillars. The average dip (45°) of the lodes will also assist in artificial crown pillar stability.

For the PFS, 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 the three types of stope. These were longitudinal paste, transverse paste and transverse rock. Table 16-9 shows the assumptions that were used to derive the three stope duration algorithms.

Table 16-9: Stope Production Rates

Stoping Activity	**Longitudinal**		**Transverse**			
	Paste		**Paste**		**Rock**	
	Fixed (day)	**Variable (t/day)**	**Fixed (day)**	**Variable (t/day)**	**Fixed (day)**	**Variable (t/day)**
Cable Bolt and Grout	4.0		3.0		3.0	
Production Drilling		1,439		1,962		1,962
Blasting	5.0		5.0		5.0	
Loading		1,500		2,300		2,300
Barricades	4.0		4.0		2.0	
Fill		5,356		5,356		735
Curing	14.0		21.0		0.0	
Floor Base	1.0		1.0		0.0	
Totals	**28**	**0.00155d/t**	**34**	**0.00113d/t**	**10**	**0.00230d/t**
Scheduling Algorithm	0.0155*t+28		0.00113*t+34		0.00230*t+10	

16.13 Mine Physicals Summary

PFS underground mining physicals are summarised in Table 16-10.

Table 16-10: Mining Physicals Summary

Physical		Units	LOM	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Lateral Dev.	Capital dev.	m	**11,372**	3,718	2,312	1,288	1,194	934	1,022	398	427	47	32	
	Operating dev.	m	**45,564**	1,732	6,234	8,246	6,963	6,196	3,743	4,399	4,353	2,331	1,367	
Vertical Dev.	Capital dev.	M	**1,364**	609	332	177			89	128	30			
	Operating dev.	M	**0**											
Production	Total Mined	Mt	**11.6**	0.4	0.7	1.3	1.4	1.3	1.3	1.2	1.3	1.2	1.2	0.5
	Waste	Mt	**2.1**	0.3	0.3	0.3	0.3	0.2	0.2	0.1	0.2	0.1	0.1	
	Ore	Mt	**9.5**	0.1	0.3	0.9	1.1	1.1	1.1	1.1	1.1	1.1	1.1	0.5
	Contained Gold	koz	**1,112**	6	39	102	128	132	132	146	135	132	116	44
	Gold Grade	g/t	**3.6**	3.2	3.7	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.0
	Cable drill	kdm	**1,228**	60	189	214	170	150	104	116	117	68	41	
	Production drill	kdm	**1,095**		19	96	121	124	133	128	124	136	149	66
	Haulage	Mtkm	**26.5**	0.7	1.1	2.3	2.7	2.9	2.9	2.9	3.2	3.2	3.3	1.2
	Backfill Void (Paste)	Mm^3	**2,705**		52	236	311	304	325	303	303	308	385	177
	Backfill Void (Rock)	Mm^3	**630**		4	59	51	69	77	91	90	116	61	13

Totals may not sum due to rounding

16.14 Mine Equipment

The Project will require a standard small scale, underground production fleet of jumbos, LHDs, trucks and drills. The primary and secondary equipment used as a basis to design the underground mine is shown in Table 16-11.

Table 16-11: Primary and Secondary Underground Equipment

Generic Description	Type or Size
Development Jumbo	Twin boom electro-hydraulic
Underground Loaders	10t for development and production
Underground Trucks	28t 4WD articulated
Production Drill Rig	Top hammer
Scissor Lift	4wd UG specification
Charge-up Vehicle	4wd dedicated UG charge up vehicle

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 with minor servicing to be completed between shifts in basic underground service bays.

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

It has been assumed that MKK-owned equipment will be replaced as specified, based on industry standards and original equipment manufacturers (OEM) recommendations.

Table 16-12 shows the annualised LOM mobile equipment schedule for primary and secondary mobile equipment.

Table 16-12: LOM Mobile Equipment Schedule

Equipment Description				2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023
Primary	Development Jumbo			3	4	4	3	3	2	2	2	1	1	
	Underground Loaders		Dev.	2	3	3	3	3	2	2	2	2	1	
			Prod.		1	3	3	3	3	3	4	4	4	4
			Total Loaders	**2**	**4**	**6**	**6**	**6**	**5**	**5**	**6**	**6**	**5**	**4**
	Production Drill Rig			2	3	4	4	4	3	3	3	3	3	1
	Underground Trucks		Dev.	2	2	2	2	2	2	2	2	1	1	
			Prod.		1	2	3	4	4	4	5	5	6	3
			Total Trucks	**2**	**3**	**4**	**5**	**6**	**6**	**6**	**7**	**6**	**7**	**3**
Secondary	Charge Up		Dev.	2	2	2	2	2	2	2	2	2	1	
			Prod.		1	1	1	1	1	1	1	1	1	1
			Total Charge Up	**2**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**3**	**2**	**1**
	Scissor Lift	Ground Support	Dev.	2	2	2	2	2	2	2	2	1	1	
			Prod.	1	1	1	1	1	1	1	1	1	1	
		Services		1	1	1	1	1	1	1	1	1	1	
			Total Scissor Lift	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**4**	**3**	**3**	

16.15 Organisation and Structure

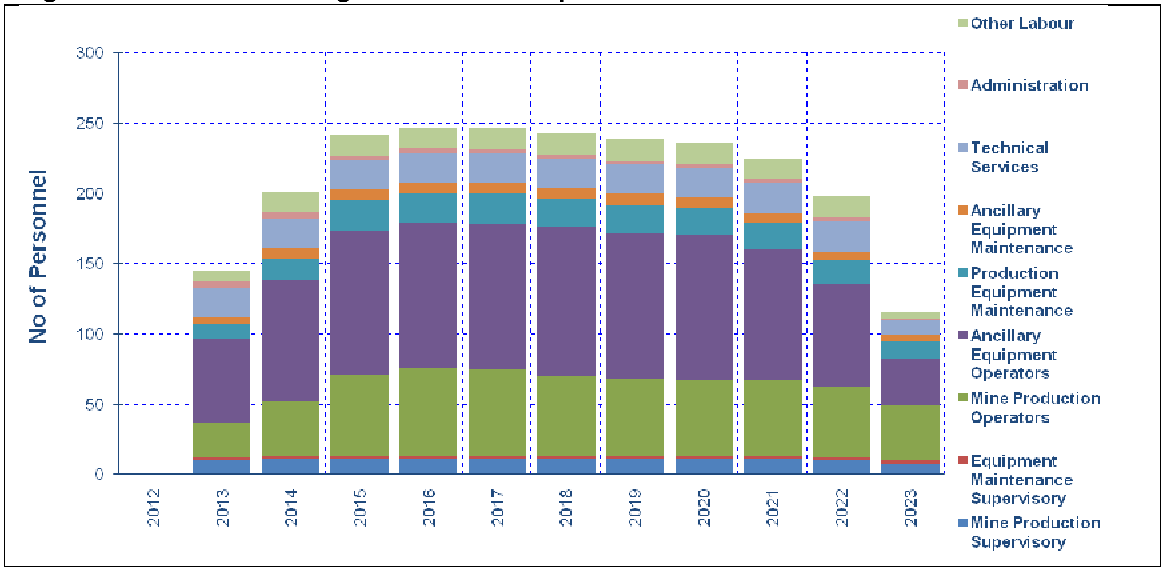
For the PFS, it was assumed that underground mining would be Owner operated. Specialist contractors would be used for specialised activities such as raise boring and diamond drilling activities.

The mine is planned to operate 24 hours per day, 365 days per year and mine operators will work a 14 days on, seven days off roster. Shifts will be of 10 hours duration.

The majority of management and technical support personnel, with some maintenance personnel, will be day shift positions only, working a five-day week. This will exclude shift geologists and underground shift supervision.

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

Figure 16-10: LOM Mining Workforce Complement



16.16 Underground Infrastructure and Services

The Owner's team will install, operate and maintain all underground infrastructure and services.

Underground infrastructure and services proposed and budgeted for the PFS include:

Controls and Communication System

- Leaky feeder system and telephone network.

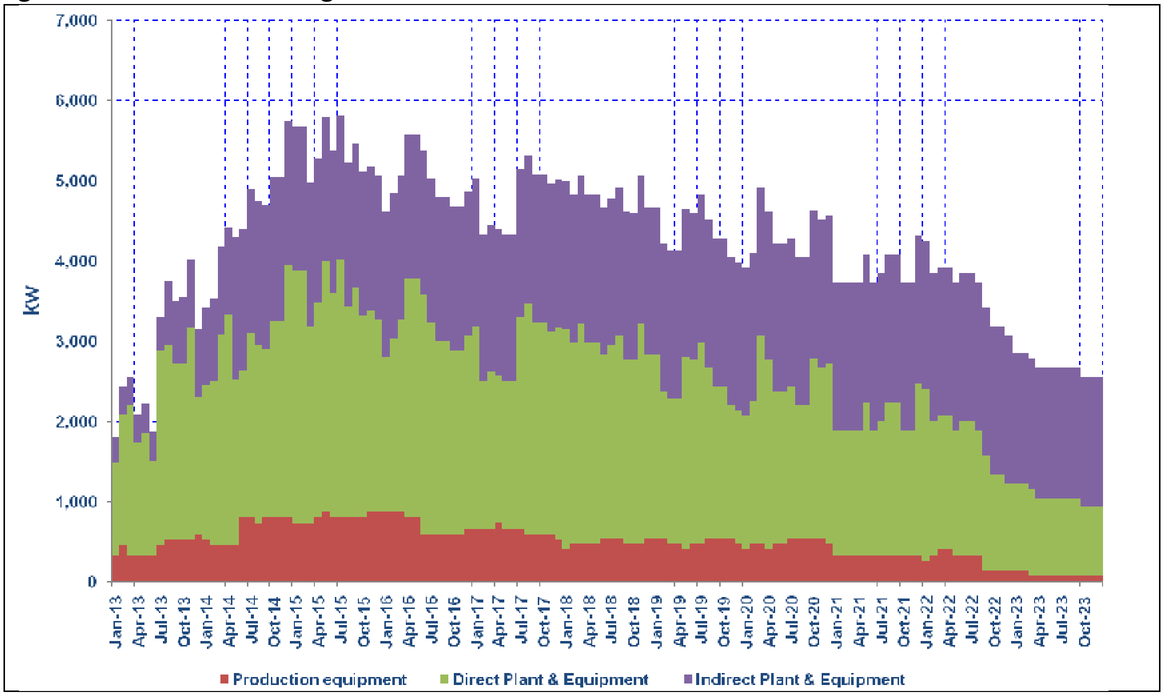
Underground Power

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

- 460kV 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 16-11.

Figure 16-11: LOM Underground Installed Power



Compressed Air

- Compressor to be located on surface and distribution network to fixed installations and level main accesses.

Potable Water

- Distribution network to be installed in the main level accesses.

Service Water

- Service water to be supplied from the processing plant. Estimated peak demand is approximately 600 m^3 per day.

- A service water pump and distribution network to main stope accesses.

Mine Dewatering

No geohydrological work was provided to Coffey Mining during the completion of the mining component of PFS. Mine dewatering requirements were based on an initial estimate of mine water inflows encountered during diamond drilling.

The main water sources for the mine are:

- ground water (estimate provided by AMEC), 83.3 L/s;

- service water, average 4.6 L/s, maximum 8.7 L/s; and

- water for flushing paste backfill distribution line (30 m^3 every second day).

The mine dewatering system consists of:

- Main pumping station and pump to be located in the vicinity of the end of the Exploration incline. The pump will be high volume, low pressure because the majority of water will free drain through service holes to the pump station;

- Mono pumps for the lower decline area in the East Minapampa domain, 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.

Underground Lunch Room

- Underground workers to use the lunch room for their mid shift break.

Refuelling and Service Bay

Underground drilling equipment, jumbos and production drills 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 dedicated underground service bays.

Explosives Magazine

Explosives will be primarily stored on surface and transported underground to a working party magazine on as needs basis.

The operation and size of the underground magazine should be reviewed during feasibility-level studies, with the final design based on Peruvian legislation and regulations.

17.0 Recovery Methods

The Ollachea mineral processing plant will include circuits for crushing, grinding, gravity concentration and intensive cyanide leach of gravity concentrate. Gravity tailings will be sent to a CIL circuit. Gold recovery from CIL solutions will be by electrowinning and refining to produce doré on site. Tailings will be treated by the INCO process for cyanide destruction then thickened and press-filtered to produce a filter cake for disposal at a dry-stack tailings storage facility (TSF). The plant will have circuits for water treatment, carbon regeneration, reagent preparation, and compressed air.

17.1 Plant Design Criteria

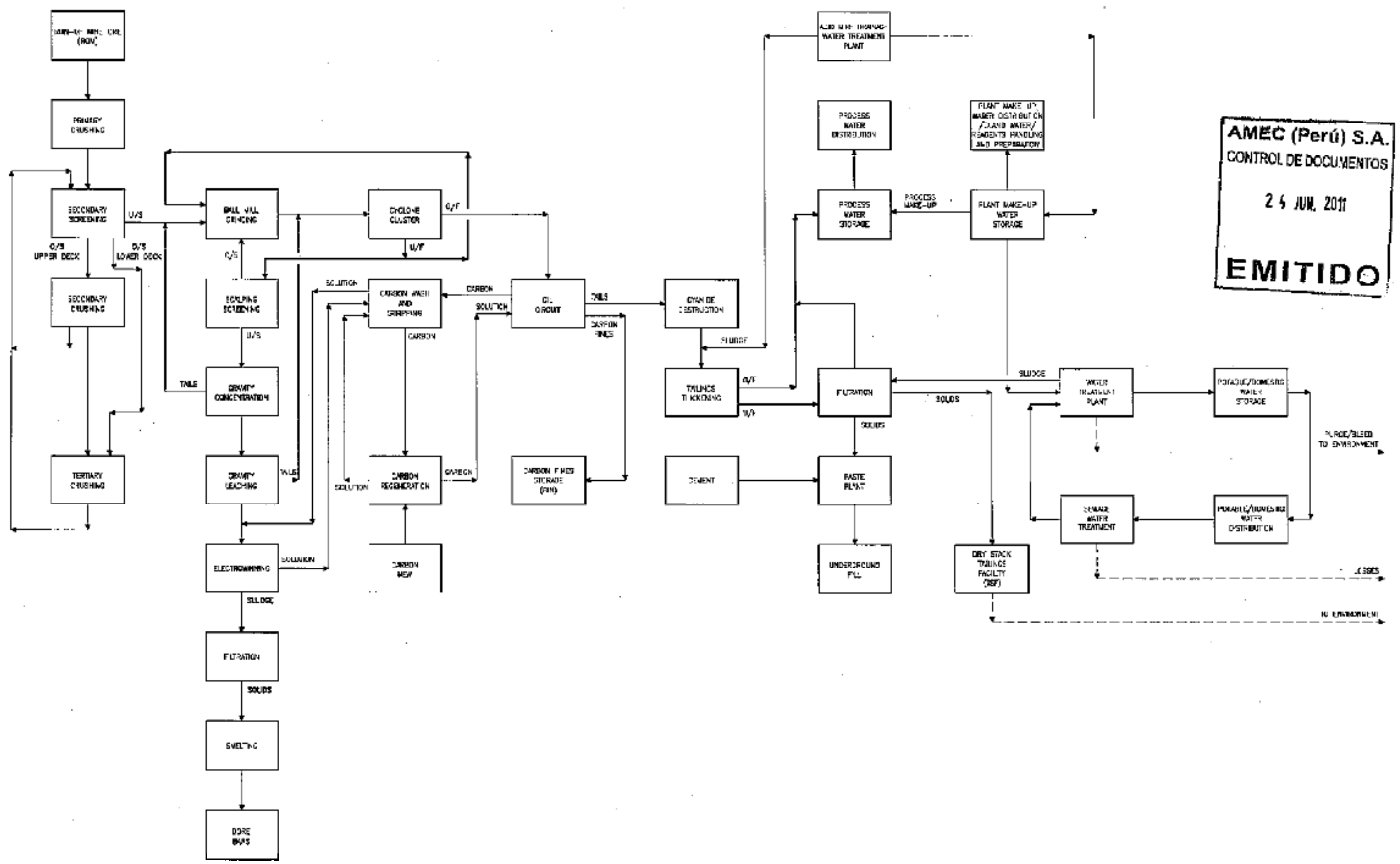
The design parameters of the plant are:

- Plant throughput: 1.1 Mt/y (137.5 t/h at 93.1% availability)
- Maximum ROM feed size: 500 mm
- Final product grind: P80 of 75 µm
- Head grade (LOM Average): 3.65 g/t Au
- Residue Grade (LOM Average): 0.32 g/t Au
- Overall Recovery: 91.3%
- CIL feed solids: 40%
- CIL residence time: 50 h
- Final tailing cyanide destruction: SO_2/Air/Cu^{2+} catalyst

17.2 Plant Design

A process block diagram is shown in Figure 17-1.

Figure 17-1: Ollachea Process Plant Block Diagram



17.2.1 Crushing

Ore coming from the mine will have a maximum size of 500 mm and will be transported by low–profile underground mine trucks of 36 t capacity. Trucks will dump ore directly to a fixed grizzly with 500 mm openings on top of a coarse ore bin (80 t) which will feed the primary vibrating grizzly and from there deliver ore to the primary jaw crusher (1,000 mm x 760 mm). As necessary, ore can be dumped to a stockpile near the mine portal and be reclaimed by a front–end loader (15 t) and transported to the mill hole to make up for mine feed. In the event that the delivery of ore to the primary crushing plant is interrupted, the stockpile live capacity will provide sufficient ore of continuous operation of the entire process downstream.

The crushed material will be directed a double deck screen (1.5 m x 4.8 m), the upper deck sieve opening will be 35 mm and the lower will be of 13 mm. The oversize of the top deck and the low deck will be directed to the secondary (914 mm) and tertiary (914 mm) cone crushers, respectively. The discharge of both crushers will be fed back to the screen. The undersize of the screen, the final crushed product, will have a grind size of P80 of about 8 mm and will be sent to the fine ore storage bins (2 x 330 t).

Crushed ore will be reclaimed through vibrating feeders (2 x 600 mm x 1,200 mm) and a conveyor belt and will be sent to the ball mill in the grinding and gravity intensive leaching circuit.

The crushing circuit will have a dust collection system. Water sprinklers will be placed at the discharge points of each conveyor belt to minimize dust emission.

17.2.2 Grinding/Gravity Intensive Leaching (GIL)

A single stage ball mill (5 x 8.5 m, drive 3 MW), in closed circuit with a cyclone cluster (10/2 x 254 mm), will be used to grind the feed ore from F80 of 8.6 mm (F100 of 13 mm) to 80% passing 75 µm. The mill will be run at a fixed speed.

A portion of the cyclone underflow will be bled off to the gravity scalping screen (2,400 mm x 900 mm) to remove the +2 mm material, which will report to the ball mill feed hopper while the screen undersize will be passed through a centrifugal unit Knelson concentrator (800 mm) to recover the free gold. Gravity tailings will be directed back to the mill feed hopper while the free gold concentrates will be discharged into a storage hopper from where they are treated through an Acacia unit of 1 t capacity. A dedicated electrowinning cell is proposed for treatment of gold–rich pregnant solution from this intensive cyanidation unit (ICU). The cyclone cluster overflow will be fed to the CIL circuit.

The ball mill will be fed freshly-crushed ore, returned slurry, process water and kerosene, the latter with the intention of inhibiting the adsorptive capacity of gold to carbon present in the ore.

Balls will be loaded to the ball mill by a hoist and a hopper. A sump pump will be installed in the grinding area to collect spillage from the ball mill.

Due to the addition of sodium hydroxide, sodium cyanide and leach aid to the ICU, and kerosene to the ball mill, a safety shower is necessary in this area. The safety shower is activated by a foot pedal and equipped with an eye wash station.

