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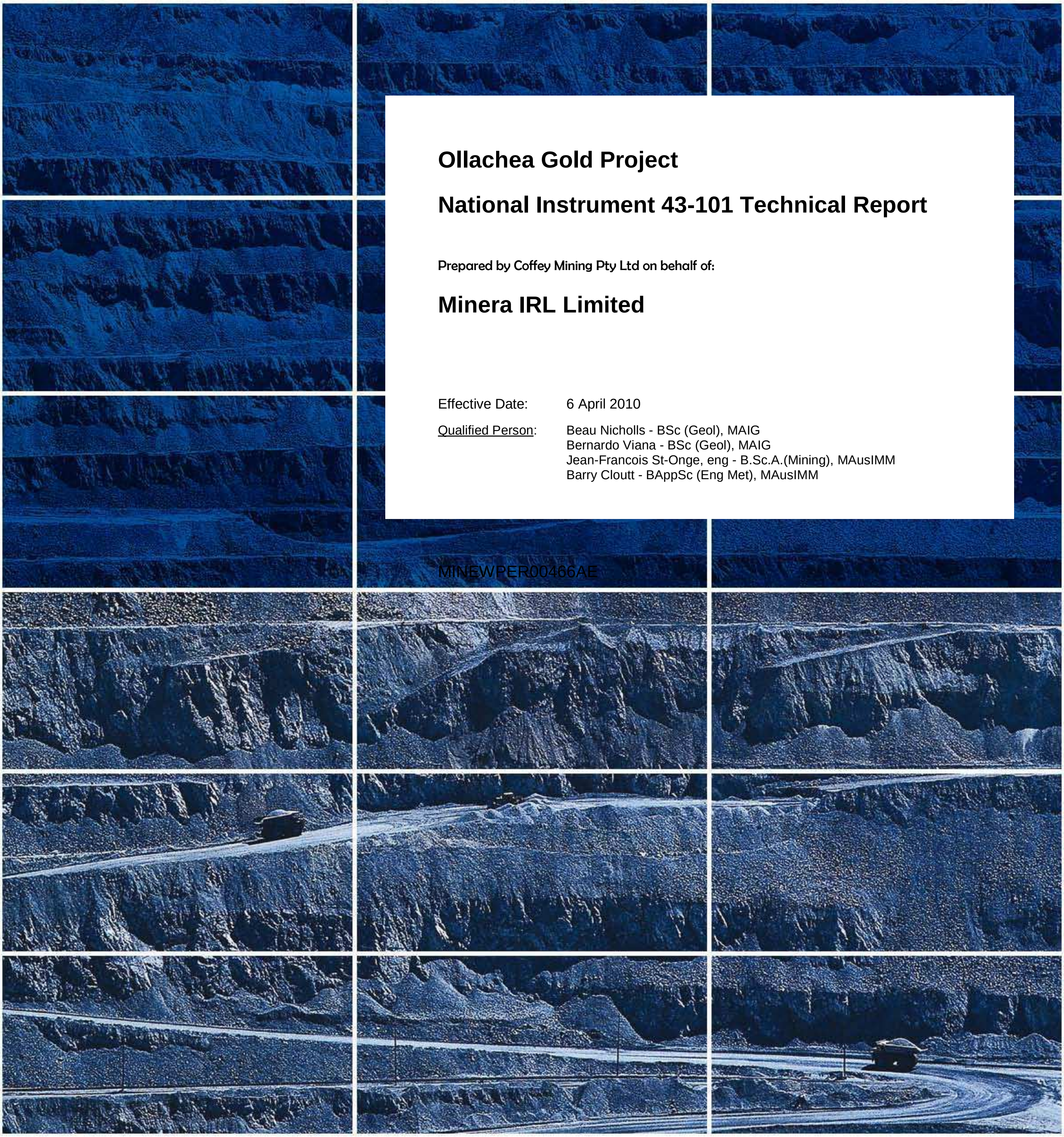

Ollachea Gold Project

National Instrument 43-101 Technical Report

Prepared by Coffey Mining Pty Ltd on behalf of:

Minera IRL Limited

Effective Date:	6 April 2010
Qualified Person:	Beau Nicholls - BSc (Geol), MAIG Bernardo Viana - BSc (Geol), MAIG Jean-Francois St-Onge, eng - B.Sc.A.(Mining), MAusIMM Barry Cloutt - BAppSc (Eng Met), MAusIMM

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DOCUMENT INFORMATION

Author(s):	Beau Nicholls	Geology Manager – Brazil	BSc (Geo) MAIG
	Bernardo Viana	Resource Geologist	B Sc (Geol), MAIG
	Jean-Francois St-Onge, eng	Specialist Mining Engineer	B.Sc.A. (Mining), MAusIMM
	Barry Cloutt	Chief Metallurgist	BAppSc (Eng Met), MAusIMM

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[signed]
Primary Author
Beau Nicholls

[signed]
Supervising Principal
Harry Warries

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1 SUMMARY

1.1 Introduction

Coffey Mining Pty Ltd (Coffey Mining) has been commissioned by Minera Kurri Kullu S.A. (MKK), a wholly owned subsidiary of Minera IRL S.A. (Minera), which in turn is a wholly owned subsidiary of Minera IRL Limited (MIRL), to complete a technical report for the Ollachea Gold Project (the Project) in Peru. Coffey Mining was requested by MKK to prepare the technical report for inclusion in a listing document to be submitted to the Toronto Stock Exchange (TSX).

1.2 Location

The Project is located in the Ollachea District of Carabaya Province in the Puno Region of south-eastern Peru. The Ollachea village, located 1.5km to the east of the Project area, has a population of approximately 2,000. This is the main population base within close proximity to the Project.

1.3 Tenure

The Project comprises 11 tenements, covering an aggregate area of 8,999ha. The mining concessions are in good standing. No litigation or legal issues related to the project are pending.

MKK is 100% owner of the tenements which is subject to a government royalty up to 3% of the sales along with a vendor royalty of 1% net smelter revenue (NSR)

1.4 Geology and Mineralization

The geology of the Ollachea project is dominated by phyllites of the Devonian Sandia Formation, while the central portion is assigned to variably bedded graphitic slates and shales of the Siluro-Devonian Ananean Formation.

The gold mineralization at Ollachea is broadly stratabound within NE to EW trending south dipping carbonaceous phyllites. Gold mineralization is associated with mesothermal quartz-carbonate-sulphide veins, with the sulphide assemblage dominantly comprising pyrite, pyrrhotite and minor chalcopyrite. Arsenopyrite and free gold have also been observed.

1.5 Resources

Coffey Mining has estimated the Inferred Mineral Resource for the Minapampa Zone of the Project as at 6th October 2009. All grade estimation was completed using Ordinary Kriging (‘OK’) for gold. The estimation was constrained within mineralized interpretations that were created with the assistance of MKK geologists.

Seven high grade domains have been interpreted using northsouth oriented, vertical transversal sections based on grade information and detailed geological observations.

The resource estimate for Ollachea has been classified as an NI 43-101 compliant Inferred Mineral Resource, in accordance with the NI 43-101 and the CIM standards, based on the confidence levels of the key criteria that were considered during the resource estimation. Table 1.5_1 below presents the grade tonnage report estimated as of the 6th of October 2009.

Table 1.5_1
Ollachea Gold Project
Grade Tonnage Report – Mineral Resource (as at 6th October 2009)
Ordinary Kriging Estimate
20mE x 30mN x 4mRL Selective Mining Unit

	Lower Cutoff Grade (g/t Au)	Million Tonnes	Average Grade (g/t Au)	Contained Gold (Kozs)
Inferred Mineral Resource	0.0	13.64	3.59	1,574
	0.5	13.62	3.59	1,574
	1.0	13.51	3.62	1,571
	2.0	11.38	3.98	1,456
	2.5	8.91	4.50	1,277
	3.0	6.55	5.06	1,067
	5.0	2.11	7.81	531

An infill drilling programme is underway by MKK to increase the drill density and resource confidence. This second phase of drilling is scheduled to be completed in September 2010. Coffey Mining has reviewed the additional drilling to date and concludes the new holes will increase the confidence in the current interpretation.

There are no mineral reserves which can be disclosed from the Inferred Resources presented in Table 1.5_1. Nonetheless, as part of a preliminary economic assessment, a scoping study (the Study) was completed by Coffey Mining. The Study considered underground mining as the most suitable mining method and the associated mining inventory was estimated to be 8.2Mt at 4g/t head grade for a possible recoverable production of approximately 1.0Moz.

1.6 Geotechnical Review

The weighted Rock Quality Designation (RQD) distribution by the core length indicates that about 25% of the measured core length has RQD value less than 10 percent - a ‘very poor’ quality rock. The low RQD values are related to the intensely foliated and weakly convoluted rock structure.

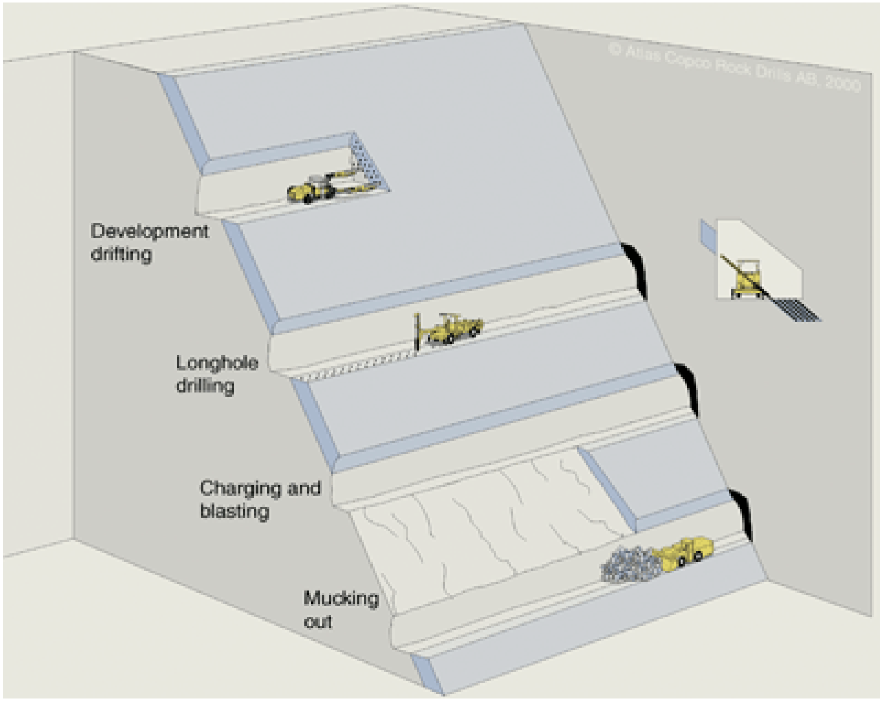
The supported stable span analysis indicates that stopes that are 30m in length and 26m in height along the dip could be considered to be stable subject to the application of cable bolting to the exposures.

1.7 Mining

Underground mining was considered to be more suitable than open cut mining based on the grade tenure and the steep undulating topography. As the basis of the shapes for selecting the specific underground mining method, mineralisation envelopes created at a cutoff grade of 1.0g/t of gold were used.

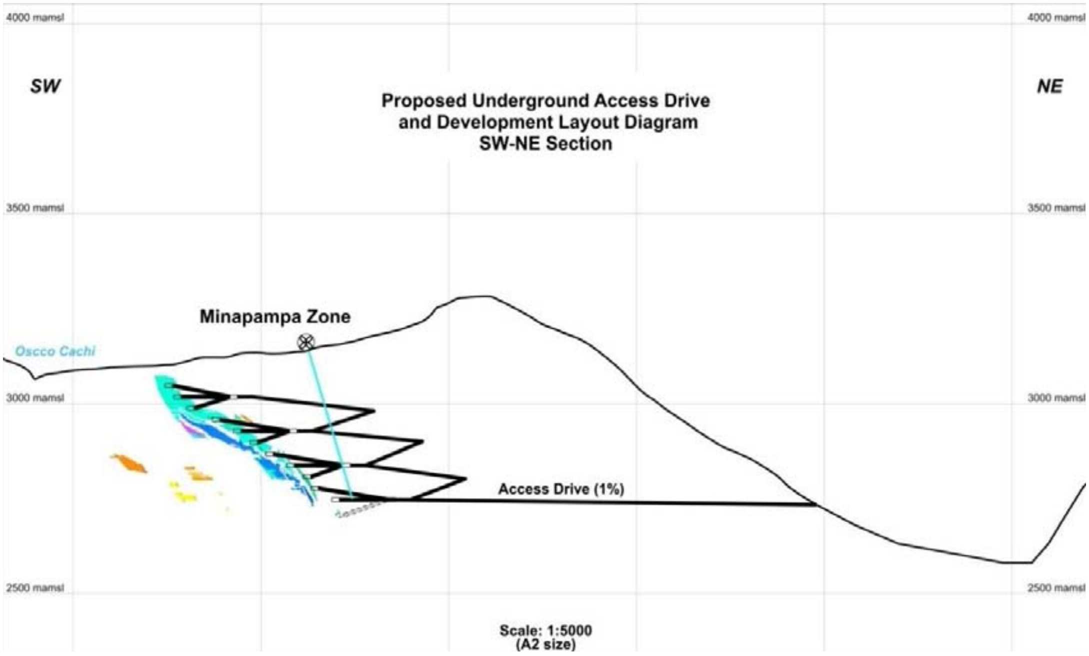
The mining method selected for the current study is sublevel stoping in a narrow vein setting as presented in Figure 1.7_1. The stopes are designed to be mined with longitudinal accesses and do not extend high vertically, with sublevels kept at only 15m distance from floor to floor in the vertical axis; stopes are 30m long in the horizontal axis. The poor rock quality is the current limiting geotechnical factor for stope size.

Figure 1.7_1
Typical View of the Selected Mining Method



The mountainous area of Ollachea provides the opportunity to access the mine by means of an access drive about 1.3km long from the proposed plant site situated in an adjoining valley, through the mountain located towards the north. Figure 1.7_2 presents a sketch of the access drive. It is proposed that this access drive will be developed during the exploration period to serve as an exploration drive which will allow drilling of deep down-dip extensions of the mineralised ore-bodies that are currently not easily accessible from the northern mountain side. The drive will then be converted to a tramming drive for ore production, and transport of personnel and materials.

Figure 1.7_2
Access Drive



Other significant aspects related to mining include:

1.8 Metallurgy

An initial metallurgical testwork program for the Project has been undertaken by Kappes Cassiday and Associates in Reno, Nevada.

Five composite samples were compiled for the testwork program in early 2009 and were considered representative of the mineralisation intersected by the drillholes used at the time. Elemental analysis was presented on one composite which did not indicate any problematic elements other than silver, arsenic and carbon. The silver content was generally one tenth of the gold grade but can be moderately elevated (5.6g/t) which may impact on the CIL and elution operations. The arsenic grade was shown to be ~2,000ppm but was not seen to adversely affect leach recoveries and the total carbon content was ~1.2%. Whilst this is not considered to be abnormally high, there appears to be a strong preg-robbing nature in the mineralised zone which is minimised via CIL processing versus CIP processing. No organic carbon assays were carried out.

Comminution testing indicates that the deposit is amenable to ball milling and that wear rates will not be an issue as the abrasion indices are expected to be medium in nature.

Mineralogical reports indicate the mineralised zone is potentially preg-robbing in nature. The gold is generally fine grained. However, the amount of gravity gold recovered from metallurgical testwork suggests that some coarse gold is present. Testwork showed a moderate gravity gold recoverable content and a gravity gold circuit is recommended to recover this gold.

Testwork using cyanide with the addition of activated carbon in the leach resulted in recoveries ranging from 81% to 95% gold extraction after 36 hours.

1.9 Processing

The chosen base case processing flowsheet consists of three stage crushing followed by a single stage overflow ball mill. The grinding circuit includes one stage of gravity separation followed by intensive leaching of the concentrate. Milled cyclone overflow is treated through a seven stage CIL circuit prior to unthickened tailings being detoxified then filtered via belt filters. Filtered tails is then made available for mine back fill or Dry Stack disposal in a Tailings Storage Facility (TSF). Loaded carbon from the CIL circuit is stripped in an AARL column with barren regenerated carbon being transported back to the tail of the CIL circuit. Pregnant solutions from the AARL and gravity circuits will be electrowon prior to smelting on site to gold doré bars.

Recommended major design criteria are summarised in Table 1.9_1 below:

Table 1.9_1
Ollachea Gold Project
Major Design Criteria

Criteria	Unit	Value
Plant Capacity	Mtpa	1.0
Gold Head Grade	g/t	4.0
Crushing Rate	t/op h	201
Crusher Utilisation	%	68
Milling Rate	t/op h	125
Milling Utilisation	%	91.3
Mill Size	m x m	4.55 x 7.28
Mill Power	kW	2,500
Gravity Gold Recovery	%	20
Leach Time	H	24
CIL Gold Recovery	%	89
Total Gold Recovery	%	91.2
Filtration Capacity	kg/m²/h	420
Elution Size	t per strip	5.5
Strips per Week		12

1.10 Tailings

Dry Stacking appears to be the most appropriate route for tailings disposal as the capital cost is the lowest and best deals with the challenging terrain in the area. While this needs to be confirmed in future studies, this option was adopted as the base case for the purposes of the scoping study.

The design concept is for Dry Stacking of tailings at a site 1.5km north of the plant site and includes an initial starter containment embankment. As the stack is constructed over the life of the mine there will be a requirement for erosion protection of the downstream stack batter and for drainage diversion works to divert runoff upslope, around and downstream of the stack. The landform for the Dry Stack could be potentially terraced to provide useful agricultural land at closure.

1.11 Costs

The estimated capital costs for the Project are summarised in Table 1.11_1. Initial Project capital was estimated at US$157M including a contingency of US$26M. The initial mining capital cost reflects only the first year of waste development and pre-production ore development. In addition to the initial capital investment, a sustaining capital of US$4.0M was included on a yearly basis as well as a US$5.0M closure plan allowance at the end of the mine life. No contingencies were added to the sustaining capital cost and closure cost in the financial model.

Table 1.11_1
Ollachea Gold Project
Capital Cost Summary (2009$)

Project Capital Cost	Amount US$M	Contingency (20%)	Total
Mining	8.0	1.6	9.6
Mining Equipment	41.5	8.3	49.8
Processing Plant	62.4	12.5	74.9
Infrastructure	11.0	2.2	13.2
Tailings	2.0	0.4	2.4
Backfill	5.8	1.2	7.0
Total	**131**	**26**	**157**
Ongoing Capital Cost	**Amount US$M per a**	**Contingency (0%)**	**Total**
Mine Development	1.4		1.4
Mining Equipment	2.6		2.6
Total	**4.0**		**4.0**
Closure Cost	**Amount US$M per a**	**Contingency (0%)**	**Total**
Closure/Rehabilitation Costs	5.0		5.0
Total	**5.0**		**5.0**

The operational costs are divided into fixed and variable costs, and include mining, processing and General and Administration (G&A.) Table 1.11_2 presents a summary of the operational costs.

Table 1.11_2
Ollachea Gold Project
Operational Costs Summary (2009$)

Site Operating Cost	Fixed (US$Mpa)	Variable (US$/t)	Total at Steady State (US$/t)	LOM Average (US$/t)
Mining	2.31	19.77	22.08	22.20
Processing	4.87	14.63	19.50	19.75
G&A	3.87	0.0	3.87	4.07
Total	**11.05**	**34.40**	**45.45**	**46.02**

1.12 Financial Analysis

The following preliminary assessment is preliminary in nature, it includes solely Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary assessment as estimated in the Study will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The input parameters and assumptions for the financial analysis are as follow:

- The mining inventory was estimated to be 8.2Mt at 4g/t head grade for 1.1M contained ounces. The mining and processing rate was set to 1.0Mtpa with a ramp-up period of 70% during the first year. The processing recovery was estimated at 91.2% for the life of mine.
- Base case metal prices used in the model were US$850 per ounce of gold and US$12 per ounce of silver.
- The life of Project unit production cost per ounce are summarised in Table 1.12_1.
- The financial model included Peru Government Royalty, a Vendor Royalty, Income tax and Workers' Profit Participation. The Peruvian Taxation System IGV (sales tax) was excluded due to the current activity of the Project. Being export of goods, IGV is assumed to be immediately recoverable, consistent with Peruvian established practice.