17.2.3 Carbon in Leach (CIL)

The CIL circuit will comprise eight CIL tanks each with a total capacity of 1,104 m^3, giving a total residence time of 36 hours. The CIL circuit will be fed from cyclone overflow via a trash removal screen.

The finer classified solids (F80, 75 µm) which will pass through the cyclone overflow as dilute slurry will be passed over a trash screen (3,700 mm x 1,200 mm, 0.6 mm opening) on the way to the CIL circuit. Any entrained trash such as woodchip fiber and plastics will be removed from the pulp during at this screen. The trash will be removed and contained in a bin for disposal. The trash screen undersize will be controlled at 42% solids and gravitates into the CIL feed distribution. The dimensions of CIL tank sizes have been calculated to accommodate a 50 hr residence time. This residence time equates to an individual live tank volume of 1,830 m^3. The CIL tanks will be mechanically agitated and provided with oxygen spargers to aid dissolution.

In the CIL circuit, slurry will be introduced into the first CIL tank (N°1) and fresh carbon will be introduced into last CIL tank (N°7). The CIL tanks will be all fitted with carbon inter-stage pumping screens and carbon transfer pumps.

The slurry will gravitate from the CIL Tank N° 1 and then be transferred to the consecutive CIL tanks by means of the inter–stage screens MPS 450 Kemix model (0.8 mm aperture) until being discharged from CIL Tank N° 7.

Two pneumatic gates will be installed in the launders of all but the first and final CIL tanks. One gate will divert slurry to the next CIL tank (normal flow), while the second gate will divert the flow to the previous CIL tank. Thus, for example, the CIL Tank N° 2 can feed either CIL Tank N° 3 (normal flow) or CIL Tank N° 1. It will be possible to by–pass any of the CIL tanks in the event of breakdown or for maintenance.

The CIL tailings slurry will be screened by the carbon safety screen (3700 mm x 1200 mm, 1 mm opening) to recover misreporting carbon which will drain to a carbon fines bin.

The pH of the slurry will be a measure of its acidity/alkalinity. At a pH below 9.3, the cyanide begins to decompose, which can have the following detrimental consequences:

- Insufficient cyanide available to dissolve gold, resulting in increased loss of gold to the tailings and elevated cyanide requirement thus increasing 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.

For these reasons it is necessary to add milk of lime to the process in order to maintain protective alkalinity, i.e. a high pH level. The pH must be maintained at 10.5 or above. For this reason, milk of lime will be added to the CIL feed distribution box ahead of the CIL tanks, and into CIL Tank N° 1,CIL Tank N° 3 and CIL Tank N° 5.

Cyanide is the most important reagent to be used in the plant as it is required for gold and silver leaching. Cyanide can also form complexes with copper, zinc and other base metals present in certain minerals in the ore in addition to gold and silver. For this reason, cyanide solution will be s added to CIL Tank N° 1 and the ability to add cyanide solution directly into CIL Tank N° 2, CIL Tank N° 3 and CIL Tank N° 5 will also be available.

Cyanide will be added as a 30% sodium cyanide (NaCN) solution. Sufficient cyanide must be added to satisfy for all these requirements. If insufficient cyanide is added, the quantity of undissolved gold losses will increase, and recovery will decrease. The cyanide consumption is projected to be 2.67 kg/t ore.

Slurry containing loaded carbon will be pumped intermittently from CIL Tank N° 1 by transfer pump N° 1 to feed the loaded carbon screen. The screen undersize will return to CIL Tank N° 1 and the washed “loaded” carbon will drop directly into the acid wash column in the carbon wash and stripping circuit.

To maintain the desired carbon concentration in the respective tanks, carbon will be advanced from any CIL tank to the adjacent upstream CIL tank, counter–current to the slurry flow by carbon transfer pumps located in each tank (transferred over 18 h/d, transfer pumps N° 2 to N° 7). Also, the design has allowed for loaded carbon to be transferred one time a day for a period of approximately four hours for each transfer (transfer pump N° 1).

Once loaded carbon has been recovered to the acid wash circuit, inter–tank carbon transfer will be performed concurrently i.e. carbon transfer pumps will run at alternate times to transport carbon from tank to tank, counter-current to the slurry flow, to rectify carbon concentrations in all seven tanks. Once the loaded carbon has been recovered from CIL Tank N° 1 to the loaded carbon screen, the carbon transfer will commence from CIL Tank N° 2 to CIL Tank N° 1. Once the desired carbon concentration (10 g/L) is achieved in CIL Tank N° 1, the transfer will be stopped and the transfer pump in CIL Tank N° 3 will be started to transfer carbon to CIL Tank 2. This procedure willo continue, until carbon has been transferred out of all the CIL tanks in the same manner.

Quenched regenerated carbon will be returned to the CIL section from the regenerated carbon kiln via the carbon sizing screen (1800 mm x 900 mm, 1 mm opening). This screen will be provided with spray water to ensure removal of carbon fines prior to introduction into the circuit. The fine carbon will be dropped over the carbon safety screen from where it is sent a carbon fines bin for disposal.

A gantry crane will be placed in the CIL area. The crane will primarily be used for periodic removal of the inter–stage screens for cleaning. A dirty screen will be replaced with a cleaned spare inter–stage screen. The dirty screen will be placed in the screen wash frame for cleaning using wire brushes and a high–pressure wash water pump. The screen wash frame will be located within the bermed CIL area.

The CIL area will be provided with two sump pumps delivering spillage back to the CIL distribution box. Due to the presence of milk of lime and the addition of cyanide at the CIL, a safety shower will be necessary in this area. The safety shower will be activated by a foot pedal and equipped with an eye wash station.

17.2.4 Carbon Wash and Stripping

The extent of calcium scaling of the carbon will dictate the required acid strength and acid wash period. It may be necessary to increase the acid strength or extend the wash period if the carbon is badly fouled. If it is required to complete the elution procedure more frequently to meet gold production targets, the acid wash period may be reduced or the acid wash step may be omitted completely.

Reduced or omitted acid wash can only be done as a temporary measure. The long–term effect of not acid washing loaded carbon will be to reduce carbon adsorption activity and may lead to poor gold recoveries in the CIL.

The elution process will be a two column split AARL process in which acid wash and elution are completed in separate columns. Each column will be capable of handling 6.7 t of carbon. The fill sequence will be initiated by pumping carbon from CIL Tank N° 1 into the acid wash column via the loaded carbon screen. Carbon will flow by gravity 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 by stopping the loaded carbon transfer pump.

The acid wash sequence comprises water injection into the column and simultaneously starting the acid dosing pump. The acid and water will be mixed in the manifold beneath the acid wash column and from there will enter the column. This will operate for a fixed period of time and will result in the column voids being filled with 3% w/v hydrochloric acid. Once the acid injection has been completed the acid pump will stop and a timer will hold the acidic solution in the column for a fixed period of time for the acid soaking of the carbon. The acid rinse cycle will then be started by pumping water through the column to displace the spent acid solution to the tailings thickener. A set volume of water (4 BV, 2 BV/h) will be pumped through the column at a preset time interval. This sequence will conclude and the carbon will then be hydraulically transferred to the elution column

The elution sequence will comprise the injection of a set volume of water into the column with the simultaneous injection of cyanide and caustic solution. A set amount of the reagents will be added to achieve a 2% w/w NaOH and 2% NaCN solution. Both additions will be stopped automatically once the prescribed volume has been added. The pre–soak period will commence where the solution is circulated through the column and pre-heated to 95 °C. At the completion of the pre–soak period the elution will commence and the gold will be stripped from the carbon. In this step the eluate emerging from the heat exchanger will be redirected to the eluate tank for electrowinning.

Elution will commence for a fixed time period to allow a set number of bed volumes of transfer water (or low gold content starter eluate from the lean eluate tank) to pass through the column (4 BV, 2 BV/h). Prior to reaching the elution column manifold water will pass through the heater heat exchanger (coupled with a diesel–fired heater) to raise the elution solution temperature to 120 °C.

A temperature probe in the eluate solution after the heater will be used to control the heater output to maintain the temperature. The eluate solution will gravitate back into either the elution tank or the lean eluate tank after passing through the heat recovery exchanger and will be recycled back through the elution column via the heater. The heat recovery exchanger will reduce the discharge eluate temperature whilst indirectly pre–heating the lean strip solution inflow and also reduce the flashing in the electrowinning cells.

After cessation of the elution, the heater will be switched off allowing the column and its contents to cool down to below 100°C.

The heat exchangers may experience scaling that will reduce their effectiveness over time. A de–scaling facility is provided. A regular check of the temperature and pressure indications at the heat exchanger inlets and outlets and calculation of the temperature and pressure differentials over the heat exchanger will indicate if it is scaled up and requires cleaning. A regular de-scaling program may be necessary to ensure optimal elution performance.

17.2.5 Carbon Reactivation

The carbon will be hydraulically transferred from the elution column after the elution process to the regeneration circuit using water. The water will be delivered to the base of the column at a suitable pressure to transfer the carbon to a carbon drain screen prior to being fed to the kiln feed hopper (fitted with a screw feeder at the bottom to control feed rate). The carbon will be fed at a rate of approximately 350 kg/h into the kiln which is operated at 650°C (low foulants assumed). The hot carbon discharge will fall into a quench tank which will be flooded with a pre–determined flow rate of water. The carbon will then be pumped onto the regenerated carbon sizing screen at the last CIL tank.

17.2.6 Electrowinning and Refining

The electrowinning sequence will be initiated by starting the eluate pump as soon as there is sufficient solution in the eluate tank. The pump will transfer the solution to the electrowinning cell (sludging basketless type, 1,000 mm x 1,000 mm cathode, 18 cathodes, rectifier 2,500 A) in the gold room. The gold will be electroplated onto the steel wire mesh cathodes. The cell discharge will be sent back to the eluate tank and will be recycled until the precious metals have been recovered from the eluate. Similarly, a gravity dedicated electrowinning cell (sludging basketless type, 600 mm x 600 mm cathode, 10 cathodes, rectifier 1,000 A) will be fed with pregnant solution coming from the gravity-intensive leaching circuit.

After the electrowinning is complete and once the cathodes are plated, they will be removed from the electrowinning cells and the gold sludge washed off the cathodes. The sludge recovered will be filtered by using a mobile pan vacuum filter and then dried in a calcine oven. The filter cake will then be mixed with a prescribed flux and then charged to the smelting diesel–fired crucible furnace. The gold will be poured into moulds, cooled and the gold bars will be cleaned, weighed and then placed in the vault. The slag produced will be recycled via the grinding circuit.

A cathode hoist will be provided for maintenance. The electrowinning and refining functions are enclosed in a secure area with limited access. A secure room with vault door is provided to store dried electrowinning cell sludge and gold bars.

17.2.7 Tailings Handling and Cyanide Destruction

CIL tailings will be sent directly to the cyanide detox tank where sodium metabisulfite, air, copper sulphate pentahydrated (catalyst) and milk of lime will be used to complex the residual cyanide. The sodium cyanide in the detoxified tailings can be reduced to below 5 ppm of weakly aid dissociable (WAD) cyanide. Residence time of the pulp in the detoxification tank will be 90 minutes. Detoxified pulp will flow by gravity to a high rate thickener (diameter, 26 m) to produce a thickened pulp of about 60% solids. The thickener underflow will then be fed to the pressure filtration circuit to produce cake with moisture of about 16.9% w/w. Depending on the need for backfill, the filtered product will be sent to either the paste plant or the filtered tailings load out area.

The paste backfill plant will contain two filtered tailings storage bins, each of 150 m^3 capacity. Each bin will have a screw feeder which will draw the cake to a continuous paste mixer. In the paste mixer, cement will be added from two storage bins of 150 m^3 each. The paste mixture produced will be discharged to a bin and sent to the mine portal by means of a positive displacement pump.

When the filtered product will be sent to the dry stack tailings disposal facility, the filter cake will be discharged from the reversible conveyor to the filter cake load out area. Once the material is deposited at the load out, it will be loaded to standard road trucks for haulage to the tailings disposal facility for their final deposition.

17.2.8 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.
- 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.
- ICU Oxidant (Leach Aid) to improve leach times and overall recovery.
- Kerosene is used as blanking agent to passivate the surfaces of activated carbon in carbonaceous ores.

Reagents will be delivered to the Ollachea site by road transport and will be transferred to the mixing tanks, and then the reagents will be pumped to the holding tanks within the plant area for final delivery to the process plant.

17.2.9 Water Treatment Circuit

Water will be used in the plant at several positions and will be sourced from the acid rock drainage (ARD) treatment plant. Treated ARD water will be fed to the plant make–up water tank at a rate of 73.9 m^3/h. From there, water will be added to the process tank at a demand of 31.9 m^3/h. It has been estimated that over the first years of operation, that the emerging acid mine water volume will not meet water plant requirement. During these occasions, river water will be available in various locations at the plant and will be treated according to the end-use requirements.

17.2.10 Compressed Air Circuit

High-pressure air will be provided in the plant by two systems (lead/lag). High-pressure plant air will be produced by rotary screw type compressors, and will be used for the instrument air and all plant services.

Low-pressure air will be produced by centrifugal type compressors. Low-pressure air will be used for carbon in leach, for carbon combustion (kiln) and for cyanide detoxification. The primary distribution system to the tanks will be by upcomers and non–return valves located at specific points below the agitators in each CIL tank.

17.3 Comment on Item 17

The proposed Ollachea plant design uses gravity and CIL technology appropriate to achieve a reasonably high recovery of coarse and fine gold from a preg-robbing pyrrhotite-bearing slate. Process design is based on metallurgical test work including comminution, gravity concentration, leaching, thickening and filtration work complete to date. The plant throughput rate selected is appropriate given the PFS mine plan presented in Section 16.

18.0 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 and to the Ollachea Project is relatively straightforward and road construction to provide access to the mine, plant, camp and TSF is minimal.

The Ollachea Project is within 200 m of the Interoceanic Highway. A road of approximately 600 m in length will be 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 require an access road that is approximately 200 m long and will be built during construction. The access road to the planned TSF is approximately 1,000 m long and has been considered in the PFS TSF design and capital cost estimate.

Current surface exploration drill roads will be used to provide access to ventilation raise surface breakthrough locations above the mine in the Minapampa area. These roads will allow ventilation fans to be installed and maintained.

No additional road construction is contemplated for the project.

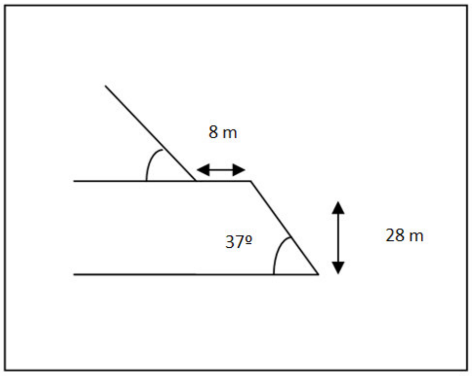
18.2 Waste Storage Facilities

The Ollachea PFS mine schedule has surface waste disposal requirements of 550 kt and maximum ore stockpile requirements of approximately 171 kt during the life of mine. Temporary waste storage will be required at the upper portal area; however, the waste temporarily stored at the upper portal will be brought back into the mine and used for backfill so no long-term or permanent waste storeage capacity is required at the upper portal near Minapampa.

18.2.1 Waste Dump Design Criteria

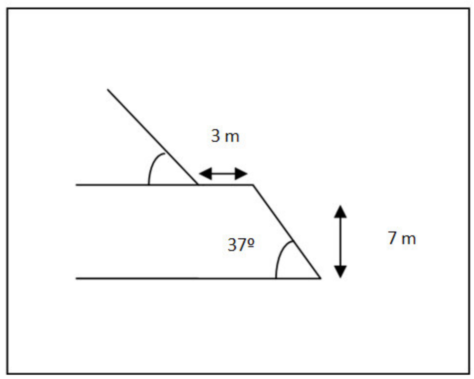
Design criteria for the dumps and stockpiles are shown in Figure 18-1.

Figure 18-1: Waste Dump and Ore Stockpile Design Criteria



- Waste Disposal (Base construction):
- Lift Slope Angle: 37º
- Lift Height: 28 m
- Bench width: 8 m
- Swell Factor: 20% (Compacted)
- Specific Gravity: 2.8
- Max Lifts: 3

- Ore and Waste Disposal (On Top):
- Lift Slope Angle: 37º
- Lift Height: 7 m
- Bench width: 3 m
- Swell Factor: 20% (Compacted)
- Specific Gravity: 2.8
- Max Lifts: 3



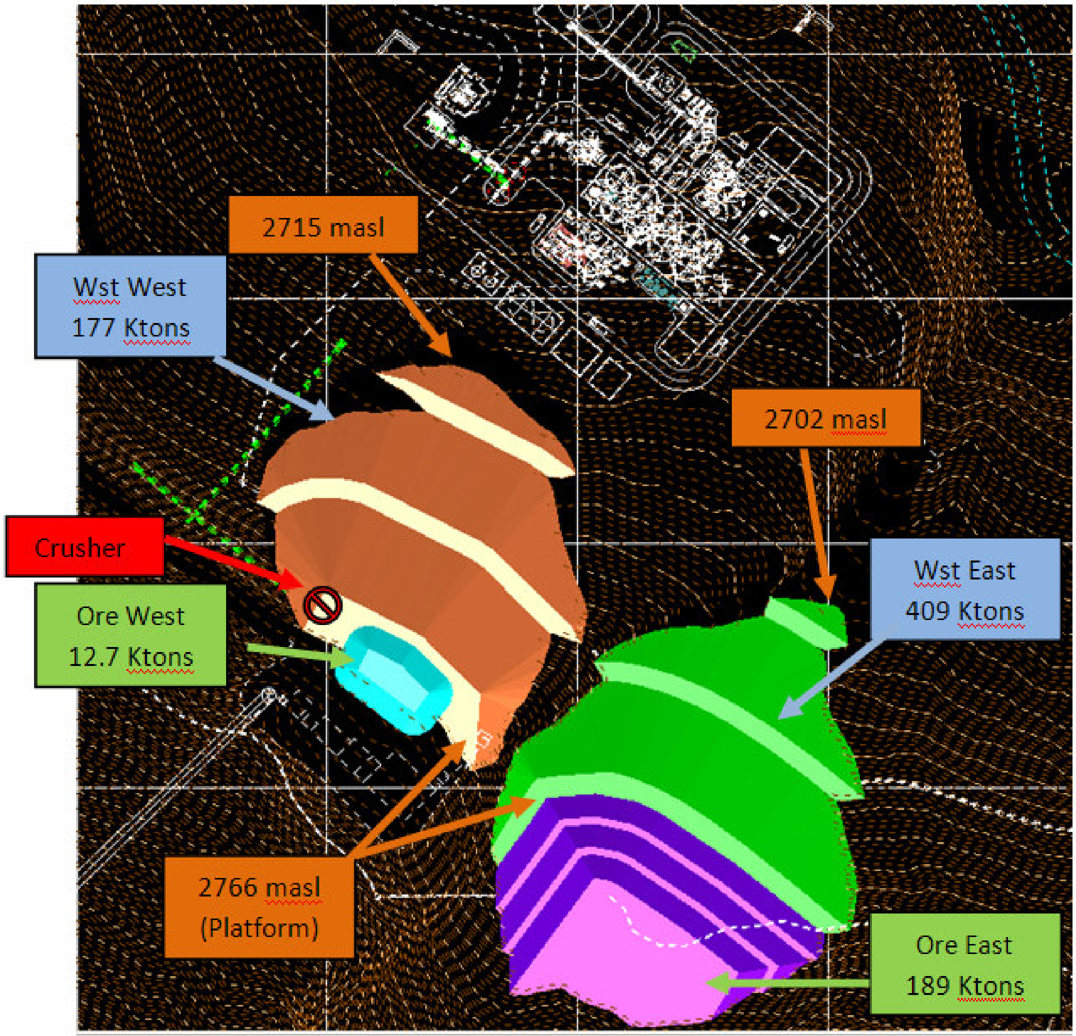
18.2.2 Waste Dump and Stockpile Design

Waste dump and stockpile volumes are listed in Table 18-1. A waste dump and stockpile design for the portal area is shown in Figure 18-2.