Table 1.12_1
Ollachea Gold Project
Unit Cost of Production per Ounce
(2009$)

Parameter	LOM Average Cost (US$/oz Au)
Mining	190
Processing	169
G&A	35
Total Site Operating Costs	**393**
Refinery Charge	6
Silver credit	(0.4)
Mine Cash Operating Cost	**399**
Royalties	20
Total Production Costs	**419**

The pre-tax (including Workers' Profit Participation) and post-tax results of the financial analysis are summarised in Table 1.12_2 and Figure 1.12_1.

Sensitivity analysis was carried out on gold price and gold head grade, operating cost and capital cost as well as minable tonnes and throughput. The sensitivity analysis showed the following:

- As with most gold projects, revenue is the most sensitive element of this study. The Project return breakeven point of gold price for the NPV @ 8% is a US$710/oz Au, whereas the IRR reaches zero when the price of gold is US$614/oz. Once a steady state operation has been achieved, the Project is cash cost positive above $400 per ounce. Table 1.12_3 illustrates the effect on cash flow, NPV and IRR for a range of gold prices from $700/oz to $1,200/oz.
- The effect of the operating cost on the Project's financial outcomes is the next most important Project driver, after gold price and head grade. Although the capital cost has a significant influence, its impact is less than the operating cost.
- The effect of either minable inventory or processing throughput is less significant.
- Current drilling by MKK outside the limits of the Minapampa mineralised zone has indicated the potential for additional resources. Table 1.12_4 shows the Project returns based on a theoretical additional 2.0Mt at a gold grade of 4.0g/t, containing 257,000 ounces or nearly 25% increase in resource. It must be noted that this material does not exist and only represent an upside scenario.

Table 1.12_2
Ollachea Gold Project
Project IRR, NPV and Payback

Parameter	Pre Tax	Post Tax
LOM Cash flow	US$221.0M	US$147.7M
IRR (real)	22.4%	17.4%
NPV at 7% real	US$113.9M	US$67.3M
NPV at 8% real	US$102.5M	US$58.7M
Payback period from commencement of production	3.7 years	4.0 years

Figure 1.12_1
Project Cash Flow After Tax

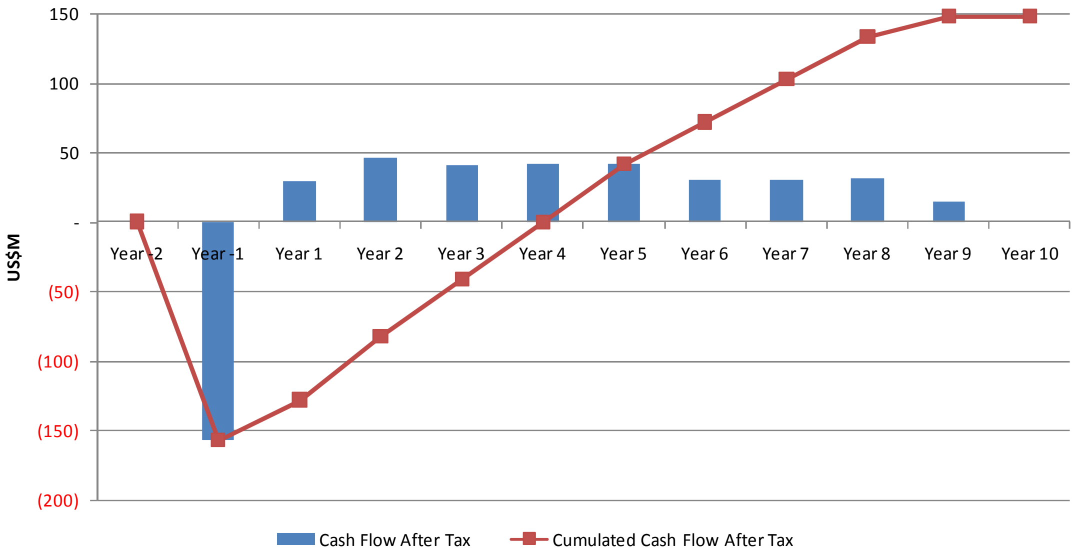


Table 1.12_3 Ollachea Gold Project Gold Price Sensitivity						
Gold Price US$/oz	**Pre-Tax**			**After-Tax**		
	IRR	**NPV @ 8% Real**	**LOM Cash Flow**	**IRR**	**NPV @ 8% Real**	**LOM Cash Flow**
700	9.4%	8.7	81.2	7.3%	-4.0	57.6
800	18.3%	71.2	174.4	14.3%	38.3	117.7
850	**22.4%**	**102.5**	**221.0**	**17.4%**	**58.7**	**147.7**
900	26.2%	133.8	267.7	20.3%	78.9	177.7
1000	33.5%	196.3	360.9	25.8%	119.4	237.7
1100	40.4%	258.5	453.5	31.0%	159.5	297.4
1200	46.9%	320.4	545.8	35.8%	199.4	356.8

Table 1.12_4 Ollachea Gold Project Project IRR and NPV with additional 2Mt at 4.0g/t Au		
Parameter	**Pre Tax**	**Post Tax**
LOM Cash flow	US$322.4M	US$213.0M
IRR (real)	24.8%	19.7%
NPV at 7% real	US$163.8M	US$99.0M
NPV at 8% real	US$147.8M	US$87.4M
Payback period from commencement of production	3.7 years	4.0 years

1.13 Risk Assessment

The current significant risks to the Project are considered to be:

- The Resource risk has the potential to have the greatest effect on the viability of the Project. Although the mineralisation appears to have reasonable continuity, the interpretation of the lenses can affect the dip of the stopes which has an impact on the choice of the mining method. However, the extent of the mineralised zones has yet to be defined and this represents significant upside.
- Geotechnical aspects of the design, in particular the rock mass rating evaluation, are based on limited data. The visit to the underground workings of local artisanal miners tended to present a more positive outlook of the rock mass. However, for the purpose of the study, the geotechnical aspect is conservative.
- The operational risks for underground mining are reduced by the simplicity of the type of operation. The main concern is the geological ability to follow the economically mineralised lenses in the development phase or grade control.
- The Project has moderate cost risk. A 20% increase in operating costs would reduce the Project cash flow by approximately 30%.
- The Project has significant revenue risk. A reduction of revenue by 15% which could be due to either a grade or metal price shortfall indicates over 50% reduction in total Project cash flow.
- Adequate surface area for infrastructure construction and disposal of tailings and waste is critical to the Project due to the topography of the area.

1.14 Recommendations

The following recommendations are made for the next phase of the Project and are discussed in further detail within the body of this report:

- **Studies**
  - As the resource is only of Inferred category, it will need to be brought to a higher level of confidence, i.e. Measured or Indicated, before an Ore Reserve can be reported.
  - It is recommended that a future study optimises the mining method selection with more detailed geotechnical input. Geotechnical considerations will also influence the development cost as ground support is an important part of the cost and the decisive factor for the rate of development.
  - A more thorough study for the tailings storage facility (TSF) including preliminary water balance, hydrogeological, geotechnical and geo-chemical reviews should be undertaken. Closure issues will need to be examined as part of any further studies. This is particularly important as the tailings could be Potentially Acid Forming (PAF).
- **Testwork**
  - Undertake slurry characterisation, waste and tailings testing. Based on the results evaluate the suitability of the tailings for use as pastefill or hydraulic fill.
  - Carry out metallurgical comminution testwork to establish the relationship between grind size, gravity recovery and overall circuit recovery.
  - Determine the amount of gravity recoverable gold so that improved CIL modelling can be carried out.
  - Conduct flotation testwork with and without gravity recovery and regrind to try to maximise gold recovery and minimise capital expenditure.
  - Determine the settling and filtration rate parameters of appropriate slurry streams.
- **Budget and Schedule**
  - MIRL has total budget of $12.3M in 2010 and $10.0M in 2011 excluding vendor payments for the Project. Incorporated in this budget is expenditure on studies of $6.8M in 2010 and $4.8M in 2011, which includes drilling to increase resource confidence, all the required test work and the completion of an access drive. This budget will allow MIRL to complete a Prefeasibility Study in 2010 and finalise a Bankable Feasibility Study by the end of year 2011. Also included in the total budget is expenditure of $2.8M in 2010 and $2.7M in 2011 on exploration and associated drilling. This exploration is well justified considering the exploration potential of the Project. Coffey Mining believes that the level of funding budgeted and schedule proposed by MIRL are appropriate to reach these objectives.

1.15 Authors

Table 1.15_1 summarises the responsibility of each qualified person as authors of this report.

Table 1.15_1
Ollachea Gold Project
Responsibility of Qualified Persons

Qualified Person	Association	Responsible for Sections	Co-Responsible for Sections
Beau Nicholls	MAIG	All sections excluding 16, 17 and 18	1
Jean-Francois St-Onge	Eng., AusIMM	17.2, 18	1
Barry Cloutt	AusIMM	16	1
Bernardo Viana	MAIG	17.1	

2 INTRODUCTION

2.1 Scope of Work

Coffey Mining Pty Ltd (Coffey Mining) has been commissioned by Minera Kurri Kullu S.A. (MKK), a fully owned subsidiary of Minera IRL S.A. (Minera), which in turn is a wholly owned subsidiary of Minera IRL Limited (MIRL) to complete a technical report for the Ollachea Gold Project (the Project) in Peru. Coffey Mining was requested by MKK to prepare the technical report for inclusion in a listing document to be submitted to the Toronto Stock Exchange (TSX).

The Project is a gold project located 1.5km from the village of Ollachea, in the Puno Region of south-eastern Peru.

This report is prepared to comply with reporting requirements set forth in the Canadian National Instrument 43-101 (NI 43-101).

2.2 Qualifications and Experience

Coffey Mining is an international mining consulting firm specialising in the areas of geology, mining and geotechnical engineering, metallurgy, hydrogeology, hydrology, tailings disposal, environmental science and social and physical infrastructure.

The "qualified persons" (as defined in NI 43-101) for the purpose of this report are Mr. Beau Nicholls, Mr. Barry Cloutt, Mr. Jean-Francois St-Onge eng, and Mr. Bernardo Viana, each of whom is an employee of Coffey Mining.

Mr. Nicholls is a professional geologist with 15 years experience in exploration and mining geology. He is Manager of Geology for Coffey Mining's Brazil operations. Mr. Nicholls is also a Member of the Australian Institute of Geosciences (MAIG) and has the appropriate relevant qualifications, experience and independence as defined in the Canadian National Instrument 43-101. Mr Nicholls visited the Ollachea Project between 7th and 10th May 2009.

Mr. Viana is a professional resource geologist with 8 years experience in resource and mining geology. Mr. Viana is a member of the Australian Institute of Geoscientists (MAIG) and has the appropriate relevant qualifications, experience and independence as defined in the Canadian National Instrument 43-101. Mr. Viana has not visited the Ollachea Project.

Mr. Cloutt is a professional metallurgist with 27 years of metallurgical experience. He is Chief Metallurgist for Coffey Mining. Mr. Cloutt is also a Member of the AusIMM and has the appropriate relevant qualifications, experience and independence as defined in the Canadian National Instrument 43-101. Mr Cloutt has not visited the Ollachea Project.

Mr. St-Onge is an engineer non-resident member of the Ordre des Ingénieurs du Québec (OIQ) with 15 years experience in mining engineering experience. He is a Specialist Mining Engineer with Coffey Mining. Mr. St-Onge is also a Member of the AusIMM and has the appropriate relevant qualifications, experience and independence as defined in the Canadian National Instrument 43-101. Mr. St-Onge visited the Ollachea Project between 7th and 10th May 2009.

2.3 Independence

Neither Coffey Mining, nor the authors of this report, have, or have had previously, any material interest in MKK or related entities or interests. Our relationship with MKK is solely one of professional association between client and independent consultant. This report is prepared in return for fees based upon agreed commercial rates and the payment of these fees is in no way contingent on the results of this report.

2.4 Principal Sources of Information

In addition to site visits undertaken to the Ollachea Project between the 7^{th} and 10^{th} May 2009 by Mr. Nicholls and Mr. St-Onge, the authors of this report have relied extensively on information provided by MKK, extensive discussion with management of MKK, and studies completed by other internationally recognised independent consulting and engineering groups. A full listing of the principal sources of information is included in Section 21 of this report and a summary is provided below:

- RSG Global Consulting Pty Ltd (April 2007) – Competent Person's Report.
- Telluris Consulting Ltd. (September 2009) – Structural Field Study of the Ollachea District
- 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

Coffey Mining has made all reasonable enquiries to establish the completeness and authenticity of the information provided and identified, and a final draft of this report was provided to MKK along with a written request to identify any material errors or omissions prior to lodgement.

2.5 Abbreviations

A full listing of abbreviations used in this report is provided in Table 2.5_1 below.

Table 2.5_1
Ollachea Project
List of Abbreviations

	Description		**Description**
$	United States of America dollars	km	kilometres
µ	microns	km²	square kilometres
3D	three dimensional	l/hr/m²	litres per hour per square metre
AAS	atomic absorption spectrometer	M	Million
Au	gold	m	Metres
bcm	bank cubic metres	MIK	Multiple Indicator Kriging
CC	correlation coefficient	ml	Millilitre
cfm	cubic feet per minute	mm	Millimetres
CIC	carbon in column	MMI	mobile metal ion
CIL	carbon-in-leach	Moz	million ounces
cm	centimetre	Mtpa	million tonnes per annum
cusum	cumulative sum of the deviations	MW	Megawatt
CV	coefficient of variation	N (Y)	Northing
DDH	diamond drillhole	NaCN	sodium cyanide
DTM	digital terrain model	NATA	National Association of Testing Authorities
E (X)	easting	NPV	net present value
EDM	electronic distance measuring	NQ_2	size of diamond drill rod/bit/core
EV	expected value	ºC	degrees centigrade
g	gram	OK	Ordinary Kriging
g/m³	grams per cubic metre	oz	troy ounce
g/t	grams per tonne	P80 -75µ	80% passing 75 microns
GW	Gigawatt	PAL	pulverise and leach
GWh/y	Giggawatt hours per year	ppb	parts per billion
HARD	half the absolute relative difference	ppm	parts per million
HDPE	high density poly ethylene	PSI	pounds per square inch
HQ_2	size of diamond drill rod/bit/core	PVC	poly vinyl chloride
h	hours	QC	quality control
HRD	half relative difference	Q-Q	quantile-quantile
ICP-MS	inductivity coupled plasma mass spectroscopy	RAB	rotary air blast
ID	Inverse Distance weighting	RC	reverse circulation
ID²	Inverse Distance Squared	RL (Z)	reduced level
IPS	integrated pressure stripping	ROM	run of mine
IRR	internal rate of return	RQD	rock quality designation
ISO	International Standards Organisation	SD	standard deviation
ITS	Inchcape Testing Services	SGS	Société Générale de Surveillance
ka	thousand years	SMU	simulated mining unit
kg	kilogram	t	tonnes
kg/t	kilogram per tonne	t/m³	tonnes per cubic metre

3 RELIANCE ON OTHER EXPERTS

Neither Coffey Mining nor the authors of this report are qualified to provide comment on legal issues associated with the Ollachea Project included in Section 4 of this report. Assessment of these aspects has relied on information provided by MKK solicitors, Francisco Tong, Estudio Rodrigo, Elías y Medrano Abogados and has not been independently verified by Coffey Mining.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Project is located in the Ollachea District of Carabaya Province in the Puno Region of south-eastern Peru. The Project is cut by the Oscco Cachi River and includes segments of the Joro Piña and Cuchi Puñunan Mountains. The Project is located approximately 160km southeast of Cuzco, 230km north-northwest of Puno and 1.5km west of the village of Ollachea (Figure 4.1_1). Central coordinates are 338,500mE and 8,474,500mN and the project lies between 2,500m and 4,000m elevation.

Figure 4.1_1
Location Plan of Ollachea Project

The boundaries of the concessions have not been surveyed as this is not a requirement of Peru's mining code. The tenement boundaries are defined by UTM coordinates with the datum of PSAD 56, Zone 19S.

4.2 Tenement Status

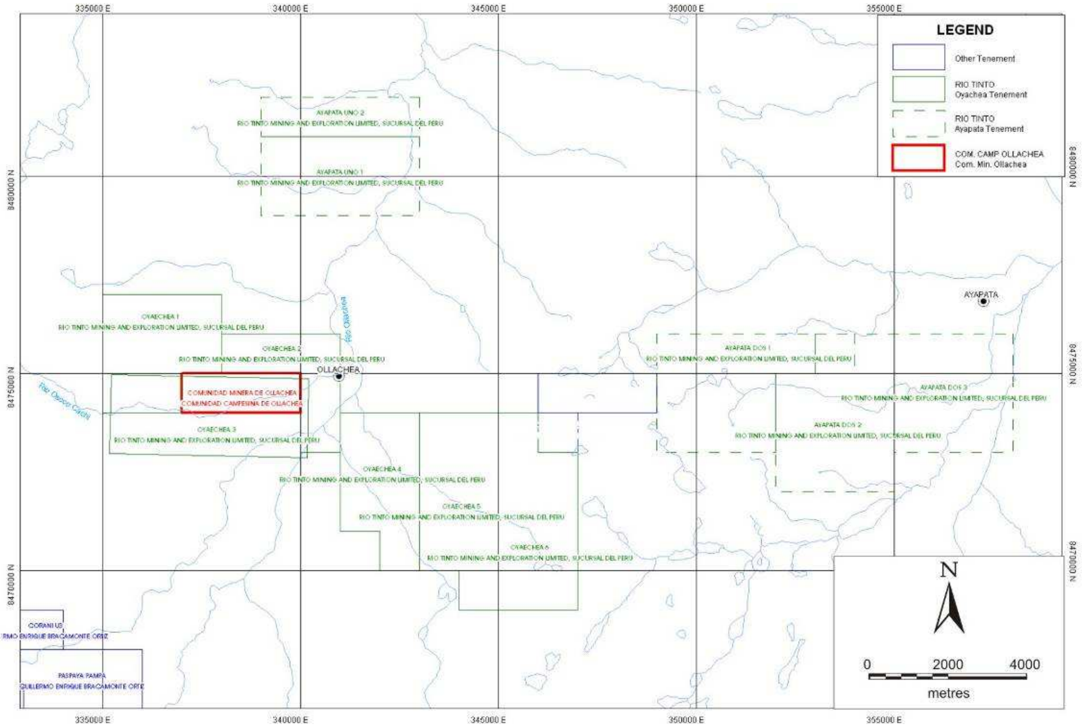
The Ollachea Project comprises 11 tenements, covering an aggregate area of 8,999ha as shown in Table 4.2_1 and Figure 4.2_1 below. MKK is 100% owner of the tenements which are subject to royalties as set forth in Section 4.4.

Table
Project
Table Description

Concession Name	Number	Type	Holder	Area (ha)	Application Date	Expiry Date
Oyachea 1	10215003	Mining Concession	Compania Minera Kuri Kullu SA	800	23/06/2003	See Note 1
Oyachea 2	10215103	Mining Concession	Compania Minera Kuri Kullu SA	500	23/06/2003	See Note 1
Oyachea 3	10218103	Mining Concession	Compania Minera Kuri Kullu SA	998.98	24/06/2003	See Note 1
Oyachea 4	10215203	Mining Concession	Compania Minera Kuri Kullu SA	700	23/06/2003	See Note 1
Oyachea 5	10215303	Mining Concession	Compania Minera Kuri Kullu SA	900	23/06/2003	See Note 1
Oyachea 6	10215403	Mining Concession	Compania Minera Kuri Kullu SA	900	23/06/2003	See Note 1
Ayapata Uno 1	10216403	Mining Concession	Compania Minera Kuri Kullu SA	800	24/06/2003	See Note 1
Ayapata Uno 2	10216503	Mining Concession	Compania Minera Kuri Kullu SA	400	24/06/2003	See Note 1
Ayapata Dos 1	10216603	Mining Concession	Compania Minera Kuri Kullu SA	1,000	24/06/2003	See Note 1
Ayapata Dos 2	10216703	Mining Concession	Compania Minera Kuri Kullu SA	1,000	24/06/2003	See Note 1
Ayapata Dos 3	10216803	Mining Concession	Compania Minera Kuri Kullu SA	1,000	24/06/2003	See Note 1

Note 1: No extinction provision applies to Mining Concessions under Peruvian legislation, as long as its titleholder complies with the administrative obligations established by law in order to maintain its validity.

Figure 4.2_1
Plan of Ollachea Project Tenements



The mining concessions are in good standing. No litigation or legal issues related to the project are pending. Concessions are generally irrevocable but may lapse or terminate in the following two circumstances:

- Failure by a concession holder to pay the mining validity fee (*derecho de vigencia*) for two consecutive years; or
- Failure by a concession holder to pay the penalty (*penalidad*) for two consecutive years, for not achieving exemption from the penalty by meeting investment requirements or for not meeting minimum annual production targets.

4.3 Permits

MKK have provided the permits that are in place for the current exploration phase as shown in Table 4.3_1. No additional permits are required until the project enters a development phase.

4.4 Royalties and Agreements

MKK will be subjected to the following royalties:

a. Peru Government Royalty

The Peru Government Royalty is based on the following:

- Companies with sales of up to the first US$60 million per year – has a royalty of 1% for that portion of sales;
- With the portion above US$60 million of sales from US$60 million to US$120 million per year – the royalty increases to 2% for that portion of sales; and
- Any sales over US$120 million per year – has a royalty of 3% for that portion of sales.

b. Vendor Royalty

A vendor royalty of 1% net smelter revenue (NSR) is included in the Model.

4.5 Environmental Liabilities

Coffey Mining is not aware, nor have we been made aware, of any environmental liabilities associated with the Ollachea Project.

Table 4.3_1
Ollachea Project
Exploration Permits

Date	Permit Type	Group	Report Number	Purpose	Expiry	Comment
27-05-08	R.A Nº 069-2008-DRA-P-ATDRHI	Puno Agricultural Regional Office		Permit for Compañía Kuri Kullu S.A. for the Use of Water from the Water Sources: "Oscco Cachi River" and "Maticuyox Cucho Spring"	27-05-09	UPDATED-
30-09-08	R.D Nº 241-2008-MEM-AAM	MEM	Report Nº 1073-2008-MEM-AAM/AD/WAL	Semi-Detailed Environmental Impact Study of Ollachea Project, Submitted by Minera Kuri Kullu to be Executed in the District of Ollachea, Province of Caravaya, Department of Puno	ND	
22-06-09	R.A Nº 479-2009-ANA/ALA HI.	ANA	Registry Application Nº 189-2009 ALA HI.	Authorizes the Use of Water, in the Process of Regularization, with Mining Exploration Study Purposes Through Diamond Drillings in the Mining Concessions	30-09-09	UPDATED-
11-12-09	R.A Nº 542-2009-ANA/ALA HI.	ANA		Extension of the Water Use Authorization with Mining Exploration Study Purposes Through Diamond Drillings of the Water Resources from the Osjo Cachi River and Maticuyoc Cucho Spring	01-03-10	
26-01-10	Report Nº 444-2010-OTVI	DIGESA	Report Nº 00302-2010/DEPA-APRHI/DIGESA	Favorable Technical Opinion to Grant the Discharge of Industrial Residual Water Authorization	ND	

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Project Access

The village of Ollachea can be reached by vehicle from Juliaca in four hours, via a good quality sealed road, with local zones of unsealed road, associated with the construction of the Southern Interoceanic Highway, (Brazil to Peru). From the Ollachea village, the Project is accessed via a steep gravel road for a further 1.5km to the west. The area is accessible for most of the year; however, access may be occasionally restricted in summer due to snow falls over the intervening high Andean mountain range and landslides that have been known to block the road completely. The Southern Interoceanic Highway (Brazil to Peru), currently under construction, passes through the centre of the Ollachea village. The construction of this road to Ollachea is nearing completion.

5.2 Physiography and Climate

The Project lies within steep sided valleys and ridges ranging in altitude from 2,700m to 3,300m above sea level. The Project is within a sub-alpine climatic regime. Precipitation is markedly seasonal and total annual precipitation averages about 950mm per year. Some 70% to 80% of annual precipitation is received between November and April. Snow is an unusual occurrence at this elevation. The vegetation is dominated by small trees, low shrubs and alpine grasses. A small perennial stream flows east through the property to the Ollachea village.

5.3 Local Infrastructure and Services

The village of Ollachea, is located 1.5km to the east of the Project area and has a population of approximately 2,000. This is the main population base within close proximity to the Project. During the exploration phase, most of the workforce of more than 100 employees is sourced from Ollachea.

The small community of Asiento lies close to and south of the Project area and relies on subsistence cropping. Approximately 200 small-scale miners working the outcrop, have established temporary residence within the currently excised licences immediately north of and adjacent to the farming community. Their main homes are in Ollachea.

The nearest major airport is located at Juliaca, a four hour drive to the south. It is serviced by regular commercial flights from Lima. Road access to the Project is sound and generally well maintained, although local sections are temporarily affected by the construction of the Southern Interoceanic Highway which is nearly completed to the Ollachea village. The San Gaban hydroelectric complex is located 43km north-northeast of the Project. The average capacity of the grid is 455MW, generating some 3,240GWh/y. The San Gaban complex connects directly to the national grid, which passes directly across the Project.

A permanent source of water is available from the Ollachea River, a major melt-water drainage that flows immediately north of the Ollachea village. It is expected to provide an adequate water supply for any future mining and processing activities. In addition, small streams and water bores are located within the Project area, the latter supplying the Ollachea village. Figure 5.3_1 shows the physiography with its limited infrastructure. The Ollachea village is approximately 1.5km from the main mineralized zone.

Figure 5.3_1
3D Raster Image looking North along Ollachea Valley

(Telluris Consulting Ltd, 2009)

6 HISTORY

6.1 Exploration History

The earliest evidence of mining at the Ollachea Project can be attributed to Spanish colonial activity during the 18th century, while subsequent informal mining activity has been actively pursued in the area since at least the 1970's and probably considerably longer.

Modern exploration commenced with Canadian listed company, Peruvian Gold Limited, which completed five diamond drill holes (501m) between 1998 and 1999. Some of the better results published by Peruvian Gold from each hole respectively include 71.05m at 0.47g/t Au, 43.75m at 0.90g/t Au, 129.05m at 0.74g/t Au (including 18m at 2.08g/t), 73.5m at 1.04g/t Au (including 24m at 3.02g/t), and 50.7m at 0.56g/t Au (including 22m at 1.02g/t).

Rio Tinto is understood to have re-discovered the area in May 2003 while following-up a regional stream sediment sampling program. Two field trips were completed in 2003 and 2004, during which period 58 rock chip samples were collected. The results were highly encouraging with 39 samples from a 1km by 1.2km area, coincident with a portion of the CCO Mining Lease, averaging 6.36g/t Au. Some 21 of these samples returned >1g/t, of which 10 returned >5g/t Au.

6.2 Resource History

No historical resource estimates have been released.

6.3 Mining History

Artisanal mining groups have been operating in the region for hundreds of years. No formal production figures are available but based on the Coffey Mining site visit which included an inspection of a number of horizontal drives, approximately 50m into the mineralization, the amount of material removed from the current resource is not considered material. Figure 6.3_1 below shows the current extent of mining.

Figure 6.3_1
Artisanal Mining

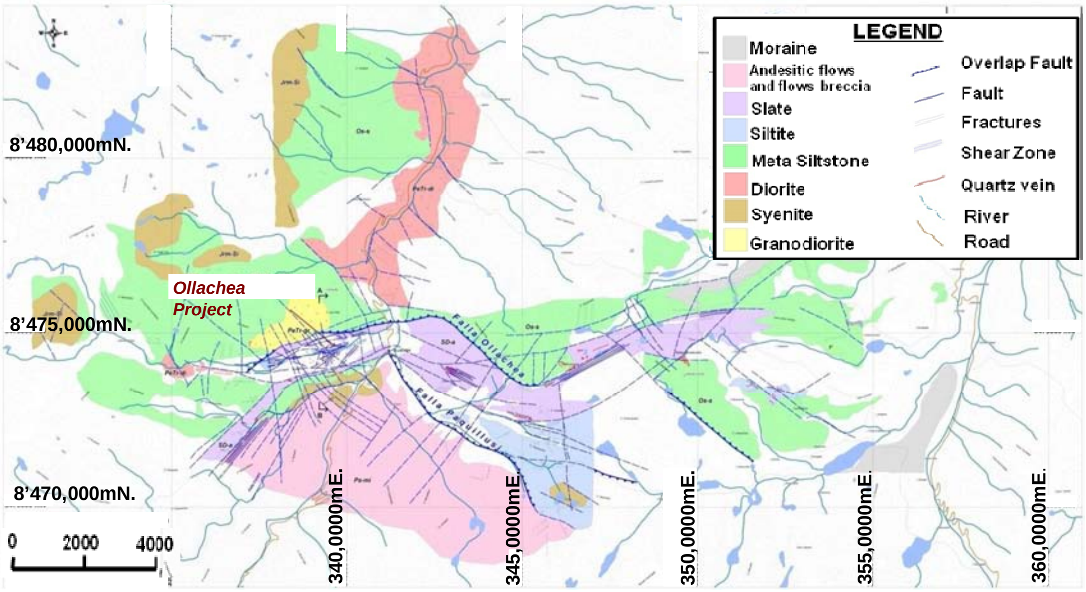
7 GEOLOGICAL SETTING

7.1 Regional Geology

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

On a regional scale, the high grade gold projects occur almost exclusively in slates/phyllites, (usually carbonaceous), and rarely in more arenaceous but only when they lie adjacent to the 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_1 shows the regional setting with respects to the Ollachea project.

Figure 7.1_1
Regional Setting



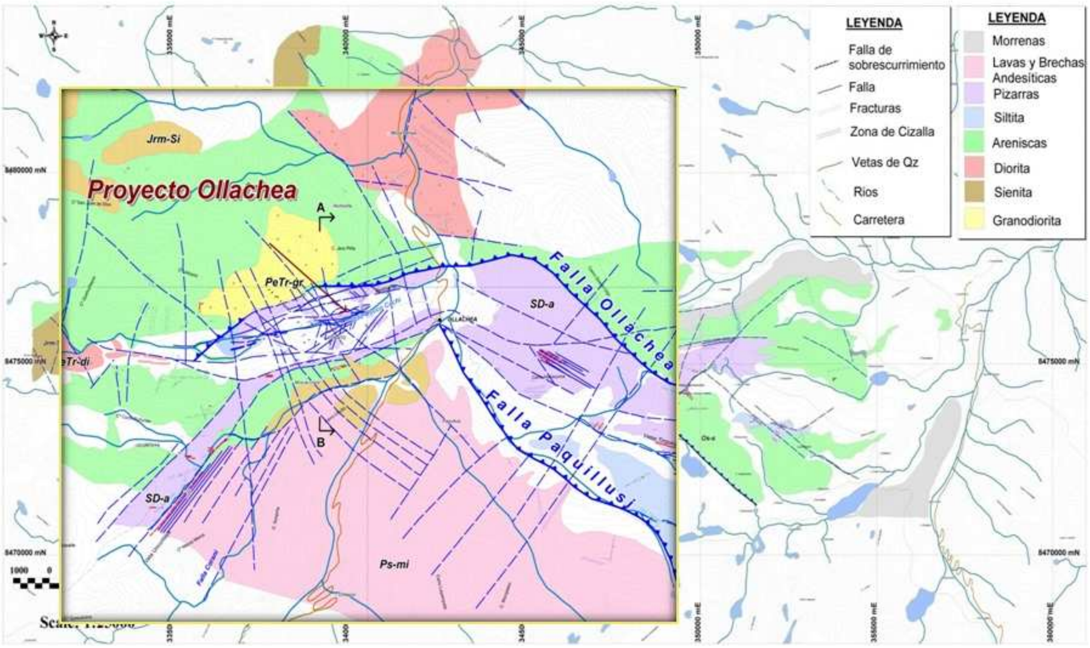
7.2 Project Geology

The geology of the Ollachea project is dominated by phyllites of the Devonian Sandia Formation, while the central portion is assigned to variably bedded graphitic slates and shales of the Siluro-Devonian Ananean Formation. A large nepheline syenite intrusion is located in the southern portion of the project.

The gold mineralization at Ollachea is broadly stratabound within NE to EW trending south dipping carbonaceous phyllites as shown in Figure 7.2_1 below. Two Principal tectonic events are recognised in the Ollachea District:

- D1 – this first event is the deformation of the slate sequence and the thrusting of the Sandia Formation over the Ananea Formation as part of the Hercynic orogenesis.
- D2 – the second phase of deformation is the start of the deformation of the Andean belt (late-Triassic approx. 220 +-10Ma)

Figure 7.2_1
Geology

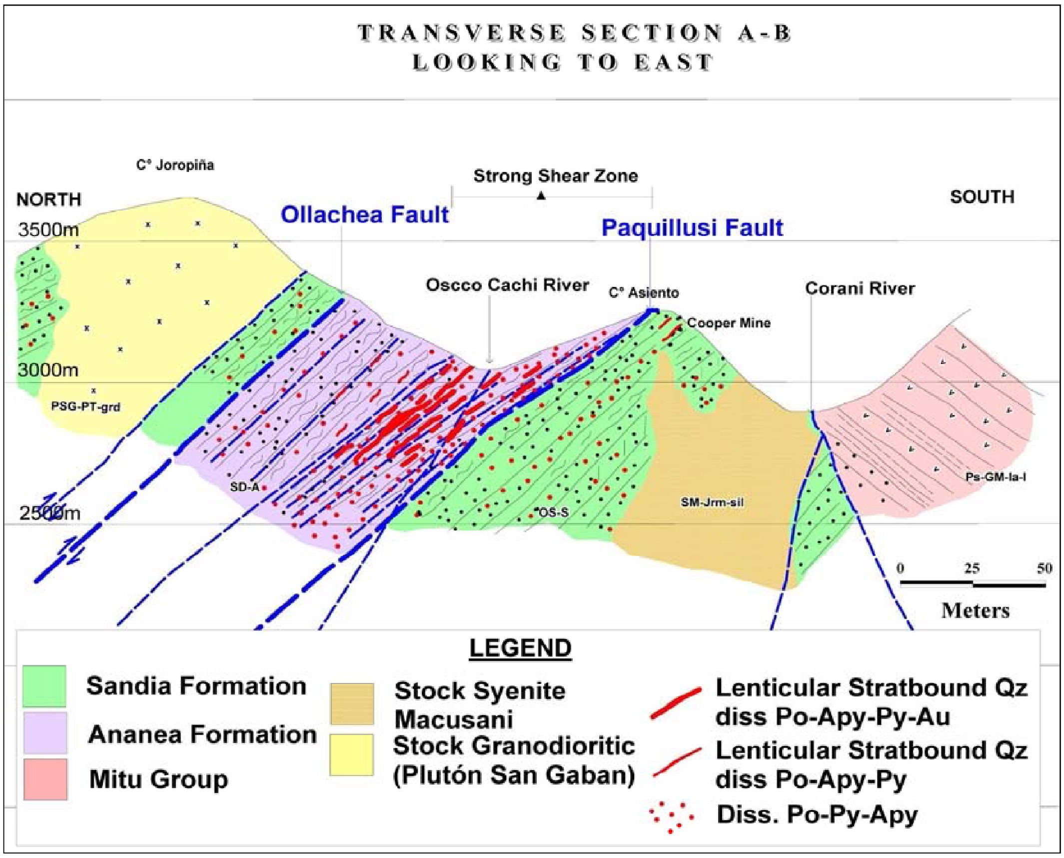


The D1 event was oriented by a NW-SE compression forming zones of shearing, folding and thrusting (inverse faults) of NE-SW strike. Gold mineralization is associated with the first event D1.

The D2 deformation consisted of a prolonged stage of compression oriented NNE-SSW forming principally reverse faults striking WNW-ESE and invoking the folding of the Ollachea District into the form of a "half-dome" thus changing the orientation of the slates in the central area to an almost E-W strike.

Figures 7.2_1 and 7.2_2 show the geology and structure in plan view along with a schematic cross section view of the geology.

Figure 7.2_2
Schematic Cross Section – Looking East y



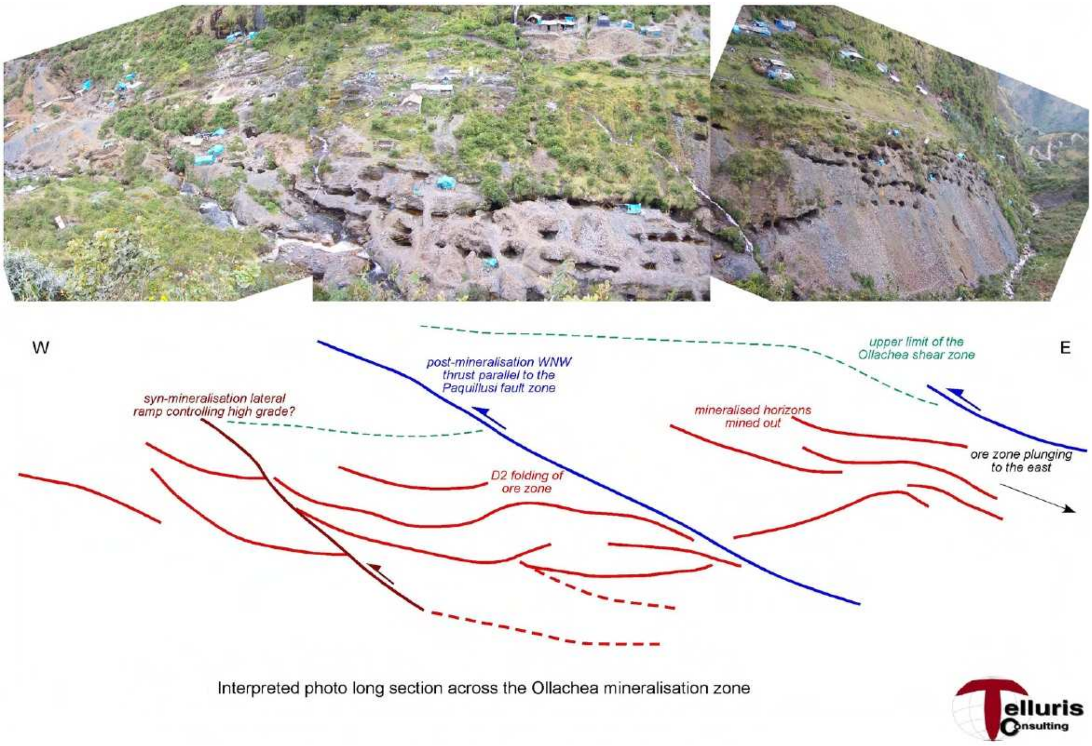
8 DEPOSIT TYPES

Telluris Consulting (Sept 2009) reported that the main stage of gold mineralization at Ollachea is associated with a D1 event comprising of shearing and folding and is largely confined to the weaker carbonaceous shales along a brittle-ductile shear zone. This style of mineralization is similar to an oregenic-style gold deposit but possibly related to late stage dioritic to granodioritic intrusions. 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.