Table 18-1: Stockpile and Waste Dump Design Volumes

	Vol (m^3)	Kt
Ore	86,700	201.7
Waste	251,200	586

Figure 18-2: Stockpile and Waste Dump Configuration



18.3 Tailings Storage Facilities

AMEC prepared a Pre-feasibility TSF design that included a TSF site selection study (AMEC, 2011a), TSF design for the recommended location, and capital and operating cost estimates for the TSF.

18.3.1 Tailings Management Background

The anticipated tailings management strategy consists of approximately 25-55% surface storage with the remainder of the tailings placed underground as paste backfill. For surface disposal, filtered, or “dry stack”, tailings disposal has been selected as the most suitable tailings management option for this Project (Coffey, 2010 and AMEC, 2011a and 2011b). Conventional slurry tailings disposal and thickened tailings disposal were discarded as viable tailings management alternatives due to topographic constraints, as well as other risk factors.

A TSF site selection study considering five TSF alternatives was completed by AMEC in early 2011 (AMEC, 2011a). The site selection study ranked the five potential TSF sites based on perceived economic, technical, and social risk factors using available information. At the completion of the PFS, at least two viable options were identified and negotiations to acquire the property rights to these sites are presently underway. The PFS considered one of the two options, but the exact location of the selected option is confidential due to the ongoing negotiations with surface rights holders. The selected preferred alternative is located within 10 km of the plant site and was selected primarily due to a superior ranking with respect to proximity perceived tailings management risk.

AMEC considers that there is a reasonable expectation that positive negotiations can be concluded, and the ground acquired. However, in the eventuality that the preferred site cannot be used, MKK has made provision for an alternative site, where the company controls a significant portion of the necessary surface rights. AMEC notes that as site investigations in the proposed development schedule are planned for later in 2011, the actual site selection will need to be reviewed in terms of the acquisition of surface rights prior to the commencement of this work.

A Pre-feasibility design has been developed for the dry stack TSF at the preferred site location. Filtered tailings will be hauled by truck from the plant site to the preferred TSF using the Interoceanic Highway. A smaller, contingency TSF is planned close to the plant site to provide short-term tailings storage during times of limited access to the principal TSF or for upset plant conditions.

Several key issues and assumptions relating to the Pre-feasibility design for the principal and contingency tailings facilities follow:

- Additional geochemical testing of the tailings material is required. This testing will be necessary to predict water quality of any TSF effluent and to refine water treatment practices and costs. While there is limited, if any, “effluent” from a dry stack facility, there will be runoff water that will interface with the tailings.

- It has been assumed that installation of a liner system will not be required for the TSF. This assumption needs to be further evaluated based on characterization of the tailings and site specific TSF foundation conditions. Moreover, compacted filtered tailings have a hydraulic conductivity of the same order of magnitude as a typical liner. It is not common practice to place a liner under a filtered tailings facility.
- No geotechnical site investigation work has been conducted at the TSF sites to support the design. Geotechnical investigations will be an important element in verifying design assumptions and refining the TSF designs for feasibility study.
- Global stability of the principal TSF site needs to be further evaluated based on results from the geotechnical site investigation.
- Usage of the Interoceanic Highway for tailings haul traffic requires further evaluation with respect to permits, regulations, costs and maintenance requirements. Fugitive dusting considerations will be an essential component of these evaluations.

18.3.2 TSF Design Concepts

Based upon evaluative work, the most appropriate concept for surface tailings storage for the Ollachea project is filtered tailings. Filtered tailings storage will allow for the safe and efficient tailings placement within the rugged terrain of the project area.

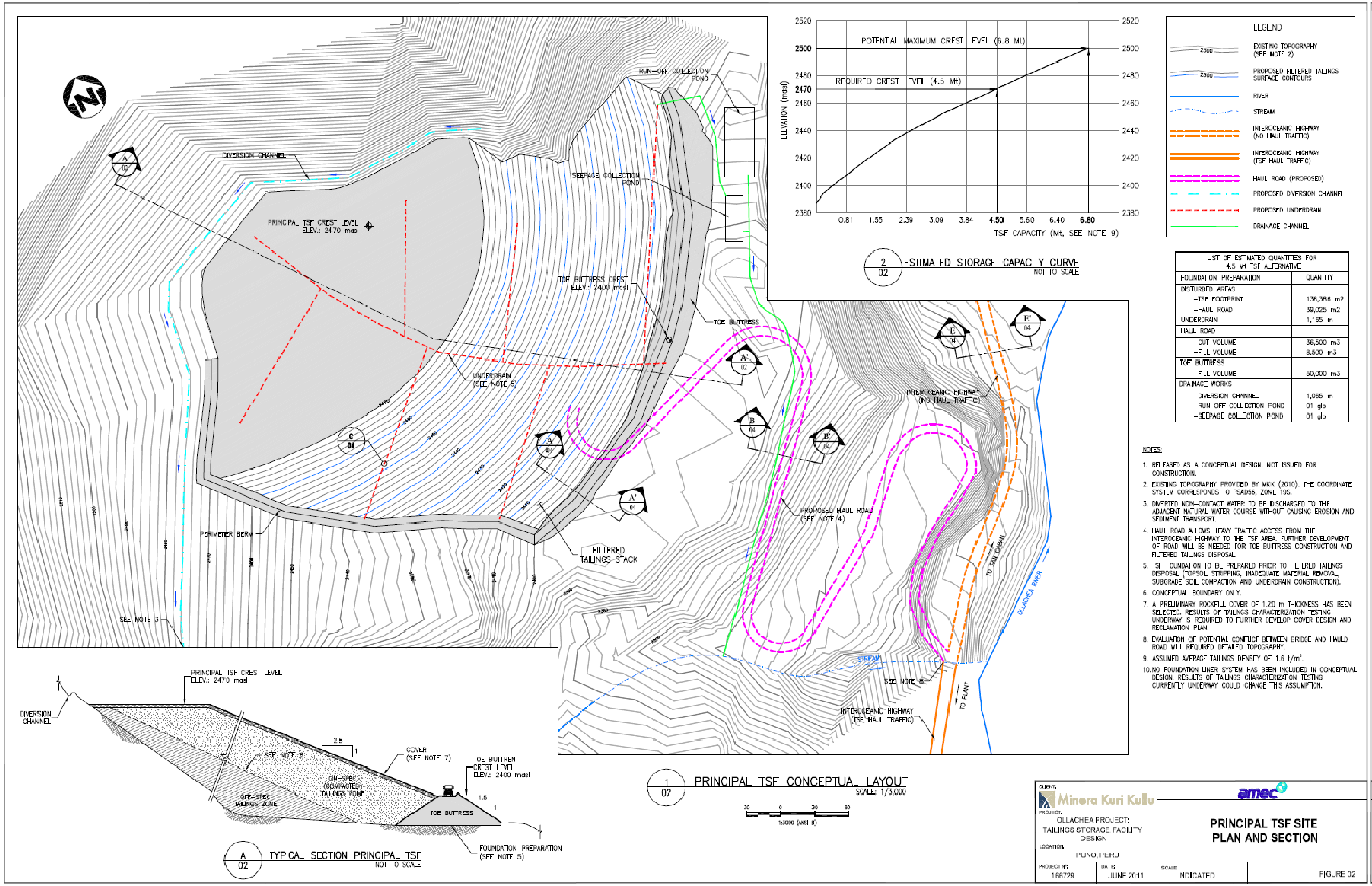
The principal TSF site has been designed for a capacity of 4.5 Mt of filtered tailings with overall dry stack slopes of 2.5H:1V. This geometry results in an approximate ultimate height of 75 m. Additional tailings storage is possible by increasing the height of the TSF, which could give a maximum estimated 6.8 Mt of total storage. A contingency TSF site has also been selected near the plant site to provide temporary tailings storage during periods of restricted access to the principal TSF or upset conditions at the plant. The contingency TSF has been detailed to provide an estimated 63,000 t of tailings capacity (approximately four weeks of tailings production). A general layout of the principal TSF is presented on Figure 18-3.

The tailings will be dewatered at the process plant site after passing through a cyanide detoxification circuit. It is assumed that the tailings can be dewatered to a moisture content at or slightly above (not more than 2%) the “optimum moisture content” for the tailings. The optimum moisture content is determined by moisture-density relationships developed by the Standard Proctor Test. At this time, given the grind and mineralogy of the tailings, pressure filtration is anticipated. Filtered tailings will be hauled by 15 m^3 trucks to the preferred TSF site, located within 10 km of the plant site.

The filtered tailings will be placed on a prepared foundation with an underdrain system to drain the foundation. This system would also conceptually capture seepage from the TSF, although properly developed dry stack facilities rarely exhibit any seepage. A small rockfill toe buttress will be constructed along the down-gradient perimeter of the TSF. Tailings will be placed in two zones: (i) a formally compacted zone at the perimeter shell of the TSF, and (ii) an "off-spec" nominally compacted zone in the interior of the TSF. The formally-compacted tailings act as a structural component for stability of the dry stack. The off-spec zone is intended for use when the design tailings compaction cannot be achieved (e.g., during wet or freezing weather or upset plant conditions). The off-spec tailings will be compacted separately to ensure they reach adequate mechanical characteristics.

A rockfill layer will be progressively placed on the tailings slope during operations to protect against erosion and will become part of the reclamation surface. Surface water run-on will be minimized by construction of perimeter surface water diversion channels. Runoff water in contact with the tailings will be routed off the TSF to a lined sedimentation pond and monitored for water quality. Effluent from the TSF, if any, will be collected in a lined pond and treated as needed to meet water quality standards.

Figure 18-3: Ollachea PFS TSF Design



18.3.3 Pre-feasibility TSF Design Criteria

The study was conducted using the design criteria presented in Table 18-1. Operating costs for tailings management have been estimated based on the design criteria presented here, then scaled to reflect the final PFS tailings production and backfill schedule for the purpose of the PFS Operating Cost (Section 11) estimates.

Table 18-2: TSF Design Criteria for Pre-feasibility Study

Criteria	Unit	Value	Basis
Mine Operations Criteria			
Total tailings production	Mt	9.1	B
Mine life	Year	11	B
Percentage surface tailings disposal	%	25-55	B
Percentage underground tailings disposal	%	45-75	B
Minimum tailings storage capacity – Principal TSF	Mt	2.3-4.5	B
Potential need for additional tailings storage	Yes/No	Yes	A
Tailings transport	-	15 m^3 Trucks	D
Dry Stack TSF Facility			
Principal TSF Storage Capacity	Mt	4.5	A
Contingency TSF Storage Capacity (minimum)	t	20,000	D
Filtered Tailings Properties			
Average dry unit weight	t/m^3	1.6	D
Effective friction angle compacted tailings	degrees	35	D
Effective cohesion compacted tailings	kPa	0	D
Effective friction angle “off-spec” tailings	degrees	30	D
Effective cohesion “off-spec” tailings	kPa	0	D
Able to achieve suitable moisture content by filtration for stacking?	Yes/No	Yes	D
Stability Evaluation			
Effective friction angle foundation soils	degrees	34	D
Effective cohesion foundation soils	kPa	0	D
Seismic design acceleration [Note 1]	g	0.24	E
Seismic coefficient	-	0.12	D
Minimum Static Factor of Safety	FS	1.5	C
Minimum Pseudo-static factor of safety	FS	1.1	C

Basis for Design Criteria:

A Information provided by MKK
B Coffey Mining (2010)
C Industry Standard of Practice
D Assumed by AMEC
E Information provided by others

Notes: Peak ground acceleration for 475-year return period based on Aguilar and Gamarra (2009).

18.3.4 Design and Operational Components

The following subsections present the considerations that will need to be involved in the detailed design and construction of the principal TSF. Most considerations are applicable to the contingency TSF as well.

18.3.5 Foundation Preparation

Dry stack facilities have the advantage over traditional slurry tailings facilities in that, like waste rock dumps, they can be developed on a wider range of foundations and still provide acceptable performance. With any facility that is constructed to 75 metres in height, a weak foundation is not an option, but discontinuous or very thin layers of soil should not form an issue for the dry stack. Foundation preparation will consist of clearing and grubbing of significant vegetation in the shell (formal compaction) area and removal of topsoil and other organic material as is practical and deemed required by the engineer. The topsoil, as it exists, can be stockpiled for reclamation purposes. Other unsuitable materials identified during geotechnical investigations or construction may require removal to form an acceptably competent foundation material. These materials, if any, should be disposed of or stored in approved containment facilities. Topsoil or inadequate material storage areas were not evaluated during this study; however, there appear to be suitable areas near the principal TSF site.

Boulders on the current ground surface at the principal TSF, both naturally occurring and in man-made walls, could be collected and used for construction materials such as channel armouring and toe buttress construction.

18.3.6 Foundation Underdrains

Given the limited filtration test information, an underdrain system for collection of tailings leachate has been conservatively included in the design. The design includes a primary underdrain running through the central portion of the TSF with lateral connecting drains branching off either side of the primary drain. The underdrains are anticipated to consist of free-draining material placed within a trench excavated into the foundation and wrapped with non-woven geotextile to act as a filter. The underdrain system will direct effluents to a lined collection pond near the dry-stack. Water quality will be monitored and treated as necessary to meet quality standards before discharge to the environment. Performance from other dry stacks would indicate that there is a possibility, if supported by site-specific tailings filtration testing, that the drainage system can be removed during the feasibility design stage of the project.

18.3.7 Liner System

This study has assumed that a liner system will not be required for the principal TSF as seepage volumes are expected to be very low due to the unsaturated condition of the dry stack and anticipated low hydraulic conductivity of the unsaturated tailings. The need for a liner system will be verified once physical and geochemical test results are available for the tailings during feasibility-level design work.

18.3.8 Toe Buttress

Construction of a compacted rockfill toe buttress will be required along the down-gradient perimeter of the TSF. The toe buttress will provide additional integrity to the dry stack to resist erosion. The toe buttress was designed with 1.5H:1V slopes with a 7 m-wide crest and maximum height of approximately 10 m.

18.3.9 Tailings Transport

Filtered tailings will be transported from the plant site to the principal TSF using 15 m^3 capacity trucks. The tailings transport route includes haulage along the existing southern Interoceanic Highway. Additionally, an approximately 1,000 m long haul road will need to be pioneered from the highway to the principal TSF site. A preliminary layout of the off-highway haul road indicates approximate maximum grades of 10%.

It is assumed that tailings haulage and placement will be carried out by a contractor and supervised by MKK. It is expected that haul trucks will require cover systems to control dusting and spillage during tailings transport.

18.3.10 Tailings Placement

It is anticipated that tailings will be hauled and dumped at the TSF site by trucks. A dozer (D7 or D8) and motor grader are expected to be used for spreading tailings into lifts and maintaining truck access corridors on the TSF. Tailings compaction will be by vibratory smooth drum compactors and truck traffic. A method specification versus a performance specification will be established prior to facility construction and calibrated each year.

The design considers two zones for tailings placement, one for formally-compacted “on-spec” tailings and another for upset “off-spec” tailings that will be provided through the same compactive effort but be considered “nominally compacted” from a stability perspective. A diagram of placement and compaction is provided in Appendix E. The formally-compacted “on-spec” tailings zone will form a structural “shell” that will be located at the down-gradient perimeter area of the TSF. Tailings would be compacted during good weather and normal operating conditions. This zone will act as a structural component of the TSF. Tailings shall be placed in 0.3 m-thick lifts that will be compacted to at least 95% of the maximum dry density as determined by the standard Proctor test (ASTM D698) and will be based upon a number of equipment passes (i.e., method specification) that will be determined at later project stages.

The “off-spec” tailings zone will be located between the formally-compacted tailings zone and the prepared foundation slope. This zone will allow for tailings placement during conditions when specified compaction cannot be assured (e.g., wet or freezing weather conditions, upset plant conditions). Tailings placement in this zone will consist of 0.5 m thick lifts, compacted as per the formally compacted zone and the tailings are assumed to provide no structural integrity.

The tailings surface in both zones will be rolled to minimize infiltration and be sloped to shed surface water away from the structural zone, toe buttress, and other sensitive areas of the TSF. Water should not be allowed to pond on the TSF surface. The runoff water will be directed off the TSF as efficiently as practicble to channels conveying the runoff to a sedimentation pond. Contact water will be monitored to ensure compliance with quality standards prior to discharge to the environment.

Control of fugitive dusting during operations and closure is an important concern for dry-stack tailings facilities. Tailings slopes should be progressively reclaimed during operations to limit dusting as discussed in Section 8.1.12. 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.

The dry stack has been designed for maximum overall slopes of 2.5H:1V. Truck access to the TSF during operations is anticipated via an 8 m-wide haul road constructed switch-backing up the finished TSF slope. The haul road will also serve to provide surface water drainage and limit exposed slope lengths for erosion control.

18.3.11 Progressive Slope Reclamation

Protection of the tailings slope from water and wind erosion is a major concern as the dry-stack tailings are considered to be highly erodable. Dry stack tailings facilities present the benefit of allowing progressive reclamation of tailings slopes. At a minimum, a rockfill cover will be maintained on the final tailings slopes to protect against erosion. Additionally, surface water channels will be constructed at regular intervals on the TSF slope to limit the unbroken runoff slope lengths.

18.3.12 Surface Water Management

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 dry stack to discharge to the natural drainage down-gradient of the facility. Efforts should be made to minimize water run-on to the TSF. 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.

Precipitation falling directly on the tailings (contact water) will be directed off the TSF as efficiently as is practical. The contact water will be directed to a sedimentation pond for monitoring and treatment if necessary. If the contact water meets quality standards it will be discharged to the environment. Diversion channels and discharge points will protect against erosion.

18.3.13 Management of TSF Effluent

Effluent quantity from the dry stack is expected to be negligible based on experience and performance of other dry stack TSFs. An underdrain system has been included in the Pre-feasibility design to capture seepage and direct it to a collection pond. Effluent will be monitored and treated as necessary to meet quality standards before discharge to the environment.

18.3.14 TSF Stability Evaluation

Analysis Methods

The minimum safety factors adopted are 1.5 and 1.1 for static and seismic analyses, respectively.

Pseudo-static analyses were used to model seismic conditions. The compacted filtered tailings will be dilatant and not strain softening, so pseudo-static evaluative procedures are valid.

The peak ground acceleration for the project site was estimated to be 0.24 g based on data presented by Aguilar and Gamarra (2009). A pseudo-static coefficient equal to one-half of the peak ground acceleration (i.e., 0.12) was assumed for the pseudo-static analyses. This approach is consistent with Hynes-Griffin and Franklin (1984); an industry standard.

Material Parameters

The materials modelled in the stability analysis included: (i) formally-compacted tailings, (ii) "off-spec" nominally-compacted tailings, (iii) rockfill, and (iv) foundation soil.

All material parameters were necessarily assumed since geotechnical investigations have not been conducted at the TSF site and no test results on the filtered tailings were available at the writing of this report. The assumed material parameters presented in Table 18-2 are based on AMEC's professional judgment and experience and observations of site conditions. The parameters should be refined after site investigation and geotechnical laboratory testing are carried out.

Table 18-3: Material Parameters Used in Stability Analyses

Material	Unit Weight (kN/m^3)	Effective Friction Angle (degrees)	Effective Cohesion (kPa)
Compacted Filtered Tailings	18	35	0
Off-Spec Tailings	17	30	0
Rockfill	21	38	0
Foundation Soils	20	34	0

Analysis and Results

A cross-section was developed through the maximum section of the TSF to represent the critical stability section. The section includes an overall tailings slope of 2.5H:1V and overall stack height of 75 m, as measured from the toe of the facility to the crest. A 65 m-wide zone of compacted tailings was assumed for the stability model.