9 MINERALIZATION

The principal zone of mineralization comprising the Ollachea Prospect is being extensively worked by artisanal miners (Figure 9_1). The main mineralized area has a strike length of at least 1km and a minimum aggregate width in the order of 100m. Mineralized vein zones within this envelope average 40m to 60m wide and individually range from a few metres up to 100m in strike length and can be traced down dip over 200m.

Figure 9_1
Principal Mineralized Zone

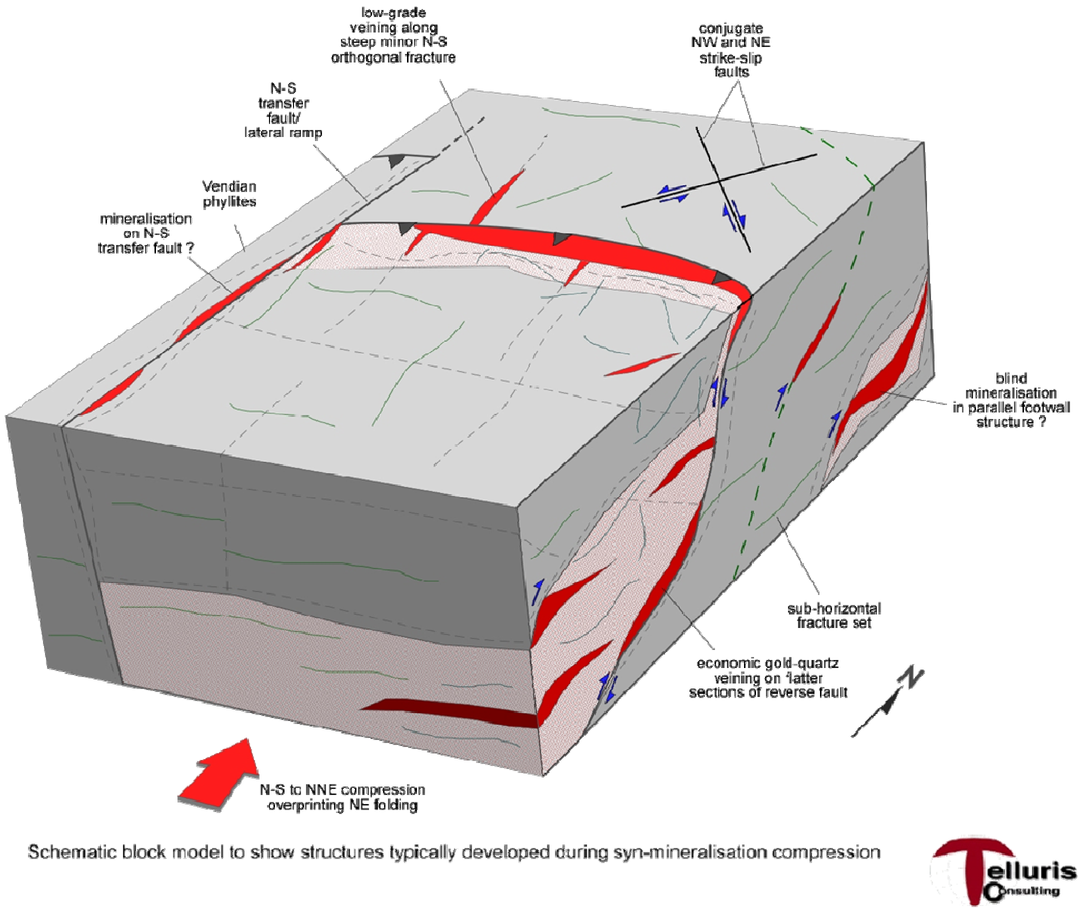


(Telluris Consulting Ltd, 2009)

Gold mineralization is associated with mesothermal quartz-carbonate-sulphide veins, with the sulphide assemblage dominantly comprising pyrite, pyrrhotite and minor chalcopyrite. Arsenopyrite and free gold have also been observed. Vein widths vary from a few centimetres up to a maximum of 40cm but do not always contain gold mineralization.

The mineralized veins are emplaced within an extensive shear zone, which dominates the entire graphitic shale package and is responsible for the well developed slaty cleavage. Mineralized veins have intruded late in the development of the shear zone and are broadly concordant to the cleavage. The veins are strongly boundinaged, resulting in the development of discontinuous lenses of mineralized veins. Figure 9_2 shows a schematic bock model of the mineralization defined at Ollachea.

Figure 9_2
Schematic Bock Model of the Mineralization



(Telluris Consulting Ltd, 2009)

10 EXPLORATION

Bedrock sampling, in conjunction with core drilling has been the dominant exploration tools of MKK for defining mineral resources at the Project. In addition they have utilised geological mapping, and geochemistry sampling, along with an aster and structural geology targeting exercise completed by Telluris Consulting in September 2009.

Most exploration has been focused on the Project. Additional mineral occurrences have been identified in the wider area of the region including the Rinconada project and the Untuca Project but these are early stage reconnaissance exploration targets.

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 (Sections 11 to 14).

Coffey Mining considers the exploration targets justify further exploration and have the potential to significantly add to the resource inventory of the Project. From an economic view the deeper down dip potential of Ollachea may be better targeted from any future underground development as diamond drilling from surface will require >1km holes due to the high topography north of the main mineralisation.

11 DRILLING

11.1 Introduction

The principal methods used for exploration drilling at Ollachea have been diamond core drilling (DDH) by MDH SAC (drilling company), using standard wireline diamond drilling of HQ diameter then reducing to NQ as ground conditions dictate. Core recovery was very good except in large fracture zones.

Table 11.1_1 summarizes pertinent drilling statistics. The central zone has been drilled at a nominal spacing of 60m to 60m.

Table 11.1_1
Ollachea Project
Summary Drilling Statistics

Company/Year	Drillholes	Metres	Contractor	Drill Type	Sample Size
Peruvian Gold Limited (1998 - 1999)	5	501	Unknown	Diamond	HQ, NQ
MKK (2008 – January 2010)	80	30575	MDH SAC	Diamond	HQ, NQ

11.2 Drilling Procedures

11.2.1 Diamond Drilling Procedures

All diamond drilling used in the October 2009 resource estimate was completed by the MKK contractor. Most diamond core holes were drilled using HQ and reducing to NQ diameter.

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.

11.3 Drilling Orientation

Drillholes were generally drilled to the south at between 60 degrees to 70 degrees dip. Holes were targeted to perpendicularly intersect the main trend of mineralization but given the access to deeper sections of mineralisation the intersections are often oblique to mineralization. The deeper sections of Ollachea will need to be targeted from underground or via >1km surface directional drilling The central zone has been drilled at a nominal spacing of 60m to 60m.

11.4 Surveying Procedures

11.4.1 Accuracy of Drillhole Collar Locations

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

Accuracy of the survey measurements meets acceptable industry standards.

11.4.2 Downhole Surveying Procedures

Downhole surveys have been undertaken by the contract driller utilising a Reflex single shot survey tool.

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

12 SAMPLING METHOD AND APPROACH

12.1 Diamond Core Sampling

HQ and NQ diameter diamond core was sampled on an average length of 2m. The core was split using a diamond core saw. Samples were numbered and collected in individual plastic bags with sample tags inserted inside. The chain of custody was noted to be very good with the remaining half core currently stored within refrigerated containers.

Core mark-up and sampling has been conventional and appropriate. Core is not orientated for structural measurements. Coffey Mining recommends orienting core in future.

Coffey Mining had recommended the re-sampling, on 1m intervals, of all the mineralized zones (>0.1g/t Au). This is, however, difficult as initial samples were taken on 2m lengths and therefore pulp material cannot be differentiated. Coffey Mining also recommended during the site visit to undertake all future sampling on 1m intervals due to the poor visual controls on gold mineralization. This had not been implemented in full by MKK to date.

12.2 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 chip logging has been conventional and appropriate.

12.3 Results

The October resource estimate as described in Section 17 reflects drill assay data up to and including hole DDH09-61. Coffey Mining has also reviewed the additional infill and extensional drilling that is currently underway by MKK. This includes drilling and assay results up to and including DDH09-80.

The infill drilling phase has not been completed and as such a new resource estimate is not practical until this phase of drilling is complete. Coffey Mining has reviewed the latest drilling in relation to the current estimate and concludes that the current drilling will effectively allow a more detailed interpretation to be undertaken which will result in an increase in the resource classification confidence.

Significant results from the recent drilling includes hole DDH 09-62 with 2m at 9.98g/t Au from 302m and 2m at 22.04 from 316m ; DDH09-64 with 19m at 2.5g/t Au from 309m DD09-67 with 25m at 2.54g/t Au from 266, DD09-74 with 6m at 3.59 from 250m and 20m at 2.98g/t Au, from 302m, DD09-79 with 10m at 1.91g/t Au from 192m and DD09-90 with 1m at 8.9g/t Au from 34m.

13 SAMPLE PREPARATION, ANALYSES AND SECURITY

13.1 Sample Security

Reference material is retained and stored on site, including half-core and photographs generated by diamond drilling, and duplicate pulps and residues of all submitted samples. All core and pulps are stored at the MKK base in Juliaca City, in refrigerated containers, to preserve the sulphides.

13.2 Sample Preparation and Analysis

13.2.1 CIMM Laboratory

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

- Samples are sorted and dried in an oven
- Samples are crushed by 2 crushers followed by a roll crusher to 2mm. (Smee 2009 identified a potential fatal flaw with the dust extractor potentially taking fines material and biasing the sample).
- The full sample is riffle split to 500g.
- A 500g pulp is prepared in 250g pulveriser bowls to 85% < 75µm (200 mesh). 50g pulps were submitted for chemical analysis.
- Chemical analysis is conducted at the CIMM Lima laboratory and consisted of fire assay (FA) with atomic absorption spectrometry (AAS) finish, using 50g sub-samples. A 32 element suite was also analysed by ICP-OES but has been stopped by MKK as no significant values for these elements were returned from this analysis.

Smee (2009) completed an audit of the preparation laboratory and identified the following 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 250g at a time and 500g is pulverized. These pulverisers need replacing.

13.3 Adequacy of Procedures

Coffey Mining agrees with the recommendations made by Smee (2009) but also recommends that a minimum of 1.2kg be pulverised. 1m sample intervals of core have not been fully implemented by MKK as recommended by Coffey Mining.

Coffey Mining has not been able to independently verify that the recommendations by Smee have been implemented at the Juliaca sample preparation laboratory.

14 DATA VERIFICATION

14.1 Introduction

Standards, blanks and pulp duplicates are inserted at approximately 1 in 20 (5%) by MKK.

14.2 MKK Standards

MKK has made eight standards, presented in Table 14.2_1, from Ollachea material which has been certified by Smee. Pulp 8004 has not been certified and completely failed over 119 samples. This highlights the lack of review that was in place at MKK at this time. The remainder of the samples six to eight are new samples that have been implemented since the site visit by Coffey Mining. Following the visit a designated database manager was implemented and as such the quality of standards analysis has improved dramatically as it is now monitored.

Table 14.2_1
Standards Utilised by MKK
Submitted Standards

Standard	Expected Value (EV)	+/-10% (EV)	Failed	No of Analyses	Min. (%)	Max. (%)	Mean (%)	% Within +/- 10 of EV	% RSD (from EV)	% Bias (from EV)
8001 (ppm)	25.36	22.82 to 27.9	**2**	17	21.66	24.85	0.87	88.24	3.63	-5.1
8002 (ppm)	6.99	6.29 to 7.69	**2**	235	1.55	7.66	7.01	0.43	6.14	0.27
8003 (ppm)	1.53	1.38 to 1.68	**20**	243	1.23	1.83	1.5	92.59	5.04	-1.82
8004 (ppb)	19.86	17.87 to 21.85	**ALL**	119						
8006 (ppm)	1.13	1.02 to 1.24	**0**	31	1.04	1.26	1.13	96.77	4.79	-0.4
8007 (ppm)	2.12	1.91 to 2.33	**0**	21	1.91	2.29	2.04	95.24	5.08	-3.61
8008 (ppm)	4.48	4.03 to 4.93	**0**	23	4.26	4.77	4.41	100	2.55	-1.57
8009 (ppm)	9.24	8.32 to 10.16	**0**	19	9.09	9.7	9.31	100	2.12	0.75
Pulp Blank	<0.1		**4**	945						

Coffey Mining considers that the current accuracy is good as shown by the zero failure rate of the new standards 8006 to 8009, but identified a number of poorly monitored issues from the earlier standards.

14.3 MKK Duplicates

14.3.1 Field Duplicates

A field duplicate is completed every 30 samples by MKK. This field duplicate compares ½ core with ¼ core. Coffey Mining considers this practice flawed in that it requires comparing two different sample sizes.

Coffey Mining compared the ½ core versus the ¼ core using the QC assure software. The precision returned is very poor for all sample data with only 69% passing 30% HARD, as shown in Figure 14.3.1_1. Coffey Mining then compared only the samples less than 1m in length (Figure 14.3.1_2). A minor improvement in precision was realised but the precision was still very poor.

Figure 14.3.1_1
Field Duplicates - All Sample Data (Includes NQ and HQ core 0.3 to 5m)

Summary
(Field Dup ALL)

	Au	Au_DUP	Units			Result
No. Pairs:	508	508			Pearson CC:	0.56
Minimum:	0.01	0.01	g/t		Spearman CC:	0.89
Maximum:	14.33	23.29	g/t		Mean HARD:	24.56
Mean:	0.39	0.44	g/t		Median HARD:	17.05
Median	0.04	0.04	g/t			
Std. Deviation:	1.30	1.87	g/t		Mean HRD:	0.60
Coefficient of Variation:	3.32	4.26			Median HRD	0.00

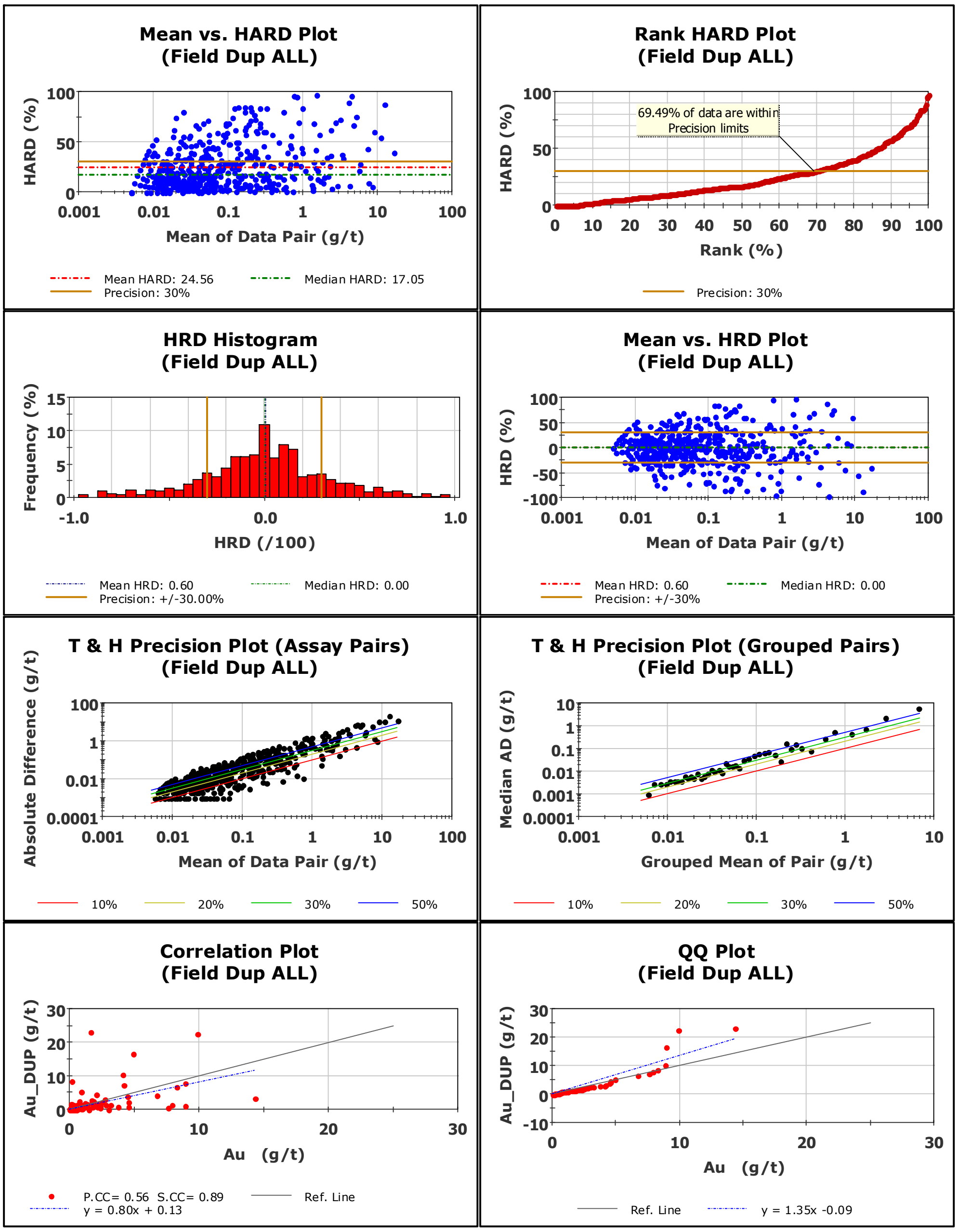


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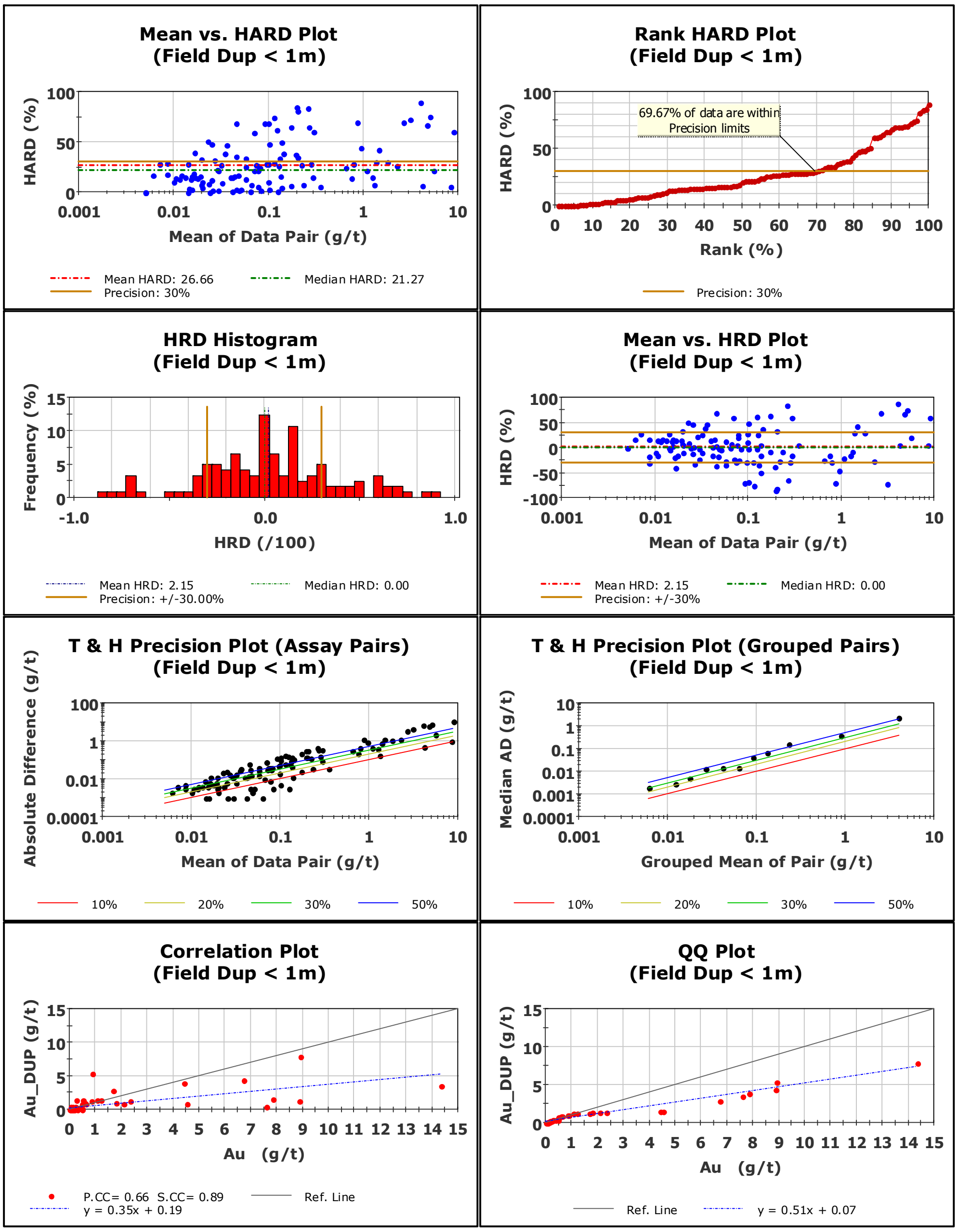
Figure 14.3.1_2
Field Duplicates - All Sample Data <=1m

Summary
(Field Dup < 1m)

	Au	Au_DUP	Units			Result
No. Pairs:	122	122			Pearson CC:	0.66
Minimum:	0.01	0.01	g/t		Spearman CC:	0.89
Maximum:	14.33	7.90	g/t		Mean HARD:	26.66
Mean:	0.69	0.43	g/t		Median HARD:	21.27
Median	0.05	0.06	g/t			
Std. Deviation:	2.09	1.09	g/t		Mean HRD:	2.15
Coefficient of Variation:	3.01	2.54			Median HRD	0.00



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A gross negative bias is noted for higher grade material which suggests that ¼ core is too small a sample to adequately estimate the higher grade, (possibly coarse gold), and will tend to underestimate the gold grade.