Factors of safety of 1.8 for static conditions and 1.5 for pseudo-static conditions were calculated for critical failure surfaces through the filtered tailings. These factors of safety are considered acceptable and are consistent with guidance literature. Global factors of safety of 1.4 and 1.1 for critical failure surfaces extending from the toe of the TSF through the the slope below the TSF were estimated for static and pseudo-static conditions, respectively. The global stability of the TSF must be verified and refined following geotechnical investigations of the TSF. TSF Closure Considerations

Closure of the dry-stack TSF is expected to include construction of a 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 should 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. The water treatment facility design criteria are for a treatment rate of 350 m^3/h which is approximately the maximum potential mine inflow rate (AMEC, 2011).

18.5 Camps and Accommodation

A permanent operations camp facility has been designed and will be located at the Cuncurchaca area, about 1,000 m north of the plant site and mine portal and within 200 m of the Interoceanic Highway. The camp will have catering and accommodation capacity for approximately 300 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 derivation 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 and onsite emergency power generation equipment. A fuel storeage 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 can be supplied from the water treatment plan and drawn from the Oscco Cachi and Ollachea Rivers as required.

19.0 Market Studies

Doré production from the Project could be sold either on the spot market or under agreements with refineries. Sales and marketing considerations can be evaluated during feasibility-level study. It is expected that any sales and refining agreements would be negotiated in line with industry norms.

The current doré marketing arrangement and costs for the Ollachea PFS are based on IRL's arrangement for doré sales from the Corihuarmi mine.

20.0 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 pertain to the Atlantic Ocean basin. The maximum monthly average flow in the Ollachea river, at the height of the tunnel is of 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, coliform bacteria and thermotolerant coliform bacteria concentrations in the Oscoccachi stream, high iron concentrations in the Ollachea river and bacteriological contamination in springs.

Air quality meets Peruvian environmental regulations for lead, arsenic, PM_{10}, $PM_{2,5}$ SO_2, CO, NO_2, H_2S and O_3 concentrations. Baseline noise levels registered in the industrial areas of the study area were below the daytime and nighttime national environmental noise standards. Noise levels recorded in the town of Ollachea were above daytime and night time standards, mainly due to Interocean 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 categorized as 'endangered', the *Vultur griphus*. Additionally, 5 species of wild animals have been observed in the study area. Of those 5 species, two are protected species, the *Tremarctos ornatus* is endangered and the *Puma concolor* is near threatened.

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 have been 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. These surveys have

A reconnaissance of archaeological sites was done on the Project area. Two archaeological sites were identified, one consisting of a pre-hispanic road, constructed with large and medium size stones to provide a path on a steep slope and which is still used to this day. The second site is a small 2m by 1m construction in the shape of a chullpa, covered by shrubs and grass in the Challuno area, north of the proposed plant site.

Project design has been carried out taking into account these archaeological sites.

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 is 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 Current Environmental Liabilities

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 shown in Figure 6-1. 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.

20.3 Closure Plan

A formal closure plan will be developed as part of the feasibility work plan for the Project.

The extent of closure plans for Ollachea is restricted to the mine portal and mineral processing plant areas and are quite limited considering 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 PFS capital cost estimate for the Project.

20.4 Permitting

Tong (2010b) summarizes the permits in place for the Ollachea Project.

MKK currently holds permits allowing them to carry out exploration activities on the property

- Authorization by the National Water Athority 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
- 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.

Tong (2010b) concludes that these permits provide MKK with all necessary rights to conduct their current and all planned operations and exploration activities.

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.

20.5 Considerations of Social and Community Impacts

MKK has conducted continuous 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.

20.6 Comment on Item 20

MKK has received permits to continue to operate exploration activities on the property and to excavate an exploration incline 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 current permitting process in Peru, there are no 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. 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. IRL has successfully obtained these permits for its Corihuarmi operation, and MKK can be reasonably expected to obtain similar permits for Ollachea.

21.0 Capital and Operating Costs

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.

21.1 Capital Cost Estimates

The Ollachea PFS capital cost estimate consists of estimates of direct and indirect capital costs for the underground mine and pastefill backfill system, the mineral process plant, surface infrastructure and auxiliary building including electrical power supply, camp site and tailings storage facility.

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 surface infrastructure including the TSF and plant were estimated by AMEC. Estimates have been combined for the purpose of developing and integrated project capital cost estimate.

21.1.1 Basis of Estimate

The underground mine capital cost was estimated by Coffey Mining based on an Owner operator strategy with specialized contractors used for raise boring and diamond drilling activities.

The underground mine estimate is based on quotes from equipment suppliers, estimates from recent other projects and data and assumptions supplied by MKK and AMEC. In general costs have been built from first principles using the Project scheduled mine physicals as a base.

Mine capital costs change to sustaining capital costs and operating costs when stope production starts. First stope production has been scheduled to start in July 2014. Mine indirect costs have been split between development and production on a tonnage pro rata basis.

Labour Assumptions

Labour costs used for the Pre-feasibility underground mine cost estimate are shown in Table 21-1. These were derived from information supplied by MKK and AMEC.

Table 21-1: Underground Mining Labour Costs

Description	Code	Annual Cost (PEN)	Monthly Cost (PEN)
Mining Manager	SM17	305,800	25,483
Technical Services Manager	SM16	183,480	15,290
Mining Superintendent, OH&S Manager, Electrical Foreman, Maintenance Foreman, Senior Mining Engineer, Senior Geologist, Training Manager	SM15	152,900	12,742
OH&S Supervisor, Shift Boss, Geotechnical Engineer, Mining Engineer, Electrical Mechanical Engineer, Mine Geologist, Mine Trainers	SM14	91,740	7,645
Mine Maintainers (electrical and mechanical), Junior Mining Engineer, Junior Geotechnical Engineer, Surveyor	SM13	34,750	2,896
Mine Operators, Geological Technicians, Survey Assistants	SM12	30,580	2,548
General labour	SM11	22,240	1,853

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. The cost of construction equipment (estimated as dollars per direct work hour by prime account) includes equipment ownership costs, depreciation, insurance, fuel oil, lubricants, maintenance, and service and repair. Temporary power and catering were excluded from the labour rates and charged to project indirect costs.

The construction labour cost was based on a CM execution approach i.e. all construction will be by contractors.

Work hours were factored from material take offs. These productivities are standards derived from AMEC's standard work hours based on data from typical mining projects and contractor input.

Labour rates have an associated productivity factor that reflects the nature of the work and the working conditions. Productivity factors (or loss of productivity in the field) were established based on AMEC's standard work base hours, contractor information and benchmarking.

The productivity factors (multipliers) incorporated into the construction labour unit work hours applied to AMEC North American unit work hour standards.

Material Take-offs (MTOs)

Material take-offs were prepared for all facilities. Table 21-2 summarizes the percentage of the direct cost estimate by discipline based on MTOs and mechanical and design specifications. Percentages are expressed with respect to the value of the estimate. Also listed are the percentages of direct capital costs estimated from quotations or factored from process equipment amounts.

Table 21-2: Proportion of Estimate Types by Value of Estimate

Discipline		Estimate from Quotation (%)	Estimated from CSA MTO (%)	Estimate from Mechanical/Design Specification (%)	Factored from Process Equipment (%)
00	Mining	51.9	0.0	48.1	0.0
01	Earthmoving	0.0	32.3	67.7	0.0
02	Concrete	0.0	100.0	0.0	0.0
03	Structural Steel	0.0	100.0	0.0	0.0
04	Architectural	0.0	37.7	62.3	0.0
05	Mechanical	42.1	12.5	45.4	0.0
06	Piping	0.0	0.0	0.0	100.0
07	Electrical	17.8	2.3	35.9	44.0
08	Instrumentation	0.0	0.0	0.0	100.0
Total		35.8	14.1	43.3	6.7

Civil Earthworks

Earthworks quantities were taken off from AutoCAD 3D reports and manual templates. Unit prices were estimated using AMEC's database, construction equipment productivities and benchmarked against AMEC's experience.

Mass earthworks estimates were based on quantities generated from the site layouts. These quantities included such items as cut and fill, drilling and blasting, and topsoil stripping. Quantities calculated for the plant site were based on 2 m contours. Detail excavation and backfill quantities for buildings and structures were developed for each area and based on estimated foundation sizes.

Fencing quantities included the plant site and other perimeter security fences and electrical substation enclosures shown on the drawings.

A civil design growth allowance of 15% of the estimated cost of earthworks was assigned to the capital cost for the Project. At the request of MKK this allowance was not reported as direct capital.

Concrete Works

Concrete quantities were determined manually from layout drawings, conceptual concrete sketches and were to neat lines with no allowances for over-pour or wastage.

All concrete was classified into standard shapes. Quantities were provided for these shapes by engineering.

An average formwork usage and rebar requirement for each classification of concrete was used as the basis for pricing based on the seismic rating at the site. Concrete unit costs are all-in and include concrete, reinforcing steel and formwork. The concrete price included supply of concrete from an on-site batch plant. A 5% allowance was included for over-pour, wastage and rework.

The concrete price was developed from first principles based on a batch plant on site provided and operated by a contractor and aggregates obtained within 15 km from the site. It is proposed that a concrete batch plant contract be arranged with an experienced contractor for the supply, manufacture, and transportation of concrete on site.

Formwork was estimated for each classification of concrete. Allowances were made for form oil, accessories, shoring and formwork materials as required. The unit price also included the fabrication, installation, stripping and cleaning of formwork.

Reinforcing steel unit costs for installation were derived from AMEC experience on recent projects.

A design growth allowance of 10% of the estimated cost of concrete works was assigned to the capital cost for the project. At the request of MKK this allowance was not reported as direct capital.

Structural Steel

Structural steel and miscellaneous steel quantities were developed directly from general arrangement drawings. An allowance was made for connections, stiffeners, clips, hardware and base plates.

Material supply, detailing, shop pre-fabrication and painting with a primer coat were included in the material unit costs which were taken from AMEC's estimation database. Labour hours shown in the estimate are for field erection.

A civil design growth allowance of 5% of the estimated cost of structural steel was assigned to the capital cost for the project. At the request of MKK this allowance was not reported as direct capital.

Architectural

A list of primary and auxiliary buildings was developed and the requirements of major buildings such as mine shop, camp dormitories, cafeteria, administration and security were determined and sketches developed for these buildings

The project building list and sketches were used to identify the size of the facilities. Costs for construction of the buildings were supplied by MKK from a quotation from the contractor responsible for the construction of the Corihuarmi camp. An additional factor was added to the construction and construction materials quotation to account for furnishings.

Plant Mechanical and Mobile Equipment

The process mechanical equipment list consists of tagged equipment by area. Major equipment prices quoted and estimated from quotations for similar equipment from the database. Estimates were based on costs for new equipment.

Major equipment costs were obtained from written quotations submitted by vendors with tendered vendor quotations representing approximately 45% of the total mechanical equipment cost. Estimates for minor equipment were based on AMEC in-house data for recently completed or quoted projects. Equipment costs were quoted delivered to Lima.

A purchase growth allowance of 2% for mechanical allowance was assigned to the capital cost for the project. At the request of MKK this allowance was not reported as direct capital.

Platework

Platework quantities were derived from the equipment list. In addition, sketches from similar projects were used as the basis for the estimated quantities for items such as bins, skirting, launders, chutes and pump boxes. The platework was priced on a cost per kilogram basis.

Field fabricated tank costs were based on budget quotations for supply and erection from Peruvian fabricators familiar with the area.

Piping

The cost of piping was estimated as a factor of process equipment installed in each area. The factor varied for dry and wet circuits within the plant. The overall average factor assigned was 7%.

Electrical

The electrical costs were based on the electrical equipment list and single line diagrams. The mechanical equipment list was used to cross check for equipment or packages requiring power.

Electrical equipment pricing for the substation and line connecting the substation to the power grid was based on vendor budget quotations. Equipment for electrical distribution on site was estimated based on specifications on the electrical equipment list. Costs for electrical equipment within each of the plant circuits were estimated to be 25% of the installed process equipment in each area.

Instrumentation

The cost of instrumentation was estimated as a factor of mechanical equipment installed in each area. The factor used was 7%.

The cost of communications equipment including the radio communication system and security/CCTV system is based on data from other AMEC projects.

The cost of cables, fibre optics, and other minor materials is considered in the estimated cost per instrument.

Construction Contractor Pricing

Prices for construction were obtained from the AMEC database.

Equipment and Material Pricing

Material Pricing

Bulk material pricing was Project- specific based on AMEC's database pricing.

Concrete unit rates were developed using supply and install information provided by contractors, assuming that the concrete will be produced in a batch plant fed with aggregates obtained within a 15 km radius.

Steel unit rates were developed using South American installation rates and supply pricing from a recent project with similar quantities.

Mechanical and Electrical Equipment Pricing

Recent letter quotations were used to estimate the cost of approximately 42% of the mechanical equipment cost. These packages included:

- Jaw Crusher
- Secondary Cone Crusher
- Tertiary Cone Crusher
- Vibrating Grizzly Feeder
- Double Deck Screens
- Ball Mill
- Scalping Screen
- Cyclone Cluster
- Cil Feed Trash Screen
- Loaded Carbon Recovery Screen
- Carbon Sizing Screen
- Carbon Safety Screen

- Cil Tanks and Agitators
- Elution Heater
- Heat Recovery Exchangers
- Tailings Thickener
- Positive Displacement Pumps and Surface Piping for the Paste Plant
- Electrical Substation Equipment
- Sewage Water Treatment Plant

The remaining mechanical and electrical equipment costs were priced from budget quotes or using in-house data.

Mechanical and electrical equipment was quoted FOB in the port of Callao. AMEC estimated the freight and shipping separately as part of the indirect costs.

Construction Cost Estimate

The initial construction cost of the Project is estimated to be US$12.7 M before contingency and allowances, and not considering construction of the underground mine.

The total construction cost of the surface installations for the Project is summarized in Table 21-3. Costs for pre-production mine development, the portal access road and the exploration drift are not included.

Table 21-3: Construction Cost Summary

Area		Earthworks (US$M)	Concrete Work (US$M)	Structural Steel (US$M)	Total Estimated Construction Cost (US$M)
1200	Surface Infrastructure for Mine	0.21			0.21
2100	Plant Area Preparation	1.17	0.12		1.29
2300	Main Access Roads				
2400	Camp Site Development	0.34			0.34
3100	Crushing	0.07	0.62	0.86	1.55
3200	Grinding/Gravity Intensive Leaching	0.02	0.61	0.39	1.02
3300	Carbon In Leach (Cil)	0.04	0.56	0.93	1.54
3400	Carbon Wash And Stripping		0.08	0.18	
3500	Carbon Reactivation		0.03	0.18	
3600	Electrowinning And Refining	0.01	0.18	0.07	0.26
3700	Tailings System - Plant Site	0.06	1.03	0.87	1.97
3800	Reagents Handling & Preparation		0.08		
3900	Plant Services	0.01	0.22	0.04	0.28
5200	Power Line and Substation	0.51	0.09	0.42	1.02
6100	Tailings Storage Facility	2.61			2.61
Grand Total		5.06	3.64	3.97	12.66

Tailings Storage Facility Capital Cost Estimate

The TSF capital cost estimate was prepared for the design discussed in Section 18.3.

TSF capital cost consists of predominantly costs for earthworks associated with the haul road, site preparation and the construction of the dry stack toe buttress. Water management ponds will be excavated to capture water running off the dry stack and TSF seepage. Groundwater quality and flow will be monitored by instrumentation installed in four boreholes around the TSF site with an estimated cost of US$ 0.5 M. Tailings site engineering procurement and construction management (EPCM) and construction quality assurance (CQA) support consisting of specialist consultants is assigned in addition to the engineering, procurement, and contract management (EPCM).

It is assumed that tailings haulage, placement and compaction will be carried out by a contractor so no capital costs for mobile equipment are considered.

Capital costs were not factored to account for the increased tailings capacity from the initial estimate used for TSF design discussed in Section 18 as the size of the toe buttress will not change.

The total capital cost for the Ollachea TSF is US$ 2.6 M (Table 21-4).

Table 21-4: Tailings Storage Facility Capital Cost Estimate

Description	Unit	Qty	Unit Cost (US$)	Cost (US$)	Basis
Site Preparation					
Site Preparation	m^2	48,500	3.66	178,000	(1)
Toe Buttress Construction	m^3	50,000	21.92	1,096,000	(1)
Ancillary Facilities					
Haul Road Construction	Km	1.6	231,145	370,000	(1)
Water Management Ponds	Glb	2	100,000	200,000	(2)
Instrumentation	Glb	1	500,000	500,000	(2)
TSF EPCM	%	3.0	2,343,124	70,000	(3)
Construction Quality Assurance (CQA)	Glb	1	200,000	200,000	(2)
Total Capital Cost	US$			2,614,000	

(1) Estimated costs from PFS design
(2) Assumed value factored from PFS design
(3) Factor

21.1.2 Indirect Construction Costs

Indirect construction costs include all-in rate labour cost. Costs are assigned based on factored estimates from the plant and surface infrastructure for the project. Factors are based on AMEC's recent experience in construction projects in the Americas. Table 21-5 lists factors used to estimate indirect capital costs for the construction of the project. Indirect capital costs are estimated to be 32.1% of direct Project capital costs for surface installations.

Table 21-5: Indirect Construction Cost Factors

Indirect Construction Costs	Percent of Direct Capital
Construction Management & Services	8.50
Temporary Construction Buildings & Facilities	1.50
Temporary Construction Services	0.80
Temporary Construction Equipment & Tools	0.19
Temporary Construction Camp	2.00
Catering	2.48
Construction Field Office Expense	1.50
General Engineering & Procurement	6.00
Vendor Representatives	0.75
Commissioning & Ramp-Up	0.20
Temporary Facilities	2.00
First Fills	1.00
Safety / Medical	0.75
Process Plant Spare Parts	0.68
Ocean Freight	2.75
Truck Freight	1.50

21.1.3 Owner's 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. It is assumed that these owners costs will 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. Owner's costs will be incurred over seven quarters from Q1 2013 to Q3 2014. Estimated owners costs are listed in Table 12-6.

Table 21-6: Owner's Costs

Project G&A		
Lima Office Overheads	US$	250,000
Owner Project Management	US$	1,904,500
Management & Secretarial	US$	765,000
Human Resources (HR)	US$	260,000
Accounting	US$	277,500
Purchasing & Warehouse	US$	333,000
Travel & Accommodation	US$	215,000
Catering	US$	231,000
Camp Cleaning & Maintenance	US$	32,200
Occ Health & Safety	US$	620,000
IT & Communications	US$	380,000
Office Furnature & Outfitting	US$	120,000
Office Maintance and Supply	US$	33,000
Insurance	US$	370,000
Legal	US$	85,000
Security	US$	125,000
Mining Leases & Rates	US$	40,000
Subtotal Owner's G&A	US$	6,041,200
EIA & Environment		
Environmental Management	US$	196,000
Environmental Programs	US$	14,000
Water Treatment	US$	88,500
Monitoring & Reporting	US$	63,000
Waste Management	US$	7,000
Post Commitment Permitting	US$	50,000
Subtotal EIA and Environmental	US$	418,500
Owners Capital Schedule		
IT Hardware	US$	175,000
Warehouse Outfitting	US$	150,000
Light Vehicles	US$	740,000
Subtotal Owner's Capital	US$	1,065,000
Total Owner's Cost	US$	7,524,700

21.1.4 Contingency and Escalation

Contingency is a monetary provision in the estimated total cost of a project to cover uncertainties or unforeseeable elements of time and cost within the scope of the project as estimated. The contingency does not allow for any scope changes. The contingency amount is an integral part of the cost estimate. It does not cover potential scope changes, price escalation, currency fluctuations, and allowances for force majeure, other project risk factors or any of the items that are excluded from the capital cost estimate.

The contingency for the mine, plant and surface infrastructure was estimated at 20%. In discussion with process and engineering staff, no variation by area could be justified.

21.1.5 Project Capital Cost Summary

The consolidated capital cost estimate for the Project totals US$ 169.5 M. Direct capital costs are estimated to be US$ 113.8 M, indirect capital costs are estimated to be US$ 19.6 M, Owner's Costs US$ 7.5M and contingencies and design growth allowances are estimated to be US$ 26.1 M. The estimated Project capital costs are listed by area and type in Table 21-7.

Table 21-7: Capital Cost Estimate for the Ollachea Gold Project

Description	Labor Man (h)	Labor Amount (US$M)	Material Amount (US$M)	Process Equipment (US$M)	Sub Amount (US$M)	Construction Equipment (US$M)	Total Amount (US$M)
Direct Capital							
Mine Capital							53.51
Site Development							1.83
Process Plant Electrical Supply	2,698	0.05	1.22		0.25	0.004	1.53
Crushing	63,401	1.11	1.29	3.31	0.72	0.22	6.64
Grinding/Gravity Intensive Leaching	62,905	1.16	2.13	5.59	0.27	0.20	9.35
Carbon In Leach (CIL)	39,139	0.55	1.24	1.96	3.10	0.11	6.96
Carbon Wash And Stripping	9,218	0.15	0.37	0.64	0.12	0.03	1.31
Carbon Reactivation	7,101	0.11	0.15	0.64	0.14	0.02	1.06
Electrowinning And Refining	14,767	0.27	1.09	1.49	0.10	0.05	3.00
Tailings Management System - Plant Site	70,026	1.19	1.77	4.95	0.74	0.24	8.89
Reagents Handling & Preparation	9,850	0.20	0.52	0.68	0.10	0.04	1.54
Plant Services	27,582	0.55	1.31	2.51	0.13	0.10	4.60
Site Utilities	6,776	0.13		1.15	0.33	0.01	1.62
Water Treatment System			0.01		0.80		0.81
Camp And Auxillary Buildings							8.50
Tailings Storage Facility					2.61		2.61
Indirect Capital							
EPCM							9.29
Construction							6.30
Vendor Representatives							0.45
First Fills And Spare Parts							1.01
Freight And Duty							2.56
Mine Contingency (20% of Direct and Indirect Capital)							10.20
Plant Contingency (20% of Direct and Indirect Capital)							16.47
Owner's Cost					7.52		7.52
Allowance For Design Growth - Mechanical/CSA					1.93		1.93
Total	313,464	5.48	11.10	22.92	18.86	1	169.50

21.1.6 Sustaining Capital Costs

Mine Sustaining Capital Costs

Mine capital costs are considered sustaining capital costs when stope production starts. First stope production has been scheduled to start in July 2014 (Q3, 2014). Mine sustaining capital costs include:

- Plant and equipment (including mobile equipment).
- Direct development
- Indirect development.

There are no plant or infrastructure sustaining capital costs during the life of the operation.

Closure Cost

A closure cost equal to 5% of direct capital for surface installations totalling US$3.14 M has been estimated for the project. The closure cost is relatively low as the underground mine surface footprint is negligible and the TSF is progressively closed during operation. A formal closure program and cost estimate will be required for feasibility-level studies.

21.1.7 Capital Cost Schedule

A capital expenditures schedule has been developed from the project execution plan. Mine development and indirect costs come from the PFS mine schedule and have been allotted as Project Capital and Sustaining Capital with project costs passing to sustaining costs when stope production begins in July 2014.

Plant and Infrastructure direct and indirect costs except for site preparation, will be split between 2013 and 2014 with 40% of costs allocated to 2013 and 60% of costs allocated to 2014. Site preparation costs will be incurred in 2013. Although site work will begin apon approval of the Project EIA in the final quarter of 2012, this work will likely not be expensed until 2013. Likewise, costs incurred at the end of 2013 may not be expensed until 2014. Owner's costs have been estimated by MKK on a quarterly basis based on their current cost structure and the anticipated Project requirements.

Site closure costs are divided equally between the second-last and finial production years.

The capital cost schedule is provided in Table 21-10.

Table 21-8: Consolidated Capital Cost Schedule

Consolidated Capital Costs	Unit	2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	LOM
Mine Production													
Mine Production	t	57	332	945	1,101	1,098	1,096	1,102	1,095	1,100	1,098	453	9,477
Mine Production Contained Gold	oz	6	39	102	128	132	132	146	135	132	116	44	1,112
Mine Production Grade	g/t	3.2	3.7	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.0	3.6
Stockpile Movement													
Stockpile (Closing)	t	57	172.5	17.1	18.1	15.6	11.8	13.9	9.4	9.6	7.5	0.0	
Stockpile Contained Gold	oz	6	20.4	1.9	2.0	1.7	1.6	1.8	1.0	1.1	0.8	0.0	
Stockpile Grade	g/t	3.2	3.7	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.0	
Commissioning													
Commissioning Feed	t		41										
Commissioning Contained Gold	oz		4										
Commissioning Head Grade	g/t		3.7										
Plant Production													
Plant Feed	t		176	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	460	9,477
Plant Feed	oz		20	121	127	132	132	145	135	132	117	45	1,112
Plant Feed Head Gold Grade	g/t		3.7	4.7	5.7	6.7	7.7	8.7	9.7	10.7	11.7	12.7	12.7
Plant Operating Quarters			1	4	4	4	4	4	4	4	4	4	
Plant Avg. Operating Capacity	%		100	100	100	100	100	100	100	100	100	50	
Gold Production			0	0	0	0	0	0	0	0	0	0	2
Capital Cost Schedule													
Mining Equipment	US$	16.6	10.4										27.1
Mining	US$	14.6	9.3										23.9
Mining - Backfill & other	US$	1.3	1.3										2.5
Site Development	US$	0.9	0.9										1.8
Process Plant	US$	22.4	22.4										44.9
Site Utilities	US$	1.2	1.2										2.4
Ancillary Buildings	US$	4.2	4.2										8.5
Tailings System	US$	0.4	2.2										2.6
Indirect Costs	US$	9.8	9.8										19.6
Indirect - Owners Costs	US$	2.6	4.9										7.5
Design Growth Allowance		1.0	1.0										1.9
Contingency - Mining (excl. backfill)	US$	6.2	3.9		-	-							10.2
Contingency - Directs/indirects (excl. Owners costs)	US$	8.1	8.4										16.5
Sustaining Capital	US$	-	6.3	7.7	4.2	5.5	5.3	6.0	4.7	3.5	3.9	-	47.0
Closure Cost											1.6	1.6	3.1
Total Pre-production capital cost		89.4	80.1										169.5
Total Sustaining Capital Cost		0.0	6.3	7.7	4.2	5.5	5.3	6.0	4.7	3.5	5.5	1.6	50.1
Total Capital Cost		89.4	86.4	7.7	4.2	5.5	5.3	6.0	4.7	3.5	5.5	1.6	219.6

21.2 Operating Cost Estimates

The Ollachea PFS operating cost estimate consists of operating costs for the underground mine, mineral processing plant including the tailings storage facility (TSF) and General and Administrative (G&A) costs for the integrated operation.

Coffey Mining estimated the mine operating cost and provided operating cost data and a description of the cost categories to MKK and AMEC for compilation into the consolidated operating cost for the project.

21.2.1 Mine Operating Cost

Basis of Estimate

The mine operating cost estimate is based on an owner operator strategy with specialised contractors used where required. For Pre-feasibilityPFS purposes, it was assumed that contractors would be used for raise boring and diamond drilling activities.

The estimate is based on quotes from equipment suppliers, estimates from recent other projects, MKK and AMEC supplied data and assumptions. In general costs have been built from first principles using the project scheduled mine physicals as a base.

Project capital costs change to sustaining capital costs and operating costs when stope production starts. First stope production has been scheduled to start in July 2014. Mine indirect costs have been split between development and production on a tonnage pro rata basis.

Mine operating costs have been split into four categories. These are:

- Direct development.
- Direct production.
- Indirect development.
- Indirect production.

Figure 21-2 shows the total annual mine operating cost split into the four categories and Figure 21-3 shows the total percentage split for the four categories.

Figure 21-1: Total Annual Mine Operating Costs

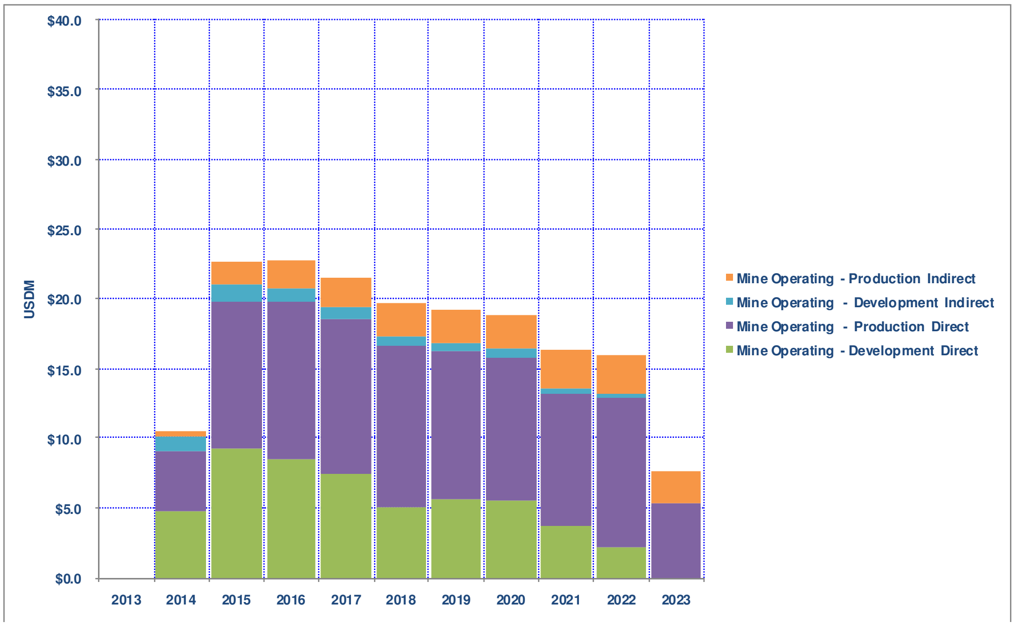
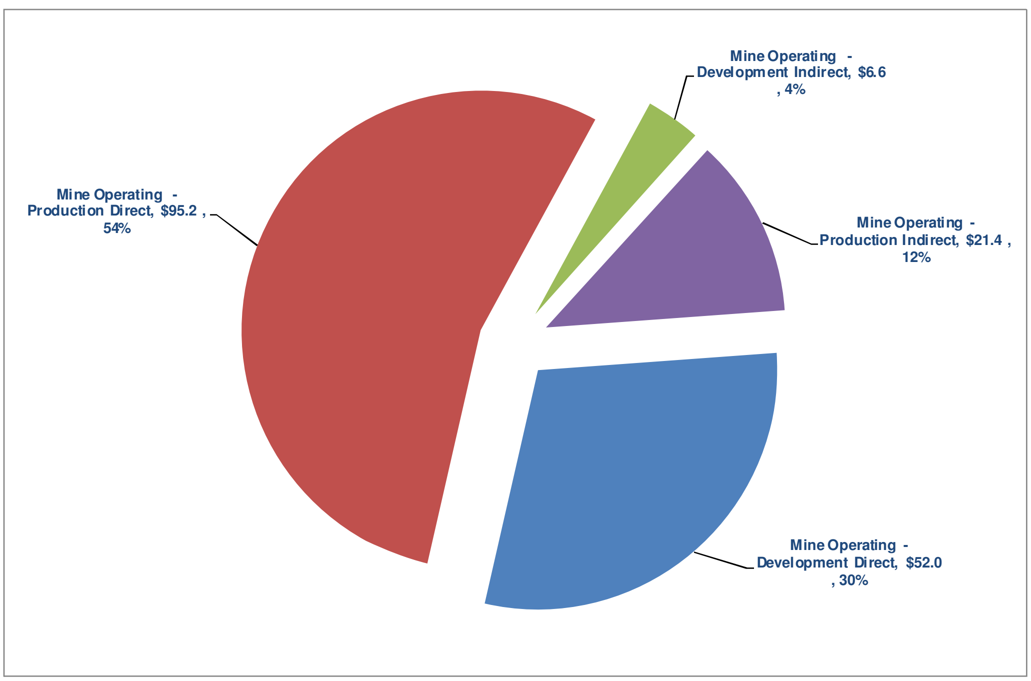


Figure 21-2: Breakdown of LOM Mine Operating Costs



Direct Production

The primary cost contributors for this category are:

- mobile and direct fixed plant operating and maintenance costs including backfill.
- operating and maintenance labour costs.
- materials and consumables costs (stope delineation diamond drilling, drill, blast, load, haul, backfill, ground support, services (including power, water, fuel), pumps and secondary fans).
- The LOM direct production cost for the operating mine is estimated to be US$ 10.15/t ore.

Indirect Production

The primary cost contributors for indirect production are:

- 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 manager, technical services manager, mining superintendent, OH&S manager and supervisors and shift bosses.
- Equipment maintenance supervisory labour, electrical and mechanical.
- Technical services labour which includes mining engineers, geologists and geological technicians, geotechnical engineers, surveyors and surveyors assistants, electrical and mechanical project engineers.
- 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.

The LOM indirect development and production costs for the operating mine are estimated to be US$ 0.70/t ore and US$ 2.28/t ore respectively.

Direct Development Ore and Waste

Costs in this category include operating development cost for stope accesses, cross-cuts and longitudinal development in ore and waste that 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). LOM cost is estimated to be US$ 1,221/m.

21.2.2 Plant Operating Cost

Operating Supplies

All supplies costs were developed from quotations received from local and overseas suppliers and from experience and contacts within the mining industry. Also, internal database on similar operations has been used.

Wear Parts

Parts replaced due to normal wear and tear are considered wear parts . Crusher and mill liners consumption were estimated based on benchmarking of similar plants. Grinding media consumption rate were calculated based on the Bond Abrasion index for the ore type considered.

Reagents and Consumables

Reagents and consumables costs were estimated based upon throughput, feed grade and metallurgical testwork. Unit prices were obtained from suppliers for all reagents and consumables.

Main Services/Utilities

Power

Power was calculated by plant area with consumed and installed power derived from the Mechanical Equipment List. Escalation of the power cost was not included.

The process plant power costs are based on an anticipated average continuous power demand. The average continuous power demand for each duty drive has been calculated from the installed power applying various utilization and mechanical efficiency factors, depending on drive type and duty.

During power outage, a dedicated diesel-based power generator (rated 30 kW) will supply energy to office buildings, security, maintenance shops and warehouses.

Plant Make-Up Water

Water consumption was obtained from flowsheets and the plant general water balance.

Fuel - Diesel

Fuel consumption rate is based on equipment performance and average fuel consumption rates, light vehicles and plant mobile fleet, diesel usage, and diesel-based generator average consumption.

Others/Miscellaneous

Miscellaneous plant costs will account for unforeseen or additional costs (internal/external laboratory assays, light vehicles and mobile equipment running costs, etc.).

Mobile Fleet - Spares and Consumables

A schedule of light vehicles for staff (administration) and mobile fleet for plant operations was developed based on the labour and departmental requirements. This was used to estimate the annual running costs.

Maintenance Supplies

Maintenance supplies costs comprise repair parts (mechanical, electrical and instrumentation equipments replacement parts), tires,major overhauls, maintenance materials, as well as pipes and fittings, etc.

Plant maintenance supplies are factored from the direct installed capital cost estimate of the plant (per area). Allowances have been considered for plant related services.

These costs excluded crusher components, crusher liners, mill liners and lifters and any other component listed as operating supplies.

Maintenance labour and diesel fuel required for mobile equipment and diesel-based generator costs are excluded.

Manpower

Manpower is based on the number of personnel required to cover the following items in accordance to typical local mining organizational charts. It does not comprise expatriate personnel for commissioning and up-front training prior to production. Only true operations personnel have been considered.

Labour areas for the plant are:

- Mill Supervision.
- Plant Operations.
- Laboratory. On-site laboratory service contract has not been considered.
- Maintenance.

Average wages were developed from Client data, a survey of national salaries in Peru (Compensation Management Tool (Deloitte, 2010)) and from benchmarking the salary structure against other local mining operations. A base yearly remuneration rate for local personnel is in accordance with skill level and responsibility. Also, the loadings or burden rates applicable to those base rates have been considered to enable determination of the total amount payable to each individual. Burden rates covers elements such as social security benefits.

Plant operations personnel will work 14 days on at 12 hours per day followed by 7 days off.

Life-of-mine plant operating costs are listed by area in Table 21-9. Plant operating costs consist of fixed costs for labour and mobile fleet and variable costs for the remaining areas. Variable costs are significantly greater than fixed costs.

Table 21-9: Plant Operating Costs

	Annual Costs		Unit Costs	
Item	1st Year US$/y	Average LOM, US$/y	1st Year US$/t	Average LOM, US$/t
Operating Supplies				
Wear Parts	406,147	2,177,544	2.3	2.3
Reagents And Consumables				
Reagents (Process Plant)	2,201,971	11,805,786	12.5	12.5
Consumables (Carbon, Fluxes...)	28,404	152,285	0.2	0.2
Services/Utilities				
Power (Average Demand)	573,231	2,033,379	3.3	2
Fuel - Diesel	39,128	818,078	0.2	0.9
Elution Circuit Thermal Oil (Heating Oil)	302	1,618	0.002	0.002
Assays And Quality Control – Lab. Equip.	13,600	72,916	0.08	0.08
Others/Miscelaneous	6,400	34,313	0.04	0.04
Plant Mobile Fleet - Spares And Consumables	7,656	41,047	0.04	0.04
Sub-total	3,276,839	17,136,967	18.6	18.2
Maintenance Supplies				
Processing Plant	99,665	534,350	0.6	0.6
General	27,200	145,832	0.2	0.2
Sub-total	126,865	680,182	0.7	0.7
Manpower				
Process Plant	304,966	1,081,781	1.7	1.1
Total Plant Operating Cost	3,708,670	18,898,930	21.1	20
Variable Costs	3,276,839	17,136,967	18.6	18.2
Fixed Costs	*431,831*	*1,761,963*	*2.5*	*1.8*

TSF Operating Cost

Operating costs for the Ollachea TSF have been estimated assuming a dry stack facility will be operated at the site identified within 10 km of the process plant.

Operating costs include load and haul from the plant site by a contractor to the TSF and placement, compaction, coverage, side berm, site prep and underdrain construction, highway maintenance, seepage and runoff coverage and construction management support and CQA (Figure 21-4). The TSF will be operated largely by contractors with a contract for tailings haulage, construction, maintenance and heavy equipment operation. Operating costs have been estimated based on a design for a 4.5 Mt TSF facility and factored to the 5.9 Mt PFS study requirement based on the PFS tailings production and stope backfill budgets (Table 21-10).