Coffey Mining recommends that this ½ core versus ¼ core duplicate be discontinued, as comparing different sample sizes does not produce conclusive results. The ¼ core is not sufficient to represent the ½ core samples.

14.3.2 Preparation Duplicate Sample

After crushing the sample to a -2mm size, the sample is split twice to 500g with the second split representing the preparation duplicate.

Coffey Mining compared the preparation duplicate data (159 samples) using the QC Assure software. The results of this data show, as presented in Figure 14.3.2_1, that the prep-duplicate has over 90% precision at 30% HARD. This is a very acceptable result for this style of Au mineralization.

14.3.3 Pulp Duplicate

During the drilling program, CIMM laboratory 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.

The pulp duplicates returned a very poor precision of 58% at 10% HARD. The results of this data are presented in Figure 14.3.3_1.

The reasoning behind this is unclear as the prep lab duplicates returned very good precision. Smee (2009) suggested that the resubmitted pulps have been contaminated in some way possibly due to humidity and or mixing of pulps.

It is recommended that pulps resubmitted should be sealed in plastic bags then repulverised to homogenise the material and screen/sizing tests completed prior to analysis. Coffey Mining also recommended pulverizing to 200#.

Figure 14.3.2_1
Preparation Laboratory Duplicates – All Data

Summary
(Prep Dup ALL)

	Au	Au_DUP	Units			Result
No. Pairs:	159	159			Pearson CC:	0.93
Minimum:	0.01	0.01	g/t		Spearman CC:	0.98
Maximum:	6.88	5.49	g/t		Mean HARD:	10.58
Mean:	0.34	0.33	g/t		Median HARD:	6.90
Median	0.04	0.04	g/t			
Std. Deviation:	0.93	0.94	g/t		Mean HRD:	0.66
Coefficient of Variation:	2.73	2.81			Median HRD	0.00

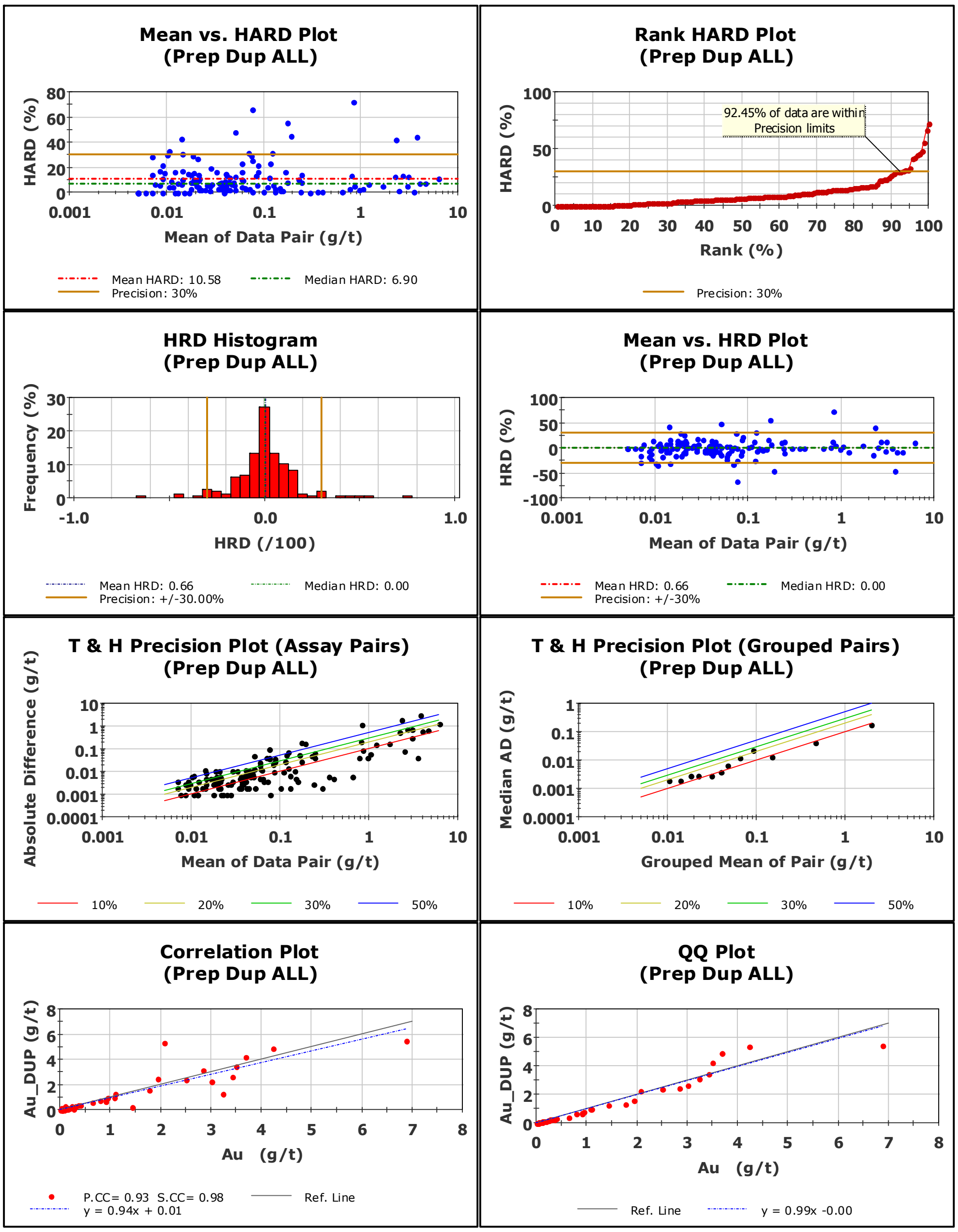

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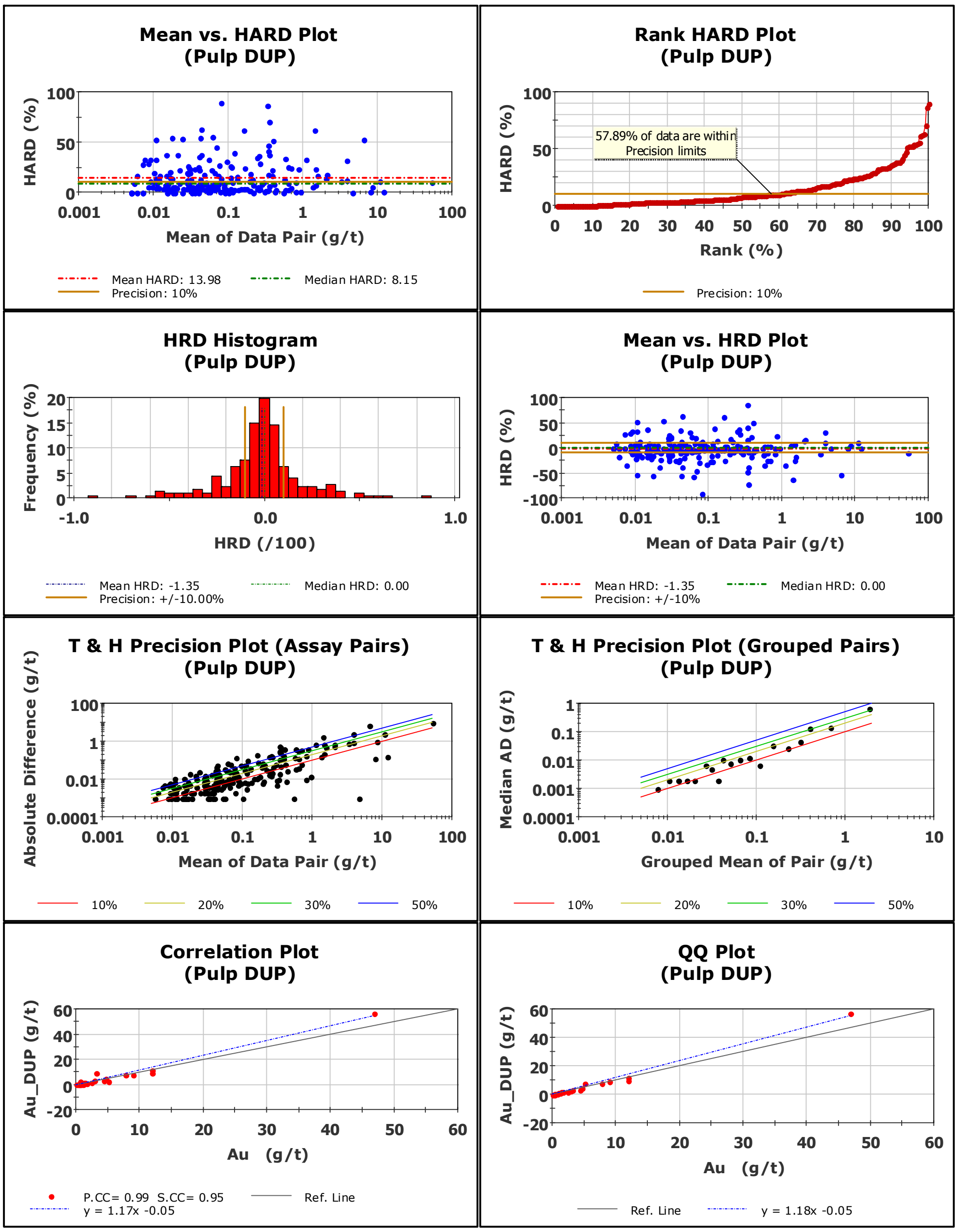
Figure 14.3.3_1
Pulp Duplicates

Summary
(Pulp DUP)

	Au	Au_DUP	Units			Result
No. Pairs:	228	228			Pearson CC:	0.99
Minimum:	0.01	0.01	g/t		Spearman CC:	0.95
Maximum:	46.74	57.19	g/t		Mean HARD:	13.98
Mean:	0.63	0.69	g/t		Median HARD:	8.15
Median	0.05	0.05	g/t			
Std. Deviation:	3.40	4.03	g/t		Mean HRD:	-1.35
Coefficient of Variation:	5.38	5.82			Median HRD	0.00



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14.4 Laboratory Internal Quality Control

14.4.1 Pulp Duplicate Analysis

For every work order, the CIMM laboratory selected five to eleven sample pulps to be re-analysed. Coffey Mining has reviewed the pulp duplicate results as performed by the CIMM laboratory. A total of 722 pulp duplicates have been analysed by CIMM as part of their internal quality control. Results have been observed by Coffey Mining as returning excellent precision.

No CIMM laboratory standards and blank data have been reviewed by Coffey Mining.

14.4.2 Fire Assay versus Screen Fire Assay

MKK completed a total of 119 screen fire assays to compare the CIMM Fire assays against ALS Chemex Screen Fire assays. The results presented in the Figure 14.5_1 show very poor precision between the two, with 49% passing 10% HARD.

The Correlation Plot indicates that the CIMM laboratory has returned a positive bias for gold values over approximately 10g/t Au, suggesting that coarse gold is an issue with the higher values returned. It should be noted that the ALS Chemex samples were not assayed with a gravimetric finish which could also attribute a bias to this dataset.

Coffey Mining would recommend that MKK undertake systematic representative Screen Fire assaying of potentially 10% of the current mineralized dataset (>1g/t Au) be undertaken to develop a larger dataset. This will develop a clearer picture.

14.5 Adequacy of Procedures

Coffey Mining has identified a number of issues with the current quality control data supplied by MKK, with their own standards and pulp duplicates failing poorly, and no systematic re-assay of failed batches being presented to Coffey Mining. It should be noted though, that Smee (2009) identified potential contamination of the pulps, due to balling of the pulps, due to humidity issues which has potentially affected the homogeneity of the pulps and resulted in the poor precision.

The Screen Fire assays by ALS Chemex have suggested coarse gold will be an issue and Coffey Mining has recommended that a larger dataset be attained by carrying out additional screen fire assays.

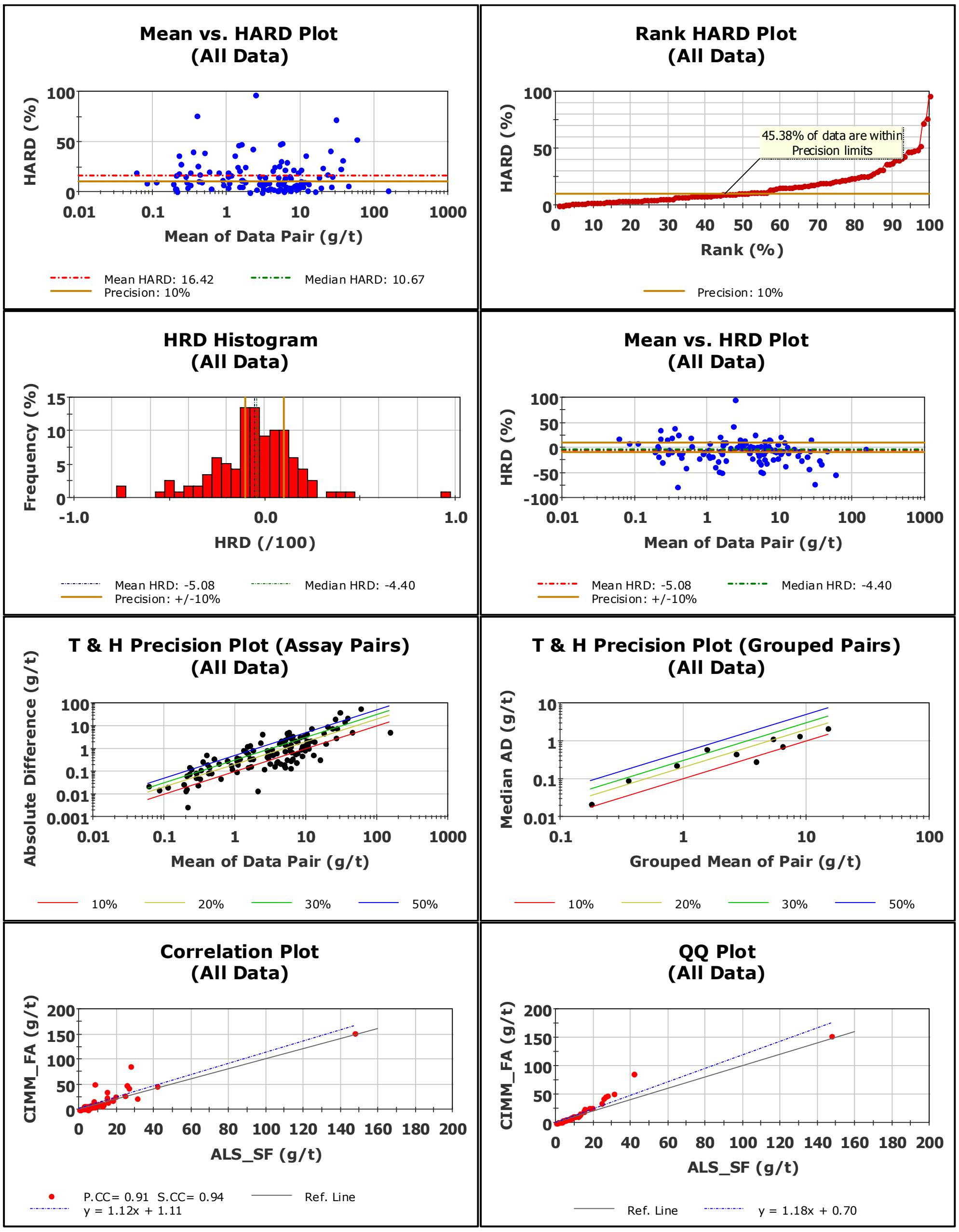
Coffey Mining would recommend that the Juliaca sample preparation laboratory be re-inspected to ensure that the recommendations by Smee have been undertaken.

There are still issues with the precision of the field duplicates. 1m sampling has been recommended to MKK on numerous occasions but has yet to by systematically applied. The gold mineralization is not visually identifiable and the practice of selective sampling is not effective.

Figure 14.5_1
ALS Chemex Screen Fire versus CIMM Fire Assay (pulps)

Summary
(All Data)

	ALS_SF	CIMM_FA	Units			Result
No. Pairs:	119	119			Pearson CC:	0.91
Minimum:	0.07	0.05	g/t		Spearman CC:	0.94
Maximum:	147.50	153.00	g/t		Mean HARD:	16.42
Mean:	6.76	8.69	g/t		Median HARD:	10.67
Median	3.46	3.64	g/t			
Std. Deviation:	14.70	18.08	g/t		Mean HRD:	-5.08
Coefficient of Variation:	2.17	2.08			Median HRD	-4.40



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15 ADJACENT PROPERTIES

There are no advanced gold properties in the immediate vicinity of Ollachea.

16 METALLURGY AND MINERAL PROCESSING

16.1 Metallurgy

16.1.1 Introduction

Scoping level metallurgical testwork has been conducted at Kappes Cassiday and Associates (KCA) in Reno, Nevada USA on samples from Ollachea. This testwork has been used for the preliminary assessment study, to develop design criteria which were used as a basis for the plant design, from which the capital and operating costs were developed. Where metallurgical information was not available, assumptions were made based on MKK's and Coffey Mining's experience and knowledge of operations of a similar size and complexity.

Coffey Mining recommends additional metallurgical testwork to allow further development of the Project.

16.1.2 Mineralogy

No mineralogical examinations were carried out on the composites tested within the metallurgical review of the deposit. Samples from two drillholes, DDH08-01 and DDH08-05 were made available for petrographic and mineragraphic examinations by Cesar Canepa (2009). The two drillholes are located approximately 50m to 150m east of the composited sample drillholes.

The examined samples could be generally described as being schistose in nature, cut by sub-parallel sulphide veinlets. The major observed minerals were quartz, muscovite and graphite (up to 10%). The sulphide mineralization is predominately pyrrhotite in the form of numerous sub-parallel and intertwined narrow seams. Other less abundant sulphide minerals are pyrite, arsenopyrite, sphalerite, and chalcopyrite. Inside some of the pyrrhotites are nests of chalcopyrite, arsenopyrite and sphalerite. Native gold was observed within the pyrrhotite and arsenopyrite in the range of 10 micron to 100 micron in size. Native gold was also observed as being quite abundant in the form of subhedral grains as inclusions inside of pyrrhotite and quartz.

16.1.3 Testwork

Summary

Testwork on five composites from the Ollachea deposit were tested and the following was noted:

- Gold in the samples occurs as fine grained inclusions in pyrrhotite, quartz, arsenopyrite and lesser pyrite and chalcopyrite.
- Gold assays in the 5 samples generally ranged from 2.21g/t to 4.58g/t with one sample (OL26-A) assaying 15.17g/t gold.
- Samples also contained silver from 0.30g/t to 1.24g/t whist samples assayed during leach testwork showed silver assays from 3.8g/t to 5.6g/t.

- Lime additions of approximately 1kg/t were needed to raise the slurry pH to 10.
- NaCN consumption varied from 1.3kg/t to 3.0kg/t.
- Direct cyanide leaching resulted in low gold recoveries of 15% to 79%.
- Cyanide leaching in the presence of activated carbon (CIL), produced gold extractions in the range of 81% to 95%, achieved at a P_{80} of 75µm.
- CIL testwork indicated that 55% more gold is able to be extracted compared to direct leaching (CIP).
- Up to 58% of the gold could be concentrated into 1% of the mass using gravity separation.
- Up to 96% gold recovery can be achieved by froth flotation into 35% of the mass.
- One sighter magnetic separation test was completed that was able to concentrate 50% of the gold, 35% of the silver and 87% of the sulphur into 11% of the mass.

Composite Samples

Five composite samples were prepared by MKK personnel to test some of the Ollachea deposit's various lenses at various depths. When the samples were collated at the beginning of 2009, they were considered to be representative of the mineralized zones of the Ollachea deposit, as they were known at the time. Samples were composited from four drillholes:

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

All diamond core samples were kept frozen to minimise oxidation. All metallurgical composites were packed in drums and purged with nitrogen for shipment to the Kappes Cassiday laboratory to prevent oxidation. Samples were kept frozen at the metallurgical laboratory.

Diamond drill core was targeted for sample selection. Sample selection was conducted jointly by MKK's metallurgical and geological representatives. Drill core sample location was largely spatially based. Assays were used to select the intervals.

Elemental Analysis

Multi-elemental scans were conducted as part of the exploration assays but only one scan, of composite sample – OL04-A, was produced. This scan indicated there were no problematic levels of any elements. It is noted the samples can have up to 0.2% arsenic. The arsenic could be seen as an indication of the minor refractory nature of the ore. The arsenic content in the ore may mean that if the flotation concentrate is not leached on site, then it could have a moderate arsenic level which could attract a smelter penalty. It is also noted that the silver grades can be comparable to the gold and could increase the elution circuit operating costs.

Testwork Program

MKK's testwork program was aimed at supplying basic results that could be used to carry out high level process flowsheet selection, design criteria and mass balancing. As such, a limited amount of work was carried out to indicate whether the ore would be suited to standard processing techniques such as cyanide leaching, gravity separation and flotation. The tests carried out, covered the areas of comminution, gravity concentration, cyanide leaching, magnetic separation and flotation.

Comminution

Only one rod and one ball work index test were carried out during the testwork program. Whilst the work index testing is not totally conclusive, it does indicate that the samples are amenable to ball milling in conjunction with a three stage crushing circuit. The rod:ball Bond work index ratio of 0.85 also indicates a likely amenability to SAG milling; however, more detailed testwork would be required to confirm this. No abrasion tests were carried out. A medium abrasion index of 0.4 (equivalent to quartz) was assumed for the ore and was used as a guide throughout the preliminary design of the processing circuit.

The comminution testwork results are shown in Table 16.1.3_1.

Table 16.1.3_1
Ollachea Gold Project
Comminution Testwork Summary

Composite Number	Sample No.	Crushing Work Index, kWh/t	Rod Mill Work Index, kWh/t	Ball Mill Work Index, kWh/t	Abrasion Index
N/A	41719	N/A	13.84 (@ 1180µm)		N/A
N/A	42286	N/A		16.2 (@ 75µm)	N/A

Grind Size

Only one set of grind optimisation tests was conducted at the beginning of the testwork program. Grind sizes with a P_{80} of 180µm, 125µm and 75µm were tested using direct cyanide leach tests. It was not known at the time of the work that the samples were preg-robbing in nature and that all subsequent leach tests would need to be carried out in the presence of activated carbon. As a consequence, the results all showed very poor recoveries (<20%) and did not indicate any significant variance of recovery versus grind size.

The summary of the results are shown in Tables 16.1.3_2 and Table 16.1.3_3 as tests 41727A, 41727B and 41727C.

Table 16.1.3_2
Ollachea Gold Project
Cyanide Leach Testwork Gold Summary

KCA		Description	Type	Calculated Head	Extracted	Average Tails	Au Extracted	Calc. P_{80} Tail Size	Leach Time	Consumption NaCN	Addition $Ca(OH)_2$	Addition NaOH
Sample No.	Test No.			g/t Au	g/t Au	g/t Au	%	mm	hours	kg/t	kg/t	kg/t
41719	41727 A	OL04-A	Direct	1.65	0.24	1.41	15%	0.180	36	0.63	0.5	--
41719	41727 B	OL04-A	Direct	1.66	0.34	1.32	20%	0.125	36	0.80	0.5	--
41719	41727 C	OL04-A	Direct	1.61	0.31	1.30	19%	0.075	36	0.80	0.5	--
41956	42247 A	OL22 - A	Direct	3.24	2.28	0.96	70%	0.099	72	2.72	1.00	--
41958	42247 B	OL25 - A	Direct	2.16	0.56	1.60	26%	0.113	72	4.44	1.00	--
41960	42247 C	OL26 - A	Direct	17.47	13.88	3.59	79%	0.098	72	3.69	1.00	--
41962	42247 D	OL26 - B	Direct	1.79	0.32	1.47	18%	0.089	72	1.29	0.50	--
Average				**4.23**	**2.56**	**1.67**	**35%**	**0.111**	**--**	**2.05**	**0.71**	**--**
41719	41729 A	OL04-A	CIL	1.63	1.38	0.25	85%	0.075	36	1.30	0.5	--
41719	41729 B	OL04-A	CIL	2.35	2.08	0.28	88%	0.075	36	1.27	--	0.5
41719	41729 C	OL04-A	CIL	1.70	1.49	0.22	87%	0.075	36	1.32	--	1.75
41956	42267 A	OL22 - A	CIL	3.42	3.26	0.17	95%	0.071	36	2.49	--	1.00
41958	42267 B	OL25 - A	CIL	2.07	1.69	0.37	82%	0.073	36	2.98	--	1.50
41960	42267 C	OL26 - A	CIL	27.21	21.98	5.23	81%	0.065	36	2.43	--	1.50
41962	42267 D	OL26 - B	CIL	2.36	2.11	0.25	89%	0.072	36	1.64	--	1.00
Average				**5.82**	**4.86**	**0.97**	**87%**	**0.072**	**--**	**1.92**	**0.50**	**1.21**

Table 16.1.3_3

Ollachea Gold Project

Cyanide Leach Testwork Silver Summary

KCA	KCA	Description	Type	Calculated Head	Extracted	Average Tails	Ag Extracted	Calc. P_{80} Tail Size	Leach Time	Consumption NaCN	Addition $Ca(OH)_2$	Addition NaOH
Sample No.	Test No.			g/t Ag	g/t Ag	g/t Ag	%	mm	hours	kg/t	kg/t	kg/t
41719	41727 A	OL04-A	Direct	3.00	0.3	2.70	9%	0.180	36	0.63	0.5	--
41719	41727 B	OL04-A	Direct	3.00	0.5	2.60	15%	0.125	36	0.80	0.5	--
41719	41727 C	OL04-A	Direct	3.10	0.3	2.70	10%	0.075	36	0.80	0.5	--
41956	42247 A	OL22 - A	Direct	3.19	0.46	2.73	14%	0.099	72	2.72	1.00	--
41958	42247 B	OL25 - A	Direct	2.88	0.17	2.71	6%	0.113	72	4.44	1.00	--
41960	42247 C	OL26 - A	Direct	3.76	1.03	2.72	28%	0.098	72	3.69	1.00	--
41962	42247 D	OL26 - B	Direct	3.09	0.19	2.90	6%	0.089	72	1.29	0.50	--
Average				**3.15**	**0.42**	**2.72**	**13%**	**0.111**	**--**	**2.05**	**0.71**	**--**
41719	41729 A	OL04-A	CIL	3.00	0.1	2.90	1%	0.075	36	1.30	0.5	--
41719	41729 B	OL04-A	CIL	4.00	1.1	2.90	27%	0.075	36	1.27	--	0.5
41719	41729 C	OL04-A	CIL	3.50	1.3	2.20	36%	0.075	36	1.32	--	1.75
41956	42267 A	OL22 - A	CIL	3.18	0.47	2.71	15%	0.071	36	2.49	--	1.00
41958	42267 B	OL25 - A	CIL	3.13	0.06	3.07	2%	0.073	36	2.98	--	1.50
41960	42267 C	OL26 - A	CIL	5.43	2.37	3.06	44%	0.065	36	2.43	--	1.50
41962	42267 D	OL26 - B	CIL	3.97	0.92	3.06	23%	0.072	36	1.64	--	1.00
Average				**3.74**	**0.90**	**2.84**	**21%**	**0.072**	**--**	**1.92**	**0.50**	**1.21**

Gravity Concentration

A Gravity recovery test was conducted on only one sample using a laboratory Knelson Concentrator for primary recovery. The result is shown in Table 16.1.3_4.

Table 16.1.3_4
Ollachea Gold Project
Gravity Recovery Testwork

Sample Number		Au Grade g/t	Knelson Gravity - Cumulative Recovery %		
			Au	Ag	Wt Recovery
41719	Head	1.496	0	0	0
	Conc. 1	53.52	35	0	1.0
	Conc. 2	24.65	58	1	2.4
	Conc. 3	7.27	62	1	3.3
	Mids	2.67	85	15	15.8
	Tails	0.27	100	100	100

The results indicate that the Ollachea deposit contains a moderate portion of gravity recoverable gold. There needs to be a substantially larger amount of testwork carried out to establish the true gravity recovery and the effect that removing the gravity recoverable gold will have on any downstream processes. An increase in the design recovery, due to gravity separation and CIL processing, may occur as a result of the further testwork.

The result showed that silver did not report to the gravity circuit when a low mass recovery is obtained, as is the case with the Knelson Concentrator. It is recommended that a gravity circuit be installed in the process plant but that little or no silver recovery can be expected.

Cyanide Leaching

The testwork program was generally split into two categories; direct cyanide leach (CIP) and CIL leaching. Seven tests were carried out within each category. At least one leach test was carried out on each of the five composite samples, with an extra two tests being carried out on the major composite – OL04-A. The tests were attempted to be conducted at a grind P_{80} of 75µm with excess cyanide to ensure that the gold leaching rate and extraction was not inhibited. The results show that the addition rates of cyanide during the initial section of the test program were not sufficiently in excess to bring about the desired effects. Summaries of the testwork results are presented in Table 16.1.3_2 and Table 16.1.3_3. All of the tests were carried out without the removal of any of the gravity recoverable gold prior to leaching.

As the results of the CIP tests came to hand, it was decided to change all future leaching testwork to CIL tests.

The presence of carbonaceous material within the ore makes it imperative that CIL processing is adopted. The high amount of reactive pyrrhotite in the ore may also mean that efficient oxygenation is required in the CIL circuit. The risk of this step is that if any residual cyanide is present in the process water, then pre-leaching and, hence, preg-robbing may occur before the slurry is mixed with activated carbon.

It is extremely important that the nature of the preg-robbing species is tested and fully understood in the next series of testwork together with oxygenation tests. An alternate theory on the cause of preg-robbing within ores similar in nature to Ollachea is that the pyrrhotite requires oxidation to release atomic lattice gold, which the testwork probably would not have done due to the amount and method of oxygen addition in the tests. It may be more efficient to ensure that there is a large excess of oxygen in the system which will oxidise the reactive sulphide sites. These sulphide sites will prefer an oxygen atom to a larger gold cyanide species during the dissolution of gold via cyanidation.

Figures 16.1.3_1 and Table 16.1.3_2 show the gold and silver leach rates for all samples tested under CIP conditions. No leach rate data was recorded for any of the CIL tests.

Figure 16.1.3_1
CIP Gold Recovery Rate vs Time

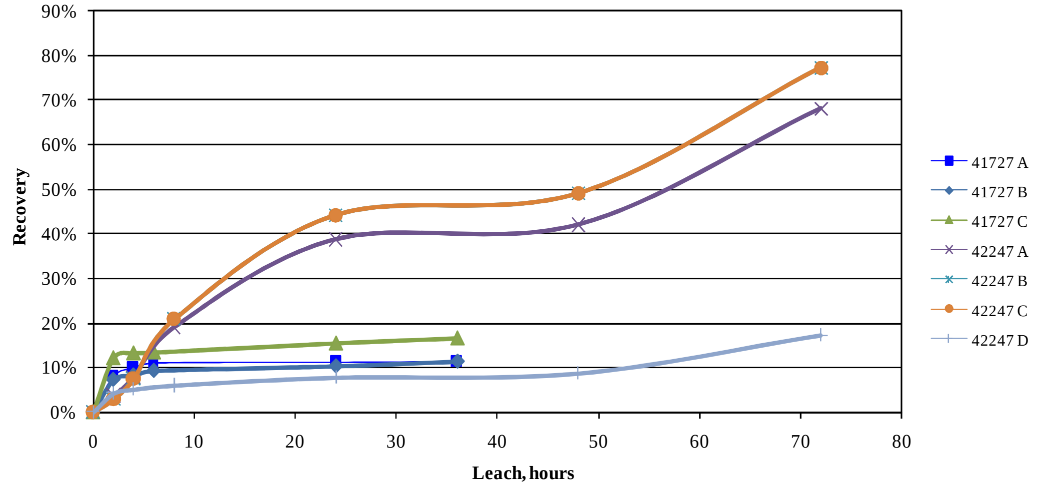
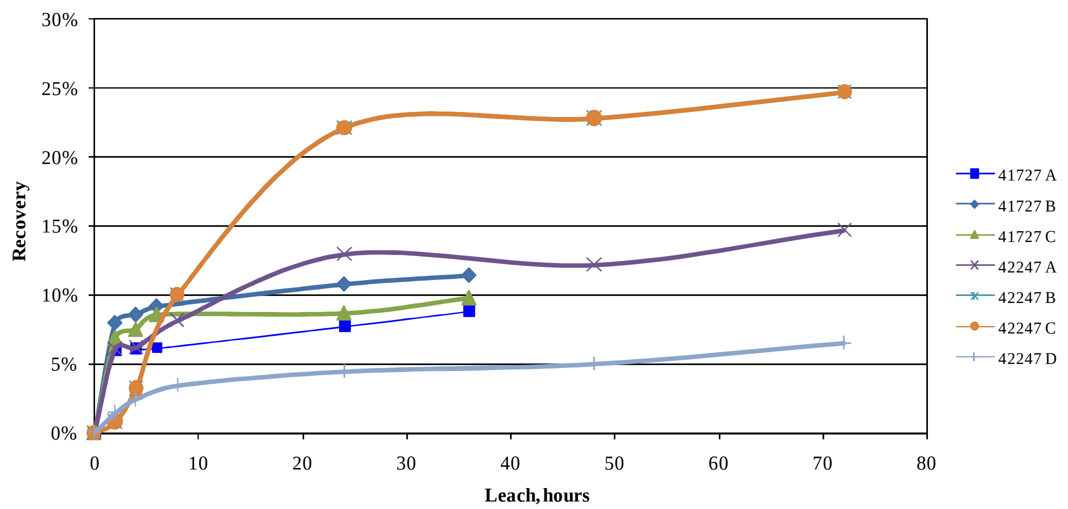


Figure 16.1.3_2
CIP Silver Recovery Rate vs Time



The carbonaceous/pyrrhotitic nature of the samples was clearly seen to have caused a significant preg-robbing effect in the leach results.

Observations from the leach testwork were:

- Gold
    - The CIP gold recovery after 24 hours (design leach time) is shown to be extremely low at between 8% and 14%. The variability of the preg-robbing nature of the ore is seen in tests 42247A and 42247C, which were both relatively slow leaching but produced recoveries of 38% and 44% respectively after 24 hours of leaching.
    - Leaching appeared to have been initially rapid in most cases until approximately eight hours. Leaching continued to occur at various rates for each of the composites; however, a significant increase in leach rate appears to have occurred after 48 hours of leaching. This could be due to either a reduction in the relative activity of the carbonaceous material in the ore in relation to the propensity of the fresh cyanide addition to redissolve gold from the preg-robbing species, or that pyrrhotite or similar reactive sulphides have preg-robbed the soluble gold and are then substituted by cyanide itself, releasing gold cyanide. If the carbonaceous material is primarily graphite, then it would not have a high affinity for gold adsorption due to its relatively low surface area (compared to porous carbons).
    - The poor recovery of gold in tests 41727A-C is believed to also be due to the low quantity of cyanide added throughout the tests. Excess cyanide must be added to drive gold dissolution away from the preg-robbing material in the ore.
    - CIP lime consumptions were low at 0.7kg/t.
    - CIP cyanide consumptions varied between 0.63kg/t and 4.44kg/t. On average, more than 20 percent of the cyanide consumption occurred in the leaching period after 24 hours.
    - The CIL gold recovery after 36 hours was significantly higher than CIP: - between 81% and 95 percent. The preg-robbing nature of the samples was able to be minimised via the addition of activated carbon; however, the leach rates were not recorded during the CIL tests. The lowest recovery occurred on the sample with the highest head grade. This may be due to the gold being either coarse or locked within sulphide minerals.
    - The CIL tests appeared to have been able to leach the ore to a relatively low range of tail grades, 0.17g/t to 0.37g/t. One test produced a tail grade of 5.23g/t Au; however, this was in relation to the extremely high head grade of 21.98g/t Au.
    - The CIL lime consumption was also low at approximately 1.21kg/t.
    - CIL cyanide consumptions varied from 1.27kg/t to 2.98kg/t during the 36 hours of leaching. Consumptions measured after 24 hours were on average 17 percent less than the 36 hour consumptions. The cyanide consumptions in the CIL tests are inflated in comparison to that which is expected in an operating circuit. It is expected that a cyanide consumption of 1.5kg/t could be achieved in the full operating circuit; however, this needs to be established as a part of any future test program. There is a risk that cyanide consumptions could be affected by the reactive pyrrhotite in the ore, if effective oxygenation of the pulp is not undertaken.

- Silver
  - The CIP silver recovery after 24 hours (design leach time) varied from 4% to 22% with an average of 13%.
  - Leaching also appeared to have been initially rapid in most cases until approximately eight hours.
  - The CIL silver recovery after 36 hours was between 1% and 44 percent with a significant improvement in the average to 21%. The silver grades at approximately 3.7g/t were significantly higher than that which is expected to be presented to the plant operation.

The results show the gold is generally free milling for all the samples when leached in the presence of activated carbon. It is not amenable to CIP, likely due to the deleterious effect of the carbonaceous and/or pyrrhotitic material within the ore. Moderate to moderately high cyanide consumptions can be expected. The average cyanide consumption for all samples tested under CIL conditions was 1.69kg/t after 24 hours of leaching. It is expected that this consumption could be slightly reduced in the operating plant. The average lime consumption was 1.1kg/t. The average overall CIL gold recovery, discounting test 42267C due to its abnormally high head grade, was 87.7% for an average head grade of 2.26g/t Au.