Figure 21-3: TSF Operating Cost Chart

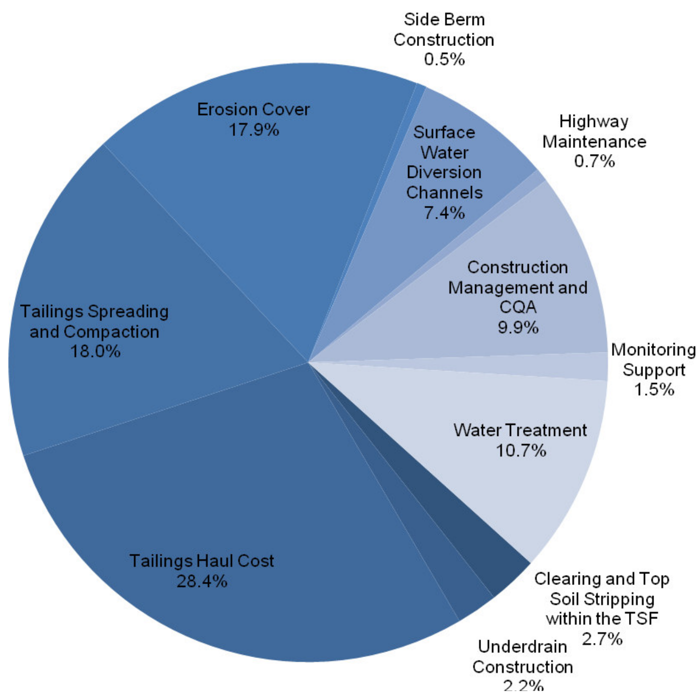


Table 21-10: TSF Operating Cost Summary

Description	Units	Qty	Unit Cost (US$)	Subtotal (US$)	Basis
TSF Operations					
Clearing and Top Soil Stripping within the TSF	m^2	300,000	3.66	1,098,162	(1)
Underdrain Construction	ml	1,500	606	908,408	(1)
Tailings Haul Cost	Tonne	5,900,000	1.97	11,613,849	(1)
Tailings Spreading and Compaction	Tonne	5,900,000	1.25	7,378,571	(1)
Erosion Cover	m^2	200,000	36.63	7,325,835	(1)
Side Berm Construction	m^3	10,000	21.92	219,151	(1)
Surface Water Diversion Channels	Glb	1.2	2,529,000	3,034,800	(2)
Highway Maintenance	Units	1.3	230,000	299,000	(2)
Construction Management and CQA	Glb	1.2	3,360,000	4,032,000	(2)
Monitoring Support	Glb	1.3	480,000	624,000	(2)
Water Treatment	Glb	1.3	3,360,000	4,368,000	(2)
Total Operating Cost	US$			40,901,776	
Unit Operating Cost	US$/t tailings			6.93	
	US$/t ore			4.32	

Basis:
(1) Estimated costs factored to 5.9 Mt capacity from 4.5 Mt PFS design
(2) Assumed value factored from 4.5 mt PFS design
(3) Factor

21.3 G&A Cost

G&A costs have been estimated from a staffing list for administrative personnel, by benchmarking to similar operations and using data supplied by MKK from operations at Corihuarmi.

General and administrative costs consist of labour and overheads for:

- General Management
- Human Resources.
- Medical Assistance/First-Aid.
- Accounting. This site will administer its accounting functions in conjunction with the Lima office.
- Purchasing. The supply chain group will be responsible for the purchase of all materials, contracting of services, management of in-bound logistics and warehousing. Ordering of materials and services will be from both, site and Lima office, as appropriate.
- Health, Safety And Environment (HSE).

In addition to labour and overheads, G&A will also cover:

- Third Party Services: Catering, Camp Cleaning, Maintenance And Rubbish Removal
- External Assays
- External Consulting And Software
- Equipment And Vehicle Rental (Heavy And Light)

- IT And Communications
- Safety/Protective Clothing (EPP)
- Postage, Courier And Light Freight
- Office/Computer Supplies/Maintenance/Supplies
- Medical Assistance/First Aid
- Travel/Accommodation/Camp
- Access And Internal Roads Maintenance
- G&A Mobile Fleet (Light Vehicles Maintenance)
- Insurance/Legal Service
- Recruitment And Training
- Environmental Monitoring And Reporting/Environmental Programs/Waste Management
- Site Security (External)
- Procurement And Importation Costs

G&A costs will cover the underground mine, process plant, tailings storage facility and all other Project infrastructure. G&A costs are fixed at US$4.1M/a from 2014 to 2022. The 2013 G&A costs are estimated assuming a transition from owner's costs to mine G&A and on one quarter of plant production. G&A for 2023 is estimated based on reduced staffing as the mine and plant throughput drops to approximately half for the final year of production.

G&A are fixed costs and average US$3.88/t over the life of mine. A breakdown of G&A costs is given in Table 11-6.

Table 21-11: General and Administrative Costs

Area	Unit Cost (US$/t Ore)	Annual Cost (US$/y)
G&A Labour (Management/HR/First-Aid/Accounting/Purchasing/HSE)	1.1	1,230,888
G&A Power (Administration/Medical Assistance/Dining Room/Security Offices)	0.01	15,794
Third Party Services: Catering, Camp Cleaning, Maintenance And Rubbish Removal	1.25	1,350,000
External Assays	0.02	24,000
External Consulting And Software	0.07	80,000
Equipment And Vehicle Rental (Heavy And Light)	0.27	292,000
IT And Communications	0.08	82,500
Safety/Protective Clothing (EPP)	0.06	60,000
Postage, Courier And Light Freight	0.00	2,000
Office/Computer Supplies/Maintenance/Supplies	0.01	10,000
Medical Assistance/First Aid	0.06	64,683
Travel/Accommodation/Camp	0.06	57,000
Access And Internal Roads Maintenance	0.06	60,000
G&A Mobile Fleet (Light Vehicles Maintenance)	0.03	31,250
Insurance/Legal Service	0.22	225,000
Recruitment And Training	0.05	60,000
Environmental Monitoring, Reporting, Programs and Waste Management	0.01	12,000
Site Security (External)	0.44	450,000
Procurement And Importation Costs	0.04	40,000
Sub-total	-	*4,147,116*

21.4 Operating Cost Summary

21.4.1 Operations Staffing List

An operations staffing list has been produced to assist in scaling for camp site and other infrastructure and in capital cost estimation. The staffing list (Table 21-12, Table 21-13) consists of national and local staff. All staff will be national and approximately 60% of staff will be based outside the local area arriving in Juliaca and being bussed to and from the mine at the beginning and end of their rotation. The proportion of local staff will likely vary over the life of the mine with fewer local employees at the beginning of the project, and, as locals are trained and move through the work force the proportion of local workers is expected to increase.

Peak labour requirements are 519 staff with 410 operations staff and an estimated 109 contract staff. AMEC estimates that roughly two-thirds or 346 staff and contractors will be on site at any time.

21.4.2 Consolidated Operating Cost Schedule

A consolidated Unit operating cost schedule has been compiled and is shown in Table 21-14. The average life of mine gold grade is 3.6 g/t Au and average gold recovery is 91.3%. Mine operating costs average US$18.48/t ore including backfill. Plant operating costs total US$24.26/t ore processed including tailings disposal and G&A costs average US$3.87/t. Total site operating costs are US$ 46.35/t ore or US$436/oz of gold.

Table 21-12: Operations Staffing List

Position	Salary	Burden	Designation L/N/E	Maximum LOM Qty.
Site General and Administrative				
General/Operations Manager	45,000	1.52	N	1
Administration Manager	32,004	1.52	N	1
Executive Secretary	21,600	1.52	L	1
Community Liasion	35,659	1.52	N	1
IT Network Technician	31,499	1.52	N	2
Security Officer (Internal Site Security)	9,996	1.52	L	14
Janitor	5,896	1.52	L	1
Human Resources				
Human Resources Generalist	32,004	1.52	N	1
Payroll/Benefits Clerk	21,600	1.52	L	2
Medical Assistance/First-Aid				
Medic	27,883	1.52	N	2
Nurse	9,996	1.52	L	2
Accounting				
Accountant	32,004	1.52	N	2
Accounting Clerk	10,000	1.52	L	1
Purchasing				
Purchasing Agent	27,654	1.52	N	2
Warehouse Supervisor	20,206	1.52	N	2
Warehouse Shipper And Receiver	12,161	1.52	L	2
Health, Safety And Environment (HSE)				
Health And Safety Supervisor/Officer	32,004	1.52	N	2
Environmental Engineer	21,600	1.52	N	2
Underground Mine				
Mine Production Supervisor	29,504	1.52	N	13
Equipment Maintenance Supervisor	36,184	1.52	N	2
Mine Production Supervisor	7,237	1.52	N	74
Ancillary Equipment Supervisor	7,345	1.52	L	140
Production Equipment Maintenance	8,224	1.52	N	28
Ancillary Equipment Maintenance	8,224	1.52	N	11
Technical Services	15,100	1.52	N	21
Administration	26,535	1.52	N	5
Other Labour	5,263	1.52	L	15
Process Plant				
Mill Supervision				
Plant Superintendent	36,000	1.52	N	1
Metallurgist	24,000	1.52	N	1
Plant Operation Shift Supervisor	21,600	1.52	N	3
Plant Operation Chief Supervisor	32,004	1.52	N	1
Plant Maintenance Chief Supervisor	32,004	1.52	N	1
Plant Maintenance Shift Supervisor	21,600	1.52	N	3
Electrical Supervisor/Foreman	24,000	1.52	N	1
Assay Lab Chief (Chemi/Met)	24,000	1.52	N	1
Clerk	4,200	1.52	L	1
Production/Maintenance Planner	21,600	1.52	N	1
Plant Operations				
Crushers And Stockpile Operator	9,996	1.52	N	3
Grinding/GIL Operator	9,996	1.52	N	3
CIL Operator	9,996	1.52	N	3
Stripping/Regeneration Operator	9,996	1.52	N	3
Detox/De-Watering/Reagents Operator	9,996	1.52	N	3
Refining/Smelt Operator	9,996	1.52	N	3
Trainees (Helper)	9,996	1.52	L	3
Laboratory				0
Sample Preparation	9,996	1.52	L	3
Assayer	10,000	1.52	N	3
Trainees	4,800	1.52	L	3
Maintenance				
Electricians	24,000	1.52	N	3
Process Control Technician	10,000	1.52	N	1
Instrument Technician	10,000	1.52	N	3
Millwright	10,000	1.52	N	3
Welders	9,996	1.52	N	3
Pipefitters	9,996	1.52	N	3
Contractor Services				
Exteranal Site Security				
Secondary Security Supervisor	25,920	1.52	N	3
Secondary Security Guard	11,995	1.52	N	15
Tailings				
Haulage	Contract		N	20
Equipment Operator - Placement, Compaction	Contract		N	16
Maintenance	Contract		N	6
Camp Services				
Camp Services Manager	Contract		N	1
Cleaning Supervisor	Contract		N	3
Cleaner	Contract		L	9
Janitor	Contract		L	9
Kitchen Supervisor	Contract		N	3
Cook	Contract		N	6
Cleaner	Contract		L	9
Service Staff	Contract		L	9

Table 21-13: Staffing Summary

Item	Qty.
Total General and Administrative Staff	41
Total Plant Staff	60
Total Mine Staff	309
Total Tailings Staff	42
Total Camp Staff	49
Total Perimeter Security Staff	18
Total Staff	519
Total Operations Staff	410
Total Contract Staff	109
Total Staff	519
Total Staff On Site	346
Total Staff On Rest	173
Total National Staff	295
Total Local Staff	224

Table 21-14: Life of Mine Operating Cost Summary

Consolidated Operating Costs		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	LOM
Mine Production													
Mine Production	kt	57	332	945	1,101	1,098	1,096	1,102	1,095	1,100	1,098	453	9,477
Mine Production Contained Gold	koz	6	39	102	128	132	132	146	135	132	116	44	1,112
Mine Production Grade	g/t	3.2	3.7	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.0	3.6
Stockpile Movement													
Stockpile (Closing)	kt	57	172	17	18	16	12	14	9	10	8	0	
Stockpile Contained Gold	koz	5.9	20.4	1.9	2.0	1.7	1.6	1.8	1.0	1.1	0.8	0.0	
Stockpile Grade	g/t	3.2	3.6	3.6	3.5	3.5	3.8	3.9	3.6	3.6	3.4	0.0	
Commissioning													
Commissioning Feed	kt		41										
Commissioning Contained Gold	koz		4.3										
Commissioning Head Grade	g/t		3.2										
Plant Production													
Plant Feed	kt		176	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	460	9,477
Plant Feed	koz		20	121	127	132	132	145	135	132	117	45	1,112
Plant Feed Head Gold Grade	g/t		3.6	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.1	3.65
Plant Operating Quarters			1	4	4	4	4	4	4	4	4	4	
Plant Avg. Operating Capacity	%		100	100	100	100	100	100	100	100	100	50	
Gold Production	**koz**		22	110	116	121	121	134	124	121	106	41	1,015
Consolildated Project Operating Costs													
Mine Operating Costs													
Mine Operating - Development Direct	US$ M		4.8	9.3	8.5	7.5	5.1	5.6	5.5	3.7	2.2	0.0	147.2
Mine Operating - Production Direct	US$ M		4.3	10.5	11.3	11.0	11.6	10.5	10.2	9.5	10.7	5.3	101.8
Mine Operating - Development Indirect	US$ M		1.0	1.3	1.0	0.9	0.6	0.7	0.7	0.4	0.2	0.0	28.0
Mine Operating - Production Indirect	US$ M		0.4	1.6	2.0	2.1	2.4	2.4	2.4	2.8	2.9	2.3	196.5
Total Mine Operating Costs	US$ M		10.5	22.7	22.7	21.5	19.6	19.2	18.8	16.3	16.0	7.7	175.1
Plant Operating Costs													
Wear Parts	US$ M		0.41	2.54	2.54	2.54	2.54	2.54	2.54	2.54	2.54	1.06	21.8
Reagents (Process Plant)	US$ M		2.20	13.76	13.76	13.76	13.76	13.76	13.76	13.76	13.76	5.76	118.1
Consumables	US$ M		0.03	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.18	0.07	1.5
Power (Average Demand)	US$ M		0.57	2.32	2.32	2.32	2.32	2.32	2.32	2.32	2.32	1.16	20.3
Fuel - Diesel	US$ M		0.04	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.99	0.21	8.2
Elution Circuit Thermal Oil (Heating Oil)	US$ M		0.000	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.002	0.001	0.02
Assays And Quality Control	US$ M		0.01	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.09	0.04	0.7
Others/Miscellaneous	US$ M		0.01	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.02	0.3
Plant Mobile Fleet - Spares And Cons.	US$ M		0.01	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.05	0.02	0.4
Maint Supplies Processing Plant	US$ M		0.10	0.62	0.62	0.62	0.62	0.62	0.62	0.62	0.62	0.26	5.3
Maint. Supplies General	US$ M		0.03	0.17	0.17	0.17	0.17	0.17	0.17	0.17	0.17	0.07	1.5
Plant Manpower	US$ M		0.30	1.24	1.24	1.24	1.24	1.24	1.24	1.24	1.24	0.62	10.8
Contract Tailings Haulage and Placement	US$ M		0.94	4.75	4.75	4.75	4.75	4.75	4.75	4.75	4.75	1.99	40.9
Total Plant Operating Cost	US$ M		4.65	26.75	26.75	26.75	26.75	26.75	26.75	26.75	26.75	11.27	229.9
Unit General and Administrative Cost													
Unit G&A	US$ M		1.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	2.4	36.7
Total Operating Cost	US$ M		16.2	53.6	53.6	52.4	50.5	50.1	49.7	47.2	46.9	21.4	441.7
Unit Operating Costs													
Mine Production Unit Cost	$/t		59.88	20.62	20.66	19.56	17.84	17.49	17.11	14.85	14.53	16.67	18.48
Plant Production Cost	$/t		26.40	24.32	24.32	24.32	24.32	24.32	24.32	24.32	24.32	24.50	24.26
G&A Cost	$/t		6.03	3.77	3.77	3.77	3.77	3.77	3.77	3.77	3.77	5.26	3.87
Total Site Operating Cost	$/t		92.30	48.71	48.75	47.65	45.93	45.58	45.20	42.94	42.62	46.42	46.61
Total Cash Operating Cost	$/oz		724	488	461	434	418	375	401	391	443	525	436

22.0 Economic Analysis

The results of the economic analysis represent forward-looking information 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. Forward-looking information includes Mineral Reserve estimates, commodity prices and exchange rates, the proposed mine production plan, projected recovery rates, exploration incline, decline, and ventilation raise construction costs and schedule, other infrastructure construction costs and schedules, and assumptions that the Project will receive appropriate development and operational permits.

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.

22.1 Valuation Methodology

A financial evaluation of the Project was undertaken using the discounted cash flow analysis approach. The financial model was built based on mine schedule, production schedule, capital and operating cost inputs discussed in Section 16, Section 17, and Section 21.

Cash flows have been projected for the life of mine (LOM), which includes construction, operation and closure phases. The cash inflows are based on projected revenues from gold sales for the LOM. The projected cash outflows, such as capital costs, operating costs and taxes; are subtracted from the cash inflows to estimate the net cash flows (NCF). A financial model was constructed on a quarterly basis to estimate the NCF over the LOM. The NCF are summarized on an annual basis.

22.2 Assumptions

The cash inflows and outflows of the cash flow model are assumed to be in constant 2nd quarter 2011 US dollar basis.

The Project has been evaluated on a project stand-alone, 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 financial results also exclude any expenditure between completion of the PFS and commencement of construction. The analysis is done on a forward-looking basis, with the exception of the sunk costs to date, which are taken into account for tax calculations.

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

The base case gold price used in the financial evaluation was US$1,100/oz. The exchange rate used for financial evaluation was 2.72 (US$ / PEN).

22.2.1 Mineral Resource, Mineral Reserve and Mine Life

Mineral Resources for the Ollachea PFS are discussed in Section 14 of this report and total 10.7 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 9.5 Mt of Probable Reserves at an average grade of 3.6 g/t Au containing 1.1 Moz of gold.

Based on the Mineral Reserves used in the PFS and the mine schedule discussed in Section 16 of this report, the mine will operate over 11 calendar years beginning in 2013 with about two years of pre-production. The mine will begin producing ore from stopes in the second-half of the second year (2014) and will achieve steady state production in the third year of mine production. Mine production will fall to approximately half of steady state production in the eleventh and final year.

Based on this mine schedule, plant production will be over a ten-year period with commissioning in Q3 of 2014, the first year, production ramp-up in Q4 and steady state production of nominally 1.1 Mt/a in years two through nine. The tenth and final year of plant production will be at an average of 41% to match mine production and draw down stockpiles with the plant running with a modified roster schedule to accommodate the lower throughput.

If mining depletion can be offset by the discovery of new Mineral Resources over the life of the project, the mine and plant production lives can be extended.

22.2.2 Metallurgical Recoveries

The average LOM metallurgical gold recovery is 91.3%. Metallurgical recovery is estimated based on a function of tailings residue gold grade with head grade as discussed in Section 13.5:

Metallurgical Recovery (%) = $(Au_H - Au_R)/Au_H * 100$

$$= (Au_H - (0.0209Au_H+0.2401))/Au_H * 100$$

Where Au_H is the head gold grade and Au_R is the tailings residual gold grade.

22.2.3 Smelting and Refining Terms

A refining charge of US$ 2.36/oz of gold has been used in revenue estimation. This refining charge and refining terms are similar to the agreement IRL has for its Corihuarmi gold operation.

22.2.4 Metal Prices

A gold price of US$ 1,100/oz Au is used for the Ollachea PFS base case economic analysis. This price is approximately equal to the consensus guideline cash-flow or short-term gold price of US$ 1,085/oz determined by AMEC in July 2011. The consensus gold price is an average of published gold prices used in mining studies, industry analyst reports, bank reports, and results from mining operations published in Q2 2011.

The financial evaluation was also undertaken using a gold price of US$1,500/oz to show the impact of a higher gold price on Project economics.

22.2.5 Operating Costs

Operating costs used in the Ollachea PFS DCF model consist of estimated mine and plant operating costs including costs for backfill and tailings management, and general and administrative costs for the operation and are discussed in Section 21 of this report. LOM operating costs average US$46.61/t.

22.2.6 Capital Costs

Capital costs used in the Ollachea PFS DCF model include underground mine, plant and surface infrastructure initial capital expenditures, indirect costs, owner's costs, contingency and allowances. A description of these costs can be found in Section 21.

22.2.7 Royalties

The Peruvian government currently levies a sliding-scale royalty on gold sales that ranges between 1% and 3%. A total of US$ 17.7 M in government royalties will be paid over the life of mine averaging approximately 1.6% of revenue from gold sales.

MKK will pay Rio Tinto a 1% vendor royalty on revenue from gold sales. The vendor royalty totals US$11.1 M over the life of mine.

22.2.8 Workers Profit Participation

The Peruvian government currently mandates that 8% of mining profits are re-distributed to workers in the form of workers profit participation. Over the life of mine MKK will pay workers US$31.1 M in profit participation.

22.2.9 Taxes

Credit & debt tax and Income tax and IGV (sales tax) have been applied to the project's taxable financial transactions, income and capital expenditures. IGV is incurred on the initial project capital cost (18%) and is recovered once in production. Once in production, IGV has been excluded from the operating assumptions due to the activity of the Project. Since the Project involves export of goods, IGV is assumed to be immediately recoverable, consistent with Peruvian established practice.

Income tax will be paid at a marginal rate of 30% and totals US$107.4 M over the life of mine.

22.2.10 Closure Costs and Salvage Value

A closure cost of US$3.1 M is estimated for the project. No salvage value has been assessed for mining or processing equipment and infrastructure.

22.2.11 Financing

Costs associated with Project financing have not been considered in the NCF model.

22.2.12 Inflation

There is no provision made for inflation in the NCF model.

22.3 Project Cash Flow

A summary of the Ollachea PFS annual cash flow is presented in Figure 22-1 and Table 22-1.

Figure 22-1: Ollachea Cash Flow Profile

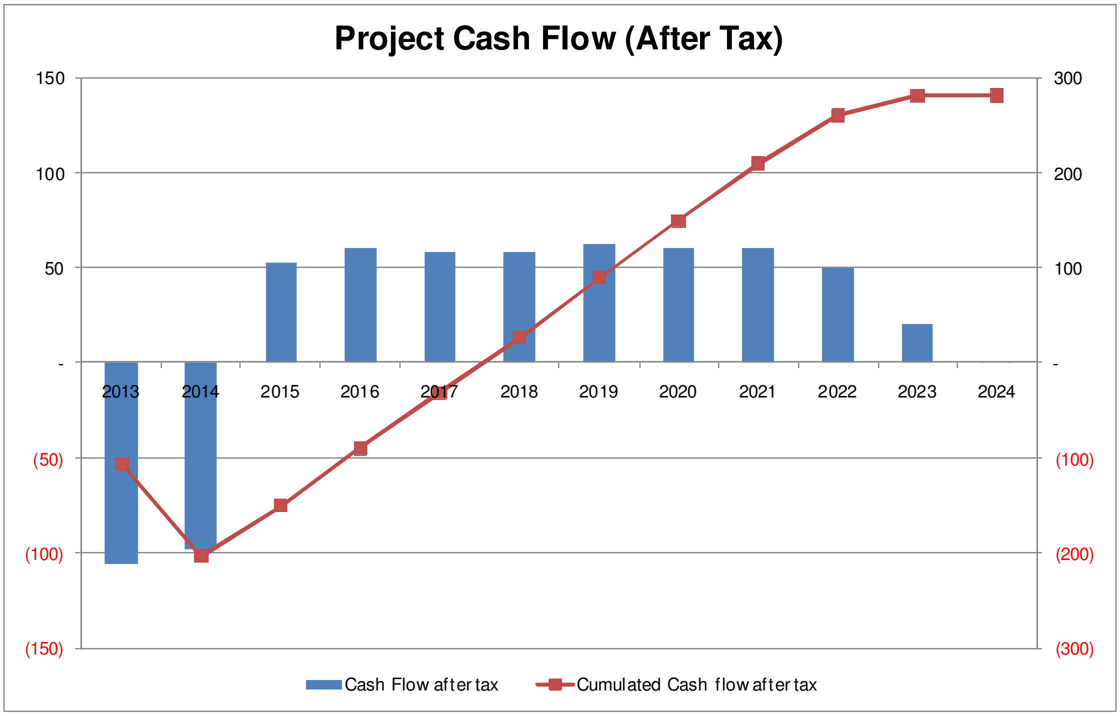


Table 22-1: Ollachea PFS Cash Flow Model Summary

Production		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	Total
Ore Processed	kt	-	217	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	460	-	9,477
Au grade	g/t	-	3.5	3.4	3.6	3.7	3.7	4.1	3.8	3.7	3.3	3.1	-	3.6
Contained Au	koz	-	24.6	120.7	127.4	132.0	132.3	145.3	135.4	132.0	116.7	45.2	-	1,112
Gold Production	koz	-	22.4	109.7	116.2	120.8	121.0	133.8	124.0	120.8	105.8	40.7	-	1,015
Gold Price	US$/oz	-	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	1,100	-	1,100
Cash Flow (2Q 2011 $)														
Revenue	US$M	-	24.5	120.0	127.2	132.1	132.4	146.4	135.7	132.2	115.8	44.4	-	1,110.8
Operating Costs	US$M	-	(16.2)	(53.6)	(53.6)	(52.4)	(50.5)	(50.1)	(49.7)	(47.2)	(46.9)	(21.4)	-	(441.7)
Royalties	US$M	-	(0.5)	(3.0)	(3.3)	(3.5)	(3.5)	(4.1)	(3.6)	(3.5)	(2.9)	(0.9)	-	(28.8)
Capital Costs	US$M	|	(88.7)	(9.2)	(6.0)	(7.0)	(2.3)	(7.9)	(3.5)	(3.3)	(3.8)	0.8	0.6	(219.6)
Workers' Profit Participation	US$M	-	(0.6)	(2.8)	(3.6)	(4.0)	(4.1)	(4.8)	(4.1)	(3.9)	(2.8)	(0.5)	-	(31.1)
IGV	US$M	(16.1)	(14.0)	10.7	12.6	6.9	-	-	-	-	-	-	-	0.0
Income Tax & Other Taxes	US$M	-	(2.1)	(9.6)	(12.7)	(13.8)	(14.1)	(16.7)	(14.2)	(13.7)	(9.6)	(1.8)	-	(108.4)
Total	US$M	(16.1)	(97.6)	52.6	60.6	58.2	57.9	62.7	60.6	60.5	49.8	20.6	0.6	281.1

22.4 Financial Results

Financial results are presented in Table 22-2. This table shows the Project at the base case gold price of US$1,100/oz, and a sensitivity case at a gold price of US$1,500/oz. A summary of the analysis of the life of mine (LOM) average unit cost of production is provided in Table 22-3.

Considering the base case scenario with a gold price of US$1,100/oz, and a discount rate of 7%, the project has a net after tax cumulative tax flow of US$281.1 M, an after tax NPV of US$133 M and an after tax IRR of 20.5% and initial capital will be paid back after approximately four years of operation.

Table 22-2: Summary of Ollachea Financial Results

Parameter	Unit	Base Gold Price US$1,100/oz	Upside Gold Price US$1,500/oz
Net Cash Flow before tax	US$ M	419	808
NPV @ 5% real (before tax)	US$ M	270	561
NPV @ 7% real (before tax)	US$ M	226	486
NPV @ 10% real (before tax)	US$ M	170	393
IRR (before tax)	%	28.1	46.5
Payback (before tax)	Years	3.1	1.9
Net Cash Flow (after tax)	US$ M	280	531
NPV @ 5% real (after tax)	US$ M	167	354
NPV @ 7% real (after tax)	US$ M	133	301
NPV @ 10% real (after tax)	US$ M	91	235
IRR (after tax)	%	20.5	34.1
Payback (after tax)	Years	3.8	2.5

Note:

5. NPVs as at commencement of construction.
6. NPVs are based on mid period discounting.
7. Before tax is before Workers' Participation Profit of 8% and Income Taxes of 30%.
8. Payback starts from the commencement of production.

Table 22-3: LOM Unit Cost of Production per Ounce of Payable Gold

Parameter	Unit	Cost
Mining	US$/oz	173
Processing	US$/oz	226
G&A	US$/oz	37
Total Site Cash Operating Costs	**US$/oz**	**436**
Realization Costs	US$/oz	5
Royalties	US$/oz	28
Total Cash Costs	**US$/oz**	**470**

22.5 Sensitivity Analysis

A sensitivity analysis was performed on the Base Case NPV using a 7% discount rate and IRR (Figure 22-2 and Figure 22-3). 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 had the least impact on the sensitivity of the NPV.

Figure 22-2: NPV @ 7% Sensitivity Analysis

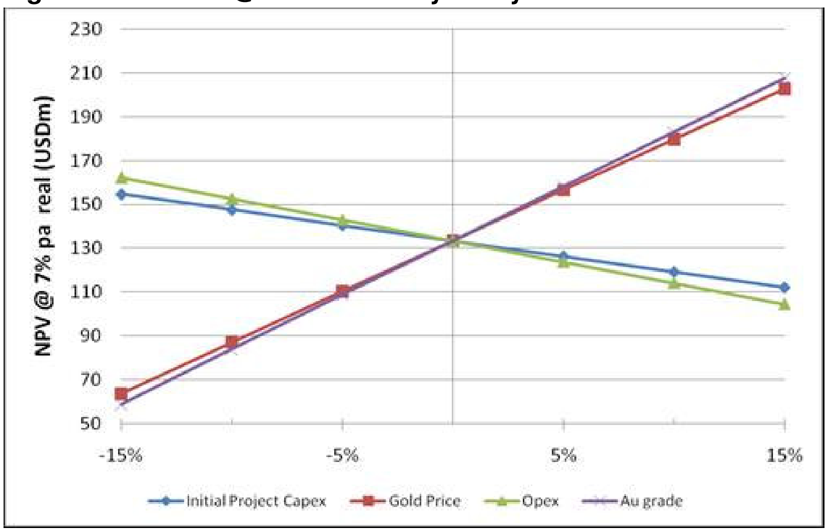
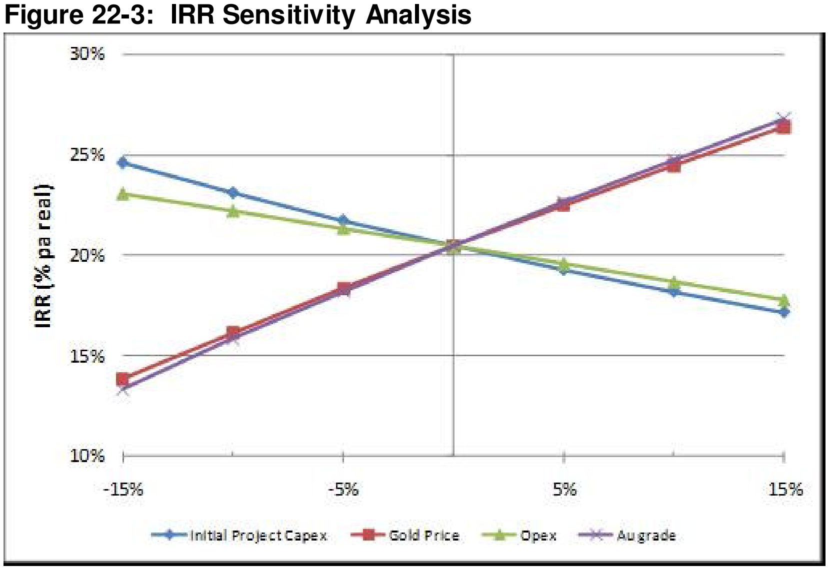


Figure 22-3: IRR Sensitivity Analysis



23.0 Adjacent Properties

There are no properties adjacent to the Ollachea Property at are of relevance to this Report.

24.0 Other Relevant Data and Information

There are no additional relevant data and information on the Project relevant to this Report.

25.0 Interpretation and Conclusions

25.1 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 activitites, and with the necessary upgrades the same means can be reasonably expected to meet needs for Project development and operations.

25.1.1 Geology

The Ollachea deposit is an example of a significant orogenic, lode, or mesothermal gold deposit. The deposit occurs in seven mineralized zones in the Minapampa and Minapampa East Zones, having a strike length of 1,000 m and a width of approximately 200 m, and is hosted by slates with a pyrrhotite, pyrite, arsenopyrite, chalcopyrite and sulphide assemblage.

The current Mineral Resource database for the Ollachea Project consists of of 120 diamond drill holes totalling 46,404 m in length. Samples have been taken at 0.5 m to 5 m lengths and have an average length of 2 m. Samples have been prepared and analysed at CIMM 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 block grades into 2 m x 2 m x 0.4 m sub-blocks from 2 m composites. 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 Property, reported at a 2 g/t Au cut-off grade, consist of 10.7 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 3.0 g/t Au. Mineral Resources were estimated by Doug Corely, MAIG, a Qualified Person under National Instrument 43-101 and have an effective date of 31 May, 2011. Mineral Resources are inclusive of Mineral Reserves.

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

The mining method proposed for the Pre-feasibilityPFS is sub-level open stoping (SLOS) with fill, also referred to as bench stoping with fill when the mining occurs along the strike direction. Stopes will be accessed either longitudinally (along strike) or transversally (perpendicular to strike) dependent on the lode thickness. The longitudinally accessed stopes are planned to be 15 x 30 x 15 m to 30 m (W x H x L). Geotechnical assessment of the rock mass has confirmed that these stope dimensions will remain stable, based on individual and sequential extraction. Transversely-accessed stopes are planned to be 15 x 30 x 45 m and the geotechnical assessment has indicated that these stope dimensions will remain stable.

The mining sequence is essentially bottom up bench and fill, which means that as a maximum the fill will exposed as soon as the adjacent stope is drilled and charged. In this scenario the mine cycle time will control the cure period available for the paste fill to gain adequate strength to allow stable exposure.

Paste backfill will be used as the primary backfill type. Approximately 75% of the mined void will be filled with paste fill. All primary stopes where secondary stopes will be mined will be filled with paste fill. Waste rock will be used for remaining backfill requirements. Waste rock not used as stope fill will be stored in surface waste dumps at the lower portal and temporarily stored at the upper portal before being brought back into the mine to be used as backfill.

Ore will be hauled to the lower portal and dumped into a grizzly and coarse ore bin at the primary crusher, which will be located in a platform outside the mine portal. The platform will be constructed from waste excavated during the construction of the exploration drive prior to project commitment. Ore mined before plant commissioning, and as necessary during the life-of-mine will be stockpiled on the waste platforms near the mine portal.

Ventilation will consist of primary, secondary and auxiliary systems. There are three primary intake airways planned: the haulage drive to the lower portal, the intake ramp at the upper portal, and the fresh air raise. Ventilation will be exhausted from two raises in the central and eastern part of the mine.

A cut-off grade (COG) of 2.0 g/t Au was used in the PFS. Pre-feasibilityConsidering the operating costs, gold recovery gold price and selling costs developed in the Pre-feasibilityPFS, the 2.0 g/t Au cut off is approximately 30% higher than the estimated break-even cut off grade for Ollachea.

The Ollachea PFS Mine development schedule consists of:

- Decline development – 5,166 m
- Level development – 6,548 m
- Vertical development – 1,405 m
- Operating development – 44,224 m

The PFS mine production schedule is based on mining Probable Mineral Reserves thattotal 9.5 Mt with an average grade of 3.6 g/t Au and contain 1.1 Moz of gold.

25.1.3 Metallurgy and Mineral Processing

The interpretation of results from metallurgical testwork carried out in five campaigns has been used to guide process plant design. Test work suggests that crushing and grinding of ore to P80 -75 µm with gravity concentration and intensive leaching of gravity concentration and CIL treatment of the tailings stream can be used achieve gold recovery of an average of 91% from the Ollachea mineralization over the life of mine. Approximately 20% of gold is relatively coarse and can be recovered by gravity concentration while the preg-robbing nature of the mineralization requires the use of the carbon in leach to recover remaining recoverable gold from the gravity tailings. Electrowinning and refining will be used to produce gold doré on site. Tailings will be thickened using a high-rate filter and filtered using press filters. The filter cake will be routed to a paste plant at the plant site to produce paste fill when backfill is required in the underground mine. When backfill is not required, the filter cake will be stacked on a loadout platform outside the plant site for haulage to a dry-stack tailings storage facility.

25.1.4 Infrastructure

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. Auxilliary buildings for administration, mine surface shops, and security facilities will be constructed around the plant site. An operations campsite will be built at the Cuncurchaca site approximately 1,500 m north of the plant site.

A preferred tailings storage facility site has been located and negotiations are underway to secure the surface rights to the site. . AMEC considers there is a reasonable expectation that these surface rights can be obtained. The TSF site is within 10 km of the plant site and can be accessed from the Interoceanic Highway A dry stack tailings facility has been designed for 4.5 Mt of tailings consisting of a toe berm, underdrains, temporary coverage system, and coverage for final TSF closure. The TSF was designed based on preliminary tailings storage requirements provided to AMEC by Coffey Mining in March. Tailings will be hauled over the toe berm and expanding dry stack, dumped and compacted in successive lifts. As the facility expands, drains, and a retaining wall will advance. Contract labour will be used for construction of the TSF and tailings haulage, placement, and compaction.

Approximately 5.9 Mt of tailings will require disposal at the TSF. The PFS design was based on the assumption of 4.5 Mt of tailings disposal capacity; however, preliminary analysis of tailings stability indicates that the TSF may be able to contain up to 6.8 Mt of tailings.

Other infrastructure including an operations camp, surface warehouse, shops and administration buildings have been developed to support cost estimation and development of general arrangements for the Project.

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$18.48/t ore including backfill. Plant operating costs total US$24.26/t ore processed including tailings disposal and G&A costs average US$3.87/t. Total site operating costs are US$46.35/t ore or US$436/oz of gold.

Capital costs include direct and indirect project capital for the mine, process plant and infrastructure. Project direct capital costs total US$113.8 M. The total indirect cost is US$19.6 M and includes indirect mine costs, EPCM, temporary facilities, duties and freight. Owner’s costs incurred in 2013 and 2014 are estimated to total US$7.5 M. A 20% contingency is placed on direct and indirect capital costs for the mine, plant and surface infrastructure. Design growth allowances have been estimated based on estimated costs of earthworks (15%), concrete works (10%), structural steel (5%) and process equipment (2%). The total contingency and design growth allowance for the project is US$28.6 M.

25.1.6 Financial Analysis

A cash flow model incorporating Project and life of mine production, operating costs and capital costs indicates that the Project has an after tax NPV of US$133 M at a discount rate of 7%. 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. Initial capital cost had the least impact on the sensitivity of the NPV.

25.2 Risks and Opportunities

25.2.1 Risks

High Water Inflow and Mine Drainage

Management of water inflow to the mine is a significant risk to the Ollachea Project identified in the Pre-feasibilityPFS. Water inflow rate and quality has major repercussions to:

- Permitting: Changes to the flow rate of the Oscocchoci River and the spring north of Minapampa due to mining will be a potential environmental and social impact of the project.
- Mining: Water management and pumping may be a burden on the operation during peak inflow years and may also have an impact on operating costs and productivity in the mine.
- Process Plant: A water treatment facility to treat mine drainage will be required at the Plant site. The nature of the composition of mine drainage is not well understood at this stage of the Project and the technology required for water treatment will need to be defined during feasibility.

The risk of high water inflows and related environmental and social impacts can be mitigated by taking the following measures during the feasibility work program:

- Provision of a new water supply for the Town of Ollachea westward and up the valley from Minapampa: MKK has already completed engineering on the water supply and it will be implemented before mining is scheduled to commence.
- Lining of the Oscocchoci river bed to limit water inflow into the mine: Design of a lining system has been initiated by MKK.
- Hydrogeological study including installation of additional piezometers, incorporation of data from the exploration tunnel, additional hydrology baseline data, three dimensional structural geology modelling, numerical modelling of flow rates

- Evaluation of a grouting program to reduce water inflow to the mine including field testing in the exploration tunnel
- Optimization of mine design to avoid areas that may be susceptible to high inflows, and in pumping and water management design to ensure that the mine has the capacity to efficiently handle likely peak water flows
- Determination of minimum ecological flow for the Oscocchoci River.
- Determination of potential water inflow composition and mine drainage considering pH, dissolved solids, suspended solids and other parameters necessary for water treatment plant design.