Whilst an average recovery of 21% silver was made during the CIL tests, the operating plant silver extraction is expected to be lower due to the inflated silver head grades presented during the test program.

Flotation

Batch flotation testwork was conducted on samples with total head sulphur grades of 2.7% with an equivalent percentage of total carbon. It was unclear which composites were tested throughout the program other than tests 42727 and 42728 which originated from composite OL04-A. No gold gravity separation was carried out prior to flotation. The target P_{80} grind size for the tests was 75µm; however, this was not confirmed as part of the test results. Two sighter tests, 41719A & B were carried out, attempting to pre-float carbon, followed by a single stage of roughing. Tests 42269 to 42271 consisted of one stage of carbon pre-float followed by four stages of roughing. Tests 42727 to 42730 and test 42289 all collected 5 concentrates.

A summary of the flotation testwork is given in Table 16.1.3_5. Figure 16.1.3_3 shows the gold recoveries versus mass recovery.

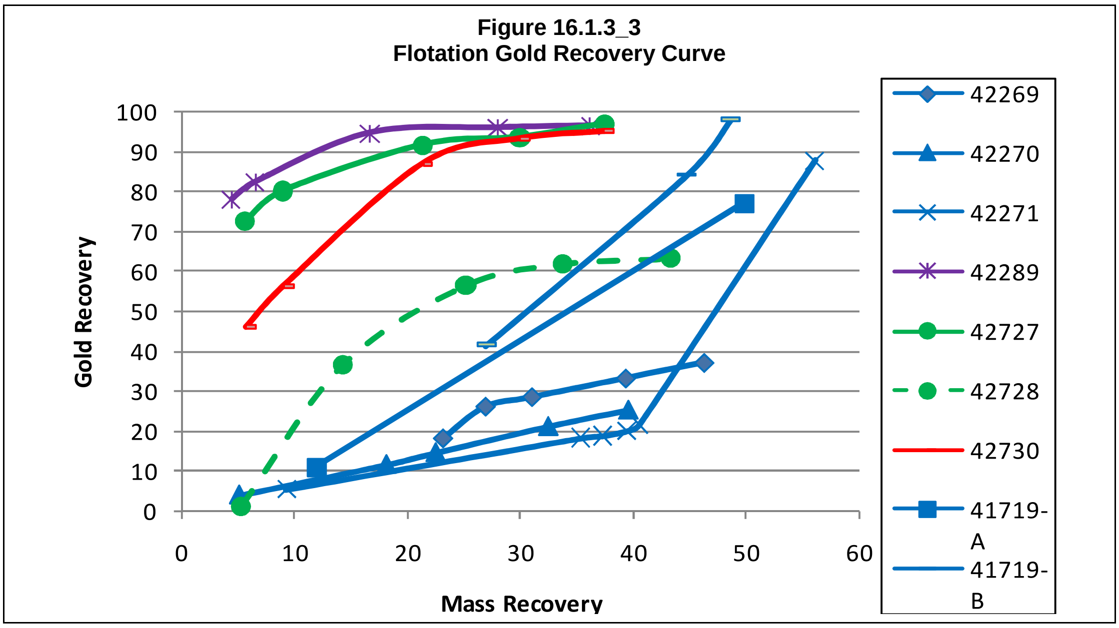
All of the lines shown in blue represent either the sighter tests or tests where Eh conditions were attempted to be controlled. The remaining four tests show the results of floating under neutral flotation conditions. Whilst it is evident that encouraging flotation performance can be achieved, a greater understanding of the outcomes of tests 42727 and 42728 needs to be made. These tests were carried out on the same sample under the same conditions and produced significantly different results.

Table 16.1.3_5

Ollachea Gold Project

Flotation Testwork Summary

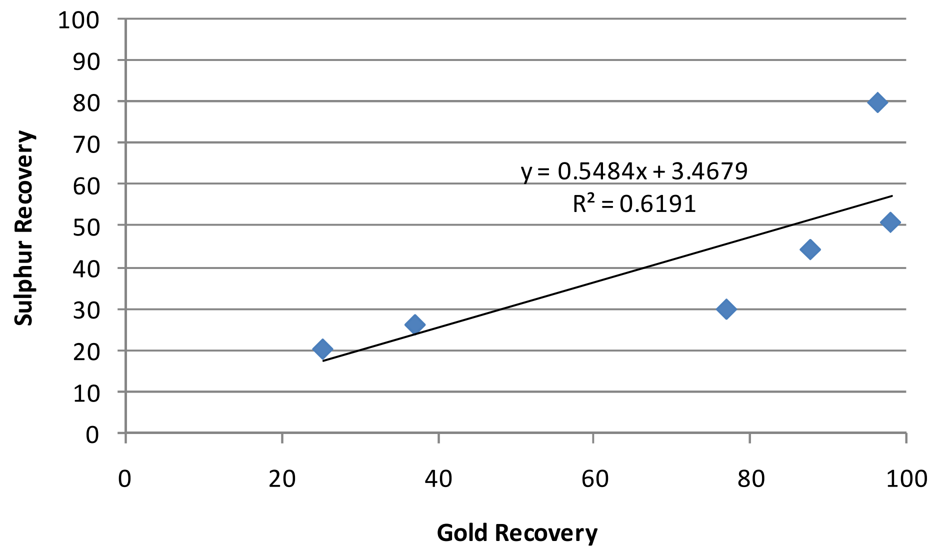
Sample	Composite	Head Grade	Flotation Conditions								Flotation Concentrate									Overall Tail Grade
			Conditioning Time	Float Time	$CuSO_4$	PAX	Na_2SiO_3	Soda Ash / H_2SO_4	6697	AF 65	Mass	Grade				Distribution %				
		g/t Au	min	min	g/t	g/t	g/t	g/t	g/t	g/t	%	g/t Au	C_{tot} %	S_{tot} %	S_{sul} %	Au	C_{tot}	S_{tot}	S_{sul}	g/t Au
41719-A		2.84	20	10		0 + (122 Diesel)	200	120		151	50	3.22	1.55	1.63	--	77	65	30	--	0.96
41719-B		2.84	30	15	200	50 + 61 (diesel)	200	120	10	76	49	2.88	1.69	2.87	--	98	69	51	--	0.05
42269		2.36	25	13	201	50	201	1790	10	161	46	1.89	1.46	1.56	1.53	37	60	26	9	2.76
42270		1.35	25	13	201	50	392	161	10	222	39	0.87	1.65	1.45	1.41	25	57	20	4	1.67
42271		2.33	20	16	301	70	390	180	90	100	41	1.25	1.72	1.07	1.04	88	75	44	18	0.65
42289		1.56	10	20	300	100			20	113	36	4.15	1.78	6.26	6.21	96	55	80	81	0.09
42727	OL04-A	1.38	10	20	303	101			20	134	37	3.58				97				0.07
42728	OL04-A	1.69	11	20	301	100			20	133	43	2.47				63				1.10
42730		3.55	10	20	303	101			20	134	37	9.06				95				0.27



Observations from the flotation testwork were:

- Flotation of the composite samples achieved gold recoveries between 25% and 98%, total sulphur recoveries between 20% and 80%, total carbon recoveries between 55% and 75% producing mass recoveries between 37% and 50%.
- The tests show that gold appears to be associated with the faster floating sulphides. There appears to be a trend of gold recovery versus sulphur recovery, as shown in Figure 16.1.3_4. This indicates the gold is associated with only certain types of sulphides as highlighted in the mineralogical study. Flotation testwork with and without gravity should be able to confirm whether the remaining gold can be recovery via gravity separation.
- Some of the samples have slow float kinetics.
- Mass recoveries are high for all of the samples. A cleaner step with a regrind may decrease the mass recovery without seriously compromising the gold recovery but requires testing.
- Repeatability was poor with tests 42727 and 42728 being carried out on the same composite and essentially the same flotation regime whilst producing significantly different results.
- The flotation reagent schemes used for the testwork were relatively simple, only requiring PAX, copper sulphate and AF 65. Adjustment of the flotation Eh via the addition of either soda ash or sulphuric acid was unable to improve flotation selectivity. Reagents dosages were moderate.

Figure 16.1.3_4
Flotation Gold versus Sulphur Recovery



<u>Magnetic Separation</u>

One magnetic separation test was conducted to examine whether it was possible to concentrate the gold into a low mass high recovery concentrate. Table 16.1.3_6 shows that whilst it is possible to upgrade the gold into a magnetic concentrate, it was not possible to produce a throw away tail. No downstream testwork was carried out on either of the products.

Table 16.1.3_6
Ollachea Gold Project
Magnetic Separation Testwork

KCA Sample No.	KCA Test No.		Wt., grams	Mass Recovery %	Grade				Distribution %			
					Au g/t	Ag g/t	Total C, %	Total S, %	Wt. % Au	Wt. % Ag	Wt. % C	Wt. % S
41719	41770 A	Mags	110.92	11%	12.24	37.4	0.74%	15.91%	50%	35%	7%	87%
	41770 B	Non-mags	875.99	89%	1.58	8.7	1.25%	0.31%	50%	65%	93%	13%
	Total		986.91	100%	--	--	--	--	100%	100%	100%	100%
	C/Head		--	--	2.78	12.0	1.19%	2.06%	--	--	--	--

16.1.4 Recoveries

The recommended recoveries and the derivation for use in the scoping study are tabulated below (Table 16.1.4_1).

Table 16.1.4_1
Ollachea Gold Project
Processing Plant Design Recoveries

Process	Metal	Recovery %	Derivation
Gravity	Gold	20	Indication from single gravity recovery test
	Silver	0	Indication from single gravity recovery test
CIL	Gold	89	Only 7 tests were conducted using CIL none at 24hr design leach time, 1 from the high grade composite OL26-A. The average of the tests after 36 hours leaching was used to obtain an indicated tail grade and extrapolated for the expected design head grade for the Project.
	Silver	10	As indicated from the CIL test program and downgraded due to the lower expected silver grade than was tested

16.2 Processing

16.2.1 Design Criteria

Preliminary process design criteria have been developed for a processing flowsheet at Ollachea. The major source of data was from the metallurgical testwork and discussions with MKK technical personnel. Where this data was deficient, typical industry values, data from similar projects, Coffey Mining experience or Coffey Mining in-house data has been used.

16.2.2 Flowsheet Development and Description

The process plant will treat 1Mtpa of ore at a nominal throughput of 125t/h. The selected flowsheet shown in Figure 16.2.2_1 has been considered for treating the ore. The process is described as follow;

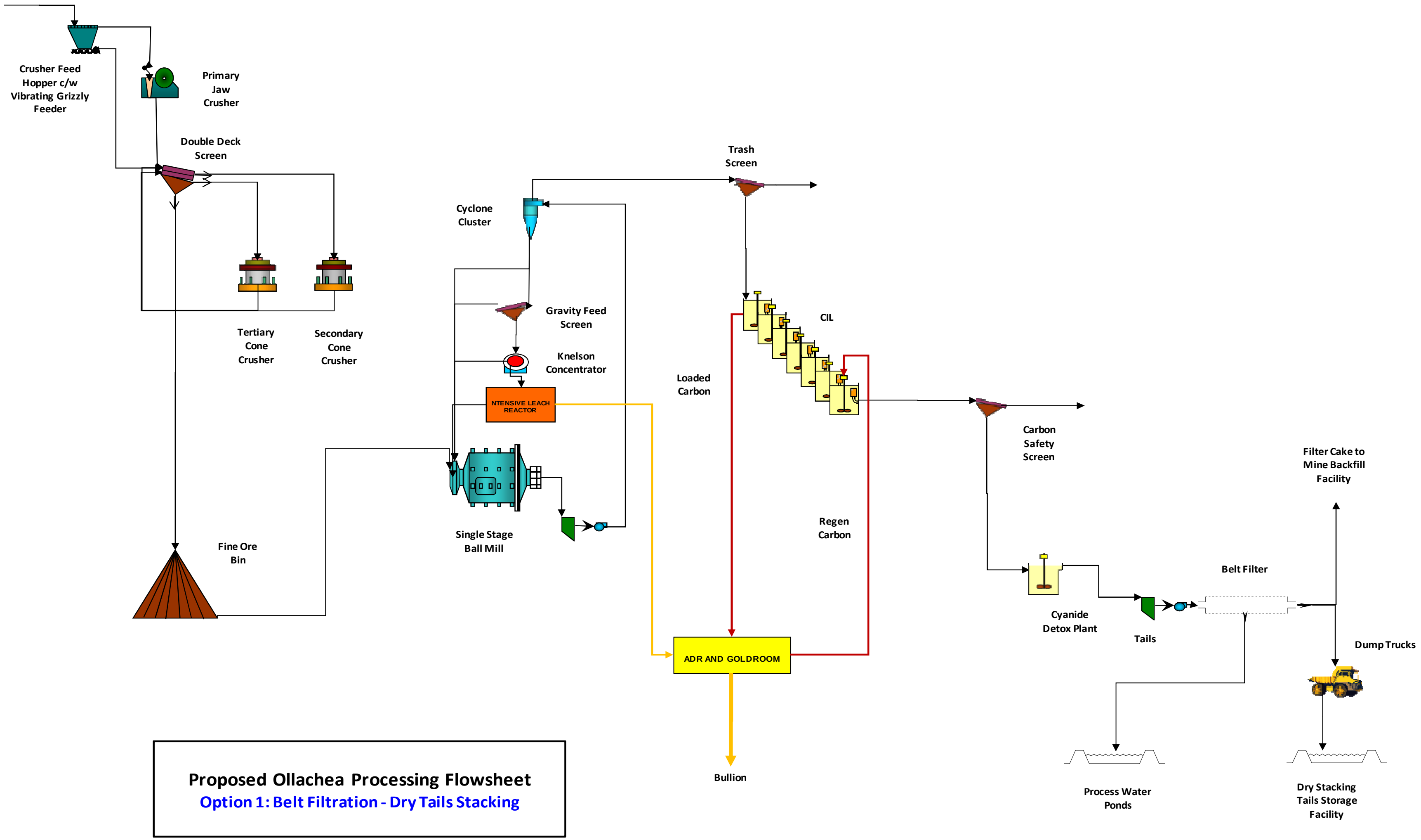
Comminution

Comminution testwork indicated that the ore is amenable to ball milling and can be milled in conjunction with a three stage crushing circuit. A three stage crushing circuit direct feeding a single stage ball mill is considered a suitable comminution circuit due to its simplicity.

Ore from the ROM pad will be either direct-fed or blended via stockpiles and a front end loader, which will load the ROM bin ahead of the primary crusher at a rate of approximately 200tph. Ore will be discharged from the bin by a feeder onto a vibrating grizzly with 100mm slots. Grizzly undersize will report to the primary crusher discharge conveyor. Grizzly oversize will report to the jaw crusher for primary crushing. The primary crusher will have a closed side setting (CSS) of 70mm. The primary crushed product will discharge onto the crusher discharge conveyor that will feed a double deck screen. The top screen will have apertures of 30mm. The bottom deck will have apertures of 10mm. The top screen oversize will report directly to the secondary crusher. The secondary crusher will have a CSS of 25mm. The secondary crushed product will be conveyed back to the head of the double deck screen. The bottom screen oversize will report directly to the tertiary crusher. The tertiary crusher will have a CSS of 8mm. The tertiary crushed product will also be conveyed back to the head of the double deck screen. The fine product from the double deck screen is expected to have a P_{80} of 8mm. This material will be conveyed to the crushed ore feed bin.

The crushed ore will be direct fed into the ball mill which will have an installed motor power of 2,500kW. Ball mill discharge will report to the mill discharge hopper and be pumped to the cyclone cluster.

Figure 16.2.2_1
Base Case – Process Flowsheet



Gravity

The one gravity separation test performed indicated that a significant quantity of gold is able to be recovered via a gravity separation circuit. A Knelson concentrator circuit is nominated in the process flowsheet. Concentrate from the Knelson will be collected and direct leached in an in-line leach reactor as described below.

Cyclone underflow will be split to feed the gravity circuit. 30 percent of the cyclone underflow will be treated through the gravity circuit with the remainder reporting back to the ball mill feed chute. This equates to 75 percent of fresh mill feed being treated through the gravity circuit. A Knelson XD30 capable of treating 100tph of feed has been chosen for the duty. It is expected that the concentrator would operate on a one hour cycle time. Gravity concentrate will be periodically treated through an intensive cyanidation process to dissolve gold and silver for treatment in a common electrowinning circuit. The leach reactor chosen is an Acacia CS500 capable of treating 1 tonne per day of gravity concentrate. Cyclone overflow will report to the downstream "Gold Recovery Process".

Flotation

The sighter flotation testwork program was carried out to establish whether a low mass high gold recovery flotation concentrate could be produced that allowed a direct disposable tail to be made. The outcomes of the flotation tests were promising; however, the large variability in the results has prevented a flotation circuit design being recommended at this stage. The recoveries were high; however, a throw away flotation tail was not able to be produced from the testwork to date.

Further flotation testwork is recommended to be carried out to establish whether repeatable results can be produced that will allow a higher level of confidence in any future designs.

Gold Recovery Process

Gold recoveries of greater than 80% were achieved for all samples tested via CIL. The cyanide consumption is expected to be moderate through a CIL circuit.

The leach circuit will have a 24 hour residence time. The circuit will have seven $952m^3$ tanks configured in series. No CIL kinetic testwork has been completed to confirm that this residence time will be able to achieve the expected recoveries within the circuit and as such, further CIL leach kinetic and variability testwork is recommended. Improved leach characteristics are expected in the processing plant in comparison to the testwork due to the fact that no pre-gravity separation was carried out prior to CIL testing. The leach kinetics are expected to be substantially improved as a result of the inclusion of the gravity circuit.

A significant aspect of the design criteria for the circuit is that the activated carbon must be maintained at a high level of activity and be moved through the circuit at a relatively rapid rate to make sure that it is able to absorb gold from solution faster than the carbonaceous components of the ore. The activated carbon is expected to load to only 1,100g/t Au and will have a residence time of only two weeks within the slurry. An allowance has been made to screen all of the recycled slurry containing loaded carbon, so that the size of tanks and intertank screens can be kept to a minimum. This will allow slurry to continue to flow down the train and only have a dense medium of carbon transferred up the train to the next tank in the series.

CIL tails will be detoxified using the SO_2/air process prior to being filtered via belt filtration. Detoxified process water that is either recovered from the belt filters or the TSF will be stored in a process water pond prior to be recycled back into the process.

Elution

A 5.5t AARL capacity circuit with acid wash and elution columns is nominated for the elution circuit. A hot elution will be conducted to strip the gold from the carbon. This will be followed by electrowinning and smelting to produce gold doré. The circuit has been designed to allow 12 strips per week (1.7 strips per day) to be completed.

17 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

17.1 Mineral Resources Estimate

Coffey Mining has estimated the Mineral Resource for the Ollachea Gold Project as at 6th October 2009. 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 mineralized interpretations that were created with the assistance of MKK geologists.

17.1.1 Data

The Ollachea resource estimate is based on diamond core (DC) drilling. The estimate contains 63 DC holes totalling 22,240.15m. The estimate contain assay data up to and including hole DDH09-61.

A total of 678 bulk density determinations have been collected from the DC campaign and used as the basis for tonnage reporting. The samples were used to give an average in-situ dry bulk density of 2.80 g/cm^3.

17.1.2 Geological Model

Seven high grade domains have been interpreted using N-S oriented vertical sections based on grade information and geological observations from Coffey Mining and MKK's geologist, Mr Willy Caceres Luna of MKK, who spent a week with Coffey Mining assisting in the geological interpretation.

Interpretation and digitizing of all constraining boundaries has been undertaken on cross sections orientated at 180º (drill line orientation). The interpretation was completed using 16 vertical sections. Figure 17.1.1_1 shows the vertical section locations.

The resultant digitized boundaries have been used to construct wireframe defining the three-dimensional geometry of each interpreted feature. The interpretation and wireframe models have been developed using the Gemcom Surpac mine planning software package.

For the purpose of resource estimation, seven main high grade mineralized domains were interpreted and modelled on a lower cutoff grade of 1.0g/t Au. The domains are shown below in Figures 17.1.2_1 and 17.1.2_2.

The Ollachea interpretation has been restricted to the high grade, relatively continuous zones. There is no low grade envelop modelled due to the inconsistent data which is mainly limited to the wide drill spacing. Coffey Mining recommends a new geological interpretation will be required to define the low grade mineralized zone once the drill density is increased.

Figure 17.1.1_1
Drillhole Type by Deposit – Plan View

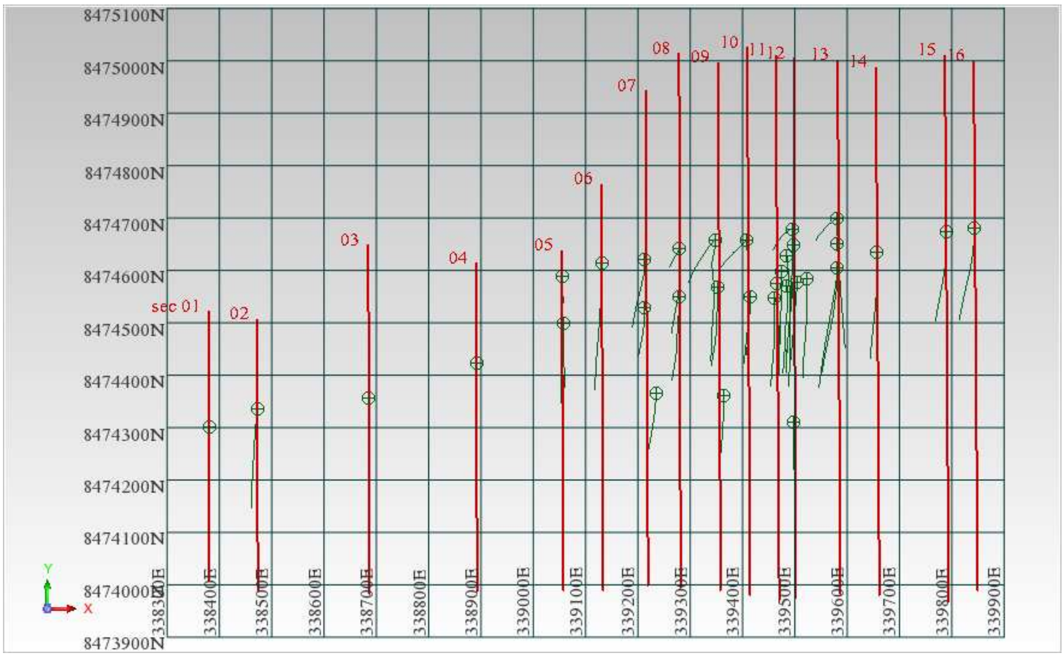


Figure 17.1.2_1
Vertical Section 339496

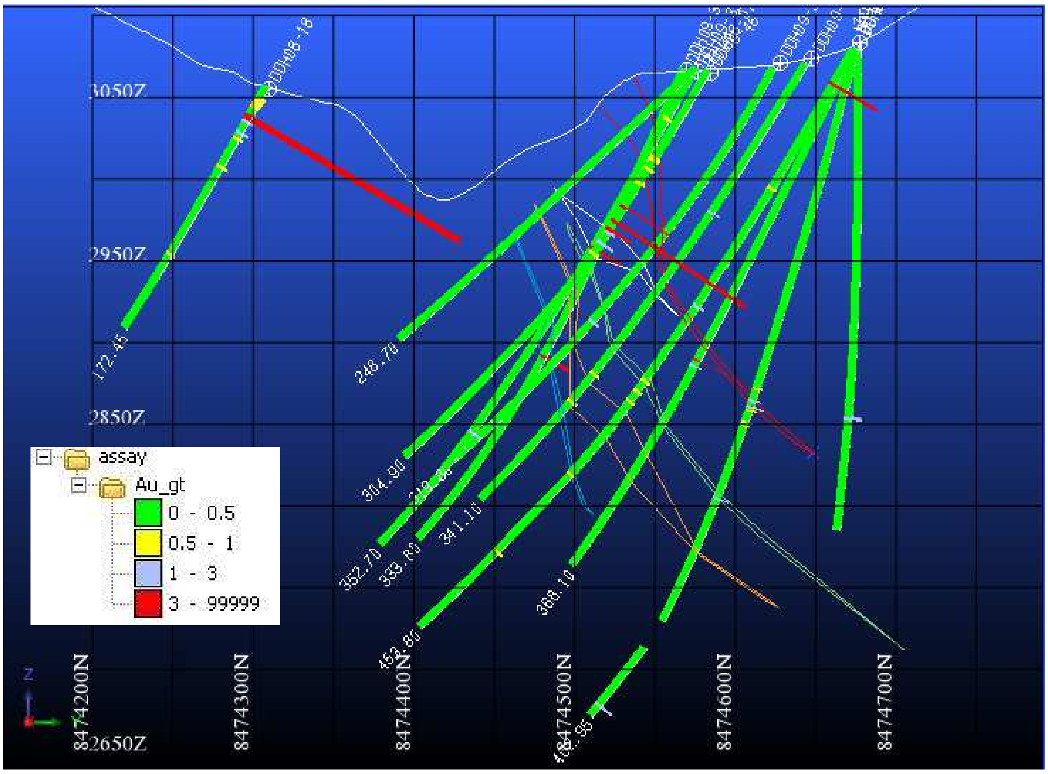
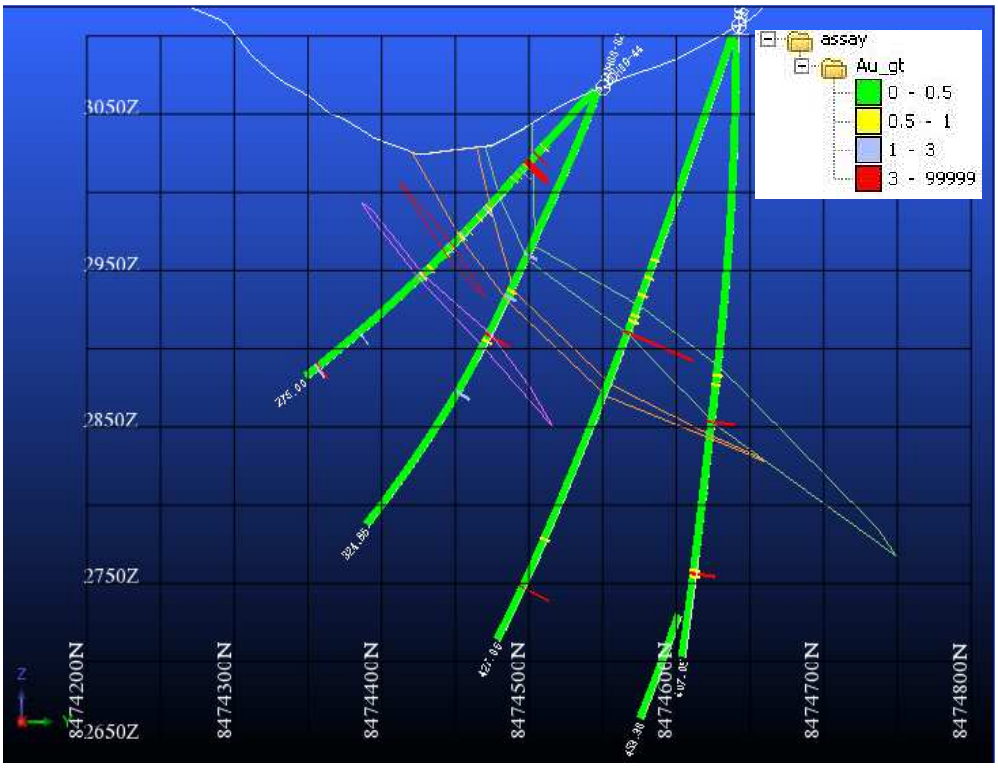


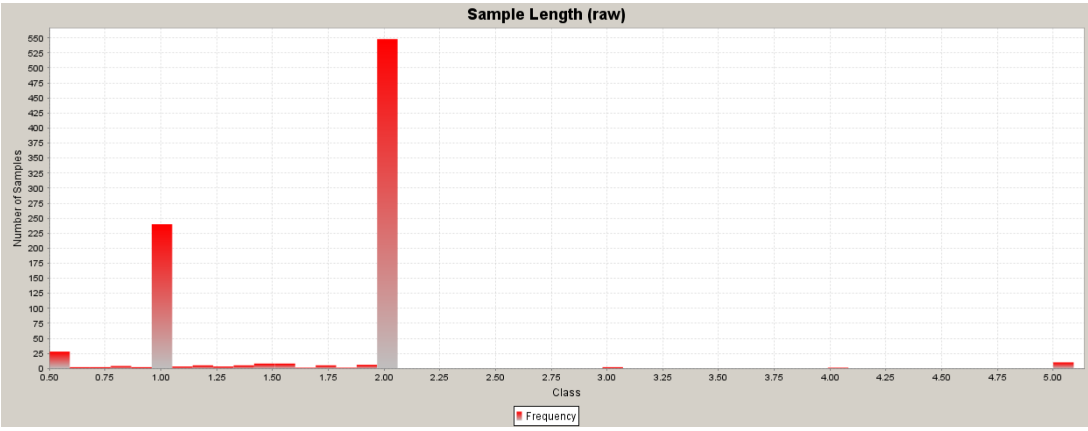
Figure 17.1.2_2
Vertical Section 339279



17.1.3 Compositing, Basic Statistics and High Grade Cuts

A statistical analysis was completed by Coffey Mining on 2m downhole composites (accepting residual lengths greater than 75% of composite length). The compositing was completed using the Gemcom Surpac mining software package. Figure 17.1.3_1 below shows the raw sample length histogram.

Figure 17.1.3_1
Samples Length Histogram



The basic statistics were calculated in the table below to assist in determining if a high grade cap was warranted.

Table 17.1.3_1 presents the summary of the statistical analysis for the complete composite data and those restricted by the mineralized wireframes.

Table 17.1.3_1
Ollachea Project
2m Composites Basic Statistics Summary

Variable	Mean	Variance	Std Dev	CV%	Samples	Min.	Max.
Total Project Database Composites	0.45	6.85	2.62	579%	10,317	0.00	153.00
Combined Mineralized Zones Composites	3.75	72.55	8.52	228%	639	0.01	152.93

Figures 17.1.3_2 and 17.1.3_3 show the composites statistics analyses.

Coffey Mining decided to set the top cut at 50g/t. There are only three values over 50g/t and cutting those three values reduces the CV from 2.27 to 1.64. Figure 17.1.3_4 shows the effect of the three high grade values has to the dataset.

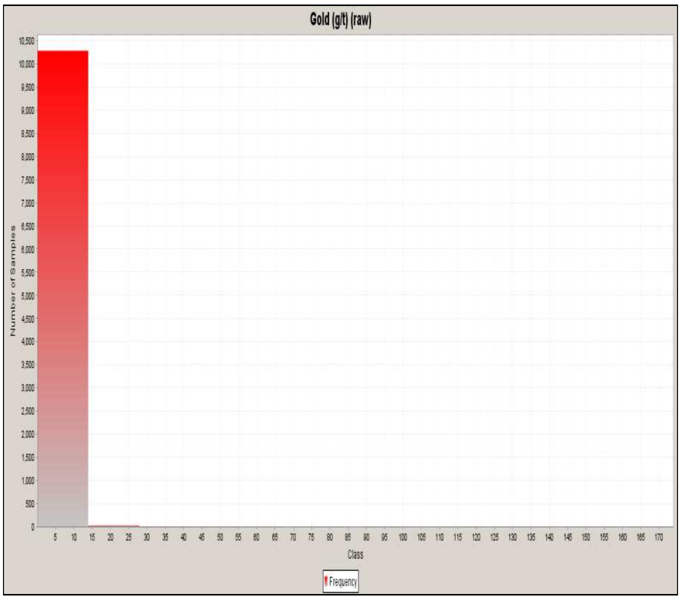
Figure 17.1.3_2
2m Project Database Composites Statistic Study

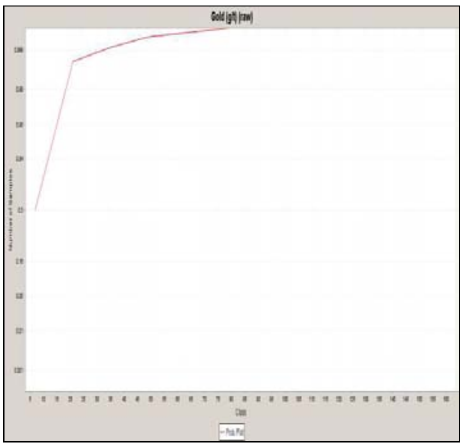
Target:	Ollachea
Zone	All Samples
Variable:	Au (ppm)

Samples:	10 317
Minimum:	0.00
Maximum:	153.00
Classes:	14
Classes Interval:	10.64

Quantiles	
2.5%:	0.003
5.0%:	0.01
25.0%:	0.02
Median:	0.05
75.0%:	0.18
95.0%:	1.80
97.5%:	3.29

Mean:	0.45
Variance:	6.85
Std Dev.:	2.62
C.V.:	579%
Range interquartil:	0.16





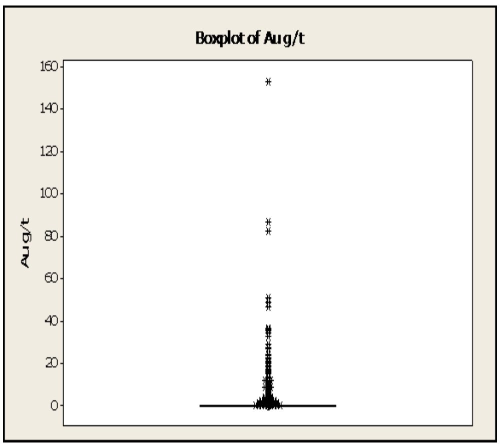


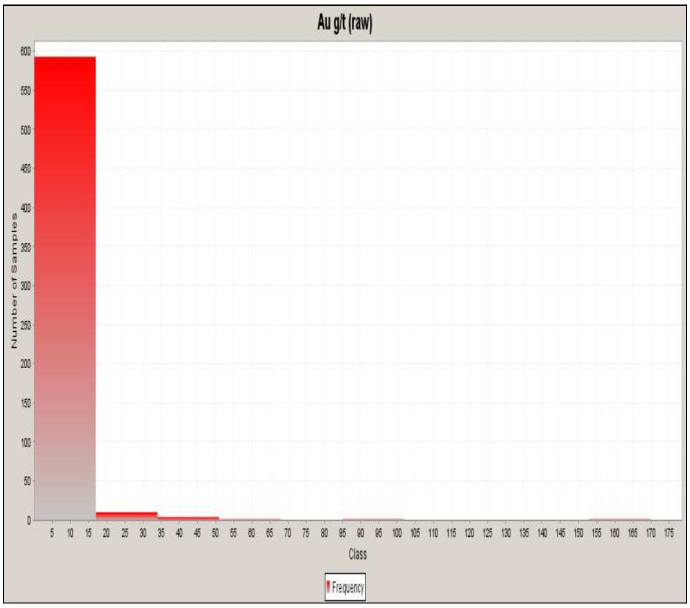
Figure 17.1.3_3
2m Mineralized Zones Composites Statistic Study

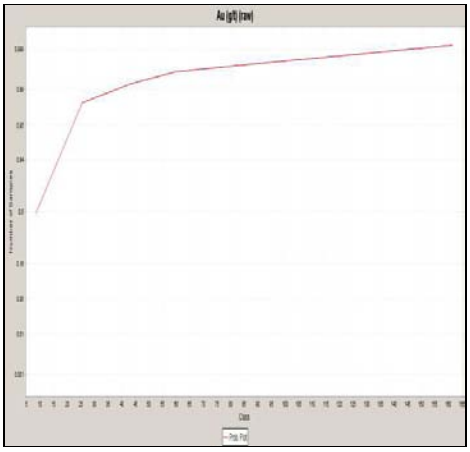
Target:	Ollachea
Zone	Mineralised
Variable:	Au (ppm)

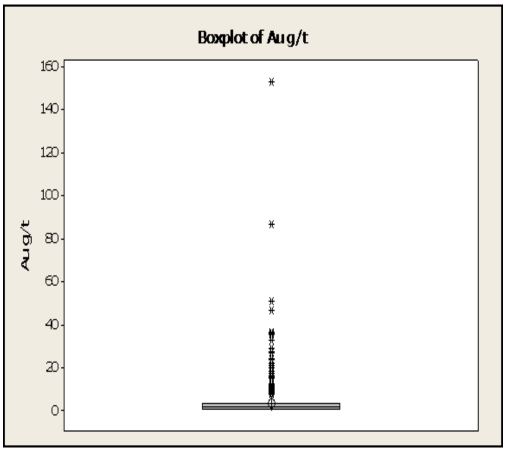
Samples:	639
Minimum:	0.01
Maximum:	152.93
Classes:	10
Classes Interval:	14.78

Quantiles	
2.5%:	0.06
5.0%:	0.12
25.0%:	1.10
Median:	1.90
75.0%:	3.74
95.0%:	11.38
97.5%:	19.25

Mean:	3.75
Variance:	72.55
Std Dev.:	8.52
C.V.:	228%
Range interquartil:	2.64







The effect of top cutting the data at a 50g/t cutoff is shown Figure 17.1.3_5. Only 0.4% of the total data is affected and 5.7% of the metal. The total mean is reduced by only 4%, however more importantly; the variance of dataset is reduced from 69.02 to 31.88.

Figure 17.1.3_4
Outlier Analysis - Composites Statistic Study

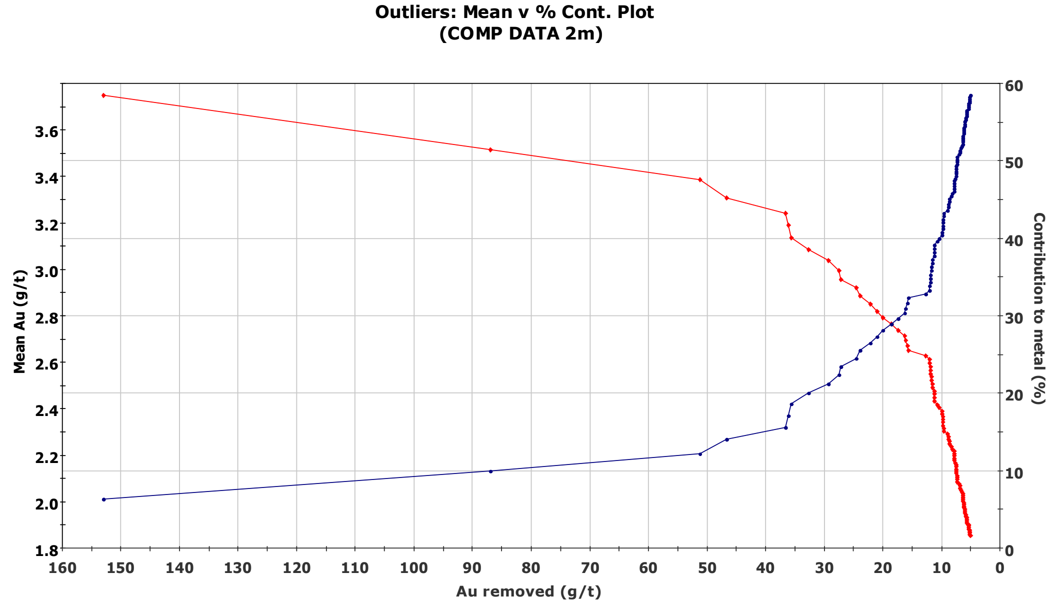