Mining

The main risks associated with mining are:

- No visual distinction between ore and waste; this therefore may impact on stope access development productivity and stope ore loss and dilution.
- Availability of skilled labour to develop and operate the mine. Assumed productivity rates not being met may impact on project development and ramp-up timing.
- Ground conditions in the vicinity of all surface primary ventilation raises. Surface topography limits suitable sites if raises have to be moved.
- Project development and ramp-up timing being delayed because of ground water inflows.
- No consideration has been given to primary mine equipment procurement lead times. This may impact on the mine development schedules and/or change project development strategies i.e. owner mining versus contractor.
- The location (stand-off) of mine access development in relation to the mineralized lodes has been estimated. Mine scale geotechnical numerical modelling analysis requires to be completed to validate assumed development stand-off distances. A negative outcome from the analysis would require access development to be located at a greater distance, which would translate into additional project development cost and time.
- At the time of completion of the mining component of the PFS Coffey Mining had not seen any testwork reports relating to pastefill. It has been reported by AMEC that tailings characterisation testwork has been conducted and that MKK had received a draft report. AMEC indicated that rheology and strength testing of pastefill samples is underway but that results were not available as at the filing date of this Report. The use of paste as a fill material will remain a risk to the Project until testwork confirms its suitability.

Leach Extraction

The use of the proposed flowsheet under the conditions typically experienced in such a circuit have shown repeatable recoveries over 90% along strike and down dip for the various ore lenses. Further work is required to explore additional variability of the ores and to quantify if any significant issues exist with regard to long-term application of the proposed flowsheet.

Pastefill

Pastefill has been selected as the backfill technology for the Ollachea Project. Initial thickening, filtration and tailings characterization work indicate that plant tailings have granulometric, mineralogic and geochemical characteristics that are favourable for the production of filtered tailings and paste backfill. However, rheology, binder and strength test work are not yet complete. The viability of the proposed pastefill system has not been completely demonstrated in the following areas:

- Rheology for pumping requirements.
- Binder content requirement.
- Curing time for stope cycle considerations.
- Strength for mining secondary stopes against pastefill walls.

To mitigate the Project's risk due backfill considerations AMEC recommends:

- Finalizing the current pastefill testwork campaign.
- A trade-off study of paste plant and pumping configurations based on the results of the current pastefill testwork campaign.
- Additional tailings characterization and pastefill testwork based on mineralized composites and using larger volumes of sample to more precisely define strengths and slump rates for the paste.

Schedule

Approval of the Project's EIA is on the critical path of its execution schedule. The schedule assumes 120 days for Ministry consideration of the study, 60 days for MKK to address the Ministry's observations as a result of the study review, and 30 days for the ministry to reconsider the study and approve the Project. Complications in addressing the Ministry's observations or additional rounds of observations may cause a delay in Project approval.

25.2.2 Opportunities

Exploration Potential

There is potential to add additional tonnage to the mine production plan by continuing to explore the Cuncorayoc Zone to the west of the Minapampa Zone. The potential to discover additional tonnage down-dip at depth also exists as well as mineralization to the east of Minapampa East which will be drilled from underground. The deposit is open ended in both directions and down-dip. Significant exploration discoveries have the potential to add considerably to the mine life. If mine scheduling permits, the addition of one or more additional zones may also support plant expansion.

Mining

Mine scale geotechnical numerical modelling analysis may provide a positive outcome that allows mine access development stand-off distance to be reduced. This would translate into reduced project development cost and time.

Pastefill is an expensive backfill system with the system proposed for the Project particularly expensive because of the long pumping distances. There is an opportunity in optimising this pastefill in terms of using rheology modifiers and changing the cement content. It may be cost effective to consider a thickened tailings option to reduce the pumping capacity required. However, this can only be realistically evaluated once the basic tailings characterisation, rheological testing and strength testing is complete.

Mine equipment fleet optimisation may provide capital and operating cost savings.

Detailed stope design and evaluation may provide reductions in access development requirements by increasing level spacing however this has to be balanced with stope dilution control.

The Study has assumed development and stope ground support will be manually installed. Dependent on the availability of skilled labour a more mechanised approach could be considered. The low labour cost would be a primary factor in the trade off.

Gold Price

A gold price of US$1,110/oz has been used for financial modelling for the Pre-feasibility study. On 11 July, 2011, the spot gold price was quoted at US$1,521/oz which is approximately 40% higher than the price used for the study. To take advantage of record-high gold prices in the near term, consideration should be given to advancing the Project in a rapid but orderly fashion so as to maximise potential revenue from higher commodity prices during mine ramp-up and operation.

Design Optimization

During the completion of the Pre-feasibilityPFS MKK identified a number of plant design optimizations that could be undertaken to save on plant capital cost. Future design work should attempt to capitalize on these comments to reduce the project capital cost.

26.0 Recommendations

A feasibility study is recommended for the Ollachea Project. A feasibility study work program would include drilling for exploration, resource modelling, geomechanical, geotechnical investigation, geotechnical site investigations for the plant, tailings disposal and other key infrastructure, hydrogeology work to quantify and characterize mine drainage, mineral processing test work for liquid-solid separation and additional filtration, backfill and tailings characterization.

26.1 Considerations for the Feasibility Study Work Program

26.1.1 Geotechnical

Additional geomechanical study is required to support feasibility-level mine design. In addition to the updated geomechanical database, backfill and hydrogeological data should also be considered in an updated mine design during feasibility. More detailed designs of underground infrastructure including mine service distribution pumping are recommended for FS and will require updated geotechnical and hydrogeological models.

The geotechnical work program should also include site investigation trenching and drilling for all surface installations including the plant site, camp site, waste dumps and tailings disposal facilities.

26.1.2 Hydrogeology

Understanding water inflows to the mine, the maximum acceptable draw-down of the Osccocachi Basin and development of an infill mitigation strategy considering mine design and grouting options is critical for feasibility study. The following work is recommended:

- A field investigation campaign including additional hydrology drill holes and piezometer installation,
- Continued base line data acquisition for hydrology, streamflow, climate and biological data for the Osccocachi Basin
- Acquisition of inflow data during tunnel excavation
- Construction of a three dimensional structural model for use in hydrogeological interpretation
- Detailed numerical hydrogeological modelling of mine inflows

26.1.3 Geology

Additional exploration drilling is recommended to improve the confidence in indicated resources, especially in the parts of the deposit that are early in the production schedule at the east end of the Minapampa Zone and at the Minapampa East Zone. Some drilling is also warranted at the west end of the Minapampa Zone to improve confidence in Inferred Mineral Resources. Based on the new drilling, the Project's resource model should be updated.

Samples for specific gravity determination should also be taken from mineralized zones. A target of 30 to 40 density determinations per zone should met to adequately characterize variability in the mineralized zone.

26.1.4 Mining

The feasibility study work program for mining should consist of the following activities

- Geotechnical data collection and study for stability modelling of the proposed underground mining layout and sequence, and optimisation of design
- Detailed hydrogeological study including the construction of a groundwater flow model is required to characterise the hydrogeological domains and establish likely groundwater levels and inflows during mining, including possible interaction with the stream locations.
- Benchmarking study to establish a good understanding of conditions and standards employed in underground mines located in Peru and internationally
- Mine design studies should include analyses of the most appropriate Project cut-off grades, local underground mine regulations and standards, slot raise development options, development and stope drill and blast patterns, as well as optimization studies including mine access development locations and sizes based on equipment selection and ventilation requirements, and geotechnical constraints
- Mine scheduling including review of sequencing constraints, optimization of stope delineation strategies and incorporation of mine productivity rates
- Assessment of backfill rheology and strength requirements based on the mine design criteria and assessment of backfill reticulation and pumping requirements
- Backfill testing to identify and test local source binders, chemical characterization of process water, and additional slump and rheology testing are recommended.
- Ventilation study to optimize the ventilation network based on the mine development sequence and review contaminant management and emergency response plans

- Review and finalize a mine operating and equipment procurement and maintenance strategy based on availability, productivity and ventilation requirements
- Assement of operations requirements for mine grade control and reconciliation, OH&S standards
- Assessment of mine services tasks including optimization of material handling system, underground controls and communications system specifications for underground power supply and distribution, compressed air, potable water and service water requirements and distribution network, mine dewatering system underground refuelling strategy and explosives storage and handling based on local regulations and mine requirements.
- Detailed assessment of mining costs to optimize capital and operating cost estimates
- Develop a risk management strategy for OH&S and technical risks and impacts and develop mitigation strategies to manage OH&S and technical risks; and
- Create a Project risk register capturing all risks and mitigation strategies.

26.1.5 Metallurgy

As the Ollachea project advances it will be necessary to undertake further testwork with consideration of some of the results already obtained. The following work will be required.

- Additional BWI and RWI determinations are required along with crushing work index and abrasion index determinations.
- Further development of the gravity process is required along with leaching of concentrates.
- Consideration as to if magnetic separation for sulphide recovery has benefit for tailings management.
- Assays for sulphide sulphur and organic carbon for all future ore-grade intersections and development of an extraction model.
- Resin-in-pulp is a flowsheet that could be applied. Further work is warranted in this area and a survey of RIP practices.
- The results of this series of tests suggested use of pre-aeration and kerosene blanking may provide advantages combined with CIL leaching. This process has been selected as the basis of the PFS flowsheet however further development is required, especially with regard to carbon kinetics and loadings in the presence of kerosene.
- Use of high pH reduces NaCN consumption but at a cost of high modifier consumption. Optimisation is warranted with appreciation there will be an increase in cyanide detoxification costs as the pH will have to be reduced prior to

the detox reactor. There is also a potential for increased extractions based on the limited work to date.

- Leach followed by desorption (late addition of carbon).
- Elevated temperature leaching may well be cost effective. This should be investigated further.
- Cyanide detoxification testwork is required to define process variables.
- Carbon kinetic and equilibrium testing is required. This will have to be done as part of a locked cycle leach program to concurrently determine the influence of partially-loaded carbon on leach recovery.

26.1.6 Mineral Process Design

- It has been estimated that over the first years of operation, the emerging acid mine water volume will not be sufficient to meet water plant requirements. River water will need to be used instead and water need to be treated according to use. The treatment process commences with a pump station and pre–settling channel and pond system which allows settlement of some of the suspended solids. This water is then pumped to the make–up water storage tank providing surge capacity prior to domestic water treatment.
- Perform a trade–off study to define the most suitable cyanide detoxification technology for the project and to evaluate cyanide recovery and cyanide management in the processing plant.
- Perform blanking agent screening testwork to further evaluate the most suitable reagent for activated carbon passivation in preg-robbing ores.
- Perform reagent optimization for carbon-in-leach testwork. Perform sulphur speciation to better understand the increased reagent consumption.
- Perform optimization of preliminary mechanical drawings (plot plan and plant layout) in order to define better equipment arrangements and reduce earth movement. In particular, explore a better configuration for filtration and paste plant.
- Further study to determine the most suitable technology for acid mine drainage, and for the domestic and drainage water treatment plant for the project.
- Perform a trade–off study to define the most suitable location of the paste plant (underground or at the surface next to plant).

26.1.7 Infrastructure

Infrastructure specification and design should be carried out for the following facilities:

- Tailings disposal facility and access road
- Camp
- Waste dumps at the main and temporary upper portal

- Magazine, surface mine shop and warehouse
- Security facilities at main gate

Surface waste dumps at both the main and upper portals will need to be designed to match the updated feasibility study mine plan for Ollachea. More detailed geotechnical studies must be performed in order to determine if the areas selected for waste disposal may support the weight, including the extra seven meter ore stockpile lifts on top of the new platform. These studies must also evaluate and recommend a set of slope angles, berms width and a maximum number of lifts, for both waste disposal and ore. Geotechnical studies may also help to determine whether additional lifts can be constructed on top if additional stockpile is required during mine construction and commissioning.

Geotechnical investigation of the tailings site is required for feasibility design. The nature of the surface on which the TSF has been design and the distance of the TSF from the crest of the slope of the talus deposit on which it rests will need to be understood in more detail. Geotechnical investigation including trenching, drilling, topography and slope stability modelling are recommended. Additional test work on filter cake are also recommended to better understand cake density and shear strength for FS dry stack design. Magnetic separation of pyrrhotite from the tailings stream could be investigated to reduce the reactivity of the filtered tailings.

26.1.8 Engineering

Significant additional engineering work including updates of mechanical, CSA and electrical design are recommended for feasibility. Engineering will provide take-offs for the feasibility study capital cost estimate. Technical specifications will need to be developed to allow timely procurement of long lead time items including a ball mill.

A detailed project execution plan will be developed during feasibility level engineering work.

26.2 Feasibility Work Program and Budget

A summary of the recommended work plan is given in Table 26-2.

The recommended work plan for the Feasibility Study begins in August 2011 and includes the following activities:

- Drilling (US$2.4M) to collect data and sample and data for:
  - resource model update
  - geomechanical study

    - hydrogeology
    - geotechnical characterization and condemnation of tailings and plant site locations
    - sample for metallurgy, tailings and backfill test work

- Mineral Processing testwork program including process flowsheet optimization, pastefill, and tailings testwork (US$0.4M).
- Geotechnical, geomechanical and hydrogeological study (US$0.4M)
- An updated Mineral Resource Model incorporating exploration data to improve confidence in Mineral Resources (US$0.1M).
- An updated mine design and mine schedule incorporating new hydrogeological, and geomechanical data and backfill testwork. (US$0.5M).
- Feasibility study including process and infrastructure design, engineering, capital and operating cost estimation and financial analysis incorporating results of the geotechnical, hydrogeological, mine design and mine schedule and metallurgical test work (US$1.5M)
- Field expenses to continue with environmental base line study, property maintenance, field staff and overheads (US$1.0M)

The recommended feasibility work plan will require a budget of approximately US$6.3M.

27.0 References

AMEC, 2011a. Ollachea Project TSF Site Selection Study. Unpublished final report for MKK regarding a tailings storage facility trade-off study dated April, 2011. 25 p.

AMEC, 2011b. Ollachea Project TSF Design Report. Unpublished draft report for MKK regarding PFS tailings storage facility design and cost estimate dated June, 2011. 25 p.

AMEC, 2011c. Interim Report on Tailings Characterization. Unpublished report by AMEC E&I regarding tailings classification work dated July 8th, 2011. 15p.

AMEC, 2011d. Ollachea Pre-feasibility Study Report. Unpublished draft study report by AMEC.

Ammtec, 2011. Metallurgical Testwork conducted upon Composites of Gold Ore Samples from Ollachea Gold Project for Minera IRL SA/Aurifex Pty Ltd. Report No. A12501 Part A and B. Unpublishe metallurgical testwork report prepared for MKK by ALS Ammtec Laboratories in Perth, Western Australia dated June 2011. Two volumes.

Arevalo, M., 2011. Acutal 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.

Box, G.E., and Jenkins,G.M. 1976. Time series analysis: forecasting and control. Holden Day Series

CIM, 2003. Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines. Document prepared by the Canadian Institute of Mining, Metallurgy and Petroleum (CIM). 55 p.

CIM, 2010. CIM Definition Standards for Mineral Resources and Mineral Reserves. Document prepared by the CIM Standing Committee on Reserve Defintions Adopted by the CIM Council on November 27, 2010. 10 p.

Coffey, 2010. Ollachea Gold Project National Instrument 43-101 Technical Report. Unpublished NI 43-101 Technical Report prepared for MKK by Coffey Mining QPs Beau Nichols MAIG, Bernardo Viana MAIG, Jean-Francois St.Onge MAusIMM, and Barry Cloutt MAusIMM, with effective date 6 April, 2010, supporting disclosure of results from the Ollachea Scoping Study. 120 p.

Coffey, 2011a. Ollachea Resource Update, November 2010, National Instrument 43-101 Technical Report. Unpublished Technical Report prepared for MKK by Coffey

Mining QP Doug Corely and MKK VP Exploration Don McIver with effective date 14 January, 2011. 145 p.

Coffey, 2011b. Memorandum on the May, 2011 Ollachea Resource Update – OLMAY11M.dm. Unpublished internal memorandum addressed to MKK regarding the final PFS Mineral Resource Model update. 31 p.

CSA, 2005. Companion Policy 43-101 CP To National Instrument 43-101 Standards of Disclosure For Mineral Projects. Documento sobre las intepretaciones y provisiones del NI 43-101 por el Canadian Securities Administrators. 9 pp.

Davis, S.N., de Wiest R. (Ariel) 1971. “Hidrogeología”

Deloite, 2010. Annual Peru Labour Survey Results. Unpublished report prepared for AMEC by Deloitte.

Dingman, L. S., (Waveland) 2002, “Physical hydrology”, Second edition.

Especialistas Ambientales, 2010, Segunda Modificatoria del Estudio de Impacto Ambiental Semidetallado del Proyecto Exploratorio Ollachea. An unpublished, approved document prepared by the environmental consulting group Especialistas Ambientales S.A.C. for MKK and submitted to the Ministry of Energy and Mines in October 2010 with application for a renewal of permits to support exploration activities for the Project.

Freeze R.A. and Cherry J.A. 1979. “Groundwater”. Hemel Hempstead: Prentice-Hall International. Xvii. 604 p.

Garcia Loyaza, M. R., 2011. Estudio Geotecnia para el Diseño Civil-Estructural de la Planta de Processos e Infrasestructura del Proyecto Ollachea a Nivel de Pre-factibilidad. Unpublished report on plant site geotechnical investigations dated June, 2011. 27 p plus appendicies.

Geoservice Ingeniería S.A.C., 2010. “Estudios técnicos para el desarrollo de ingeniería del túnel de exploración Ollachea”.

Groves, D.I., Goldfarb, R.J., Gebre-Mariam, M., Hagemann, S.G., and Robert, F., 1998. Orogenic gold deposits: a proposed classification in the context of their crustal distribution and relationship to other gold deposit types, Ore Geol. Rev., 13, p 7-27.

Hargreaves, G.H. and Z.A. Samani. 1982. Estimating potential evapotranspiration. J. Irrig. and Drain Engr., ASCE, 108(IR3):223-230.

IRL, 2011. Annual Information Forum for the 12 Months Ended December 31, 2010. Unpublished corporate document dated 11 March, 2011. 140 p.

Kolymbas, D., Wagner, P., (2007). Groundwater Ingress to Tunnels – The Exact Analytical Solution. Tunnelling and Underground Space Technology, 22. Elisiver (23-27).

Mackenzie, P., 2009. AMEC Mining and Metals Types of Economic and Feasibility Study. Definitions and Descriptions of Feasibility Study. Unpublished AMEC report. 19 p.

Mitchell, R.J., Olsen, R.S., and Smith, J.D. 1982. Model Studies on Cemented Tailings used in Mine Backfill. Canadian Geotechnical Journal, Vol.19, pp.14-28.

MKK, 2011a,. "Resultados de laboratorio ECA- 3 Frecuencia Mensual".

MKK, 2011b. "Lectura Manual de niveles de agua subterránea"

OSCB, 2011. National Instrument 43-101; Standards for Disclosure for Mineral Projects. NI 43-101 Standards of Disclosure for Mineral Projects, Form 43-101F1 Technical Report and Related Consequential Amendments published June 24, 2011. Chapter 5, Rules and Policies. 44 p.

Telluris Consulting Ltd. (September 2009) – Structural Field Study of the Ollachea District. Unpublished MKK report. 36p.

Raymer, J.H. 2001. Predicting groundwater inflow into hard-rock tunnels: Estimating the high-end of the permeability distribution. In: Proceedings of the rapid Excavation and Tunnelling Conference. Society for Mining, Metallurgy and Exploration Inc.

Smee and Associates Consulting Ltd (February, 2009) - A Review of the Minera IRL S.A Quality Control Protocol, Core and Blasthole Sampling Protocol, and Two Laboratories, Peru, Unpublished MKK report.

Tong, F., Certain mineral rights and permits held by Minera IRL S.A. and Compañía Minera Kuri Kullu S.A.. Memorandum prepared by Rodrigo, Elias & Medrano Abogados for Collins Stewart Europe Limited, Jennings Capital Inc., National Bank Financial Inc and Timm Miller VP Corporate finance for Minera IRL Ltd. and dated 10 November, 2010. 27 p.

Waltham, T., 2002. 'Foundations of Engineering Geology, 2nd ed. Tony Waltham, 2002).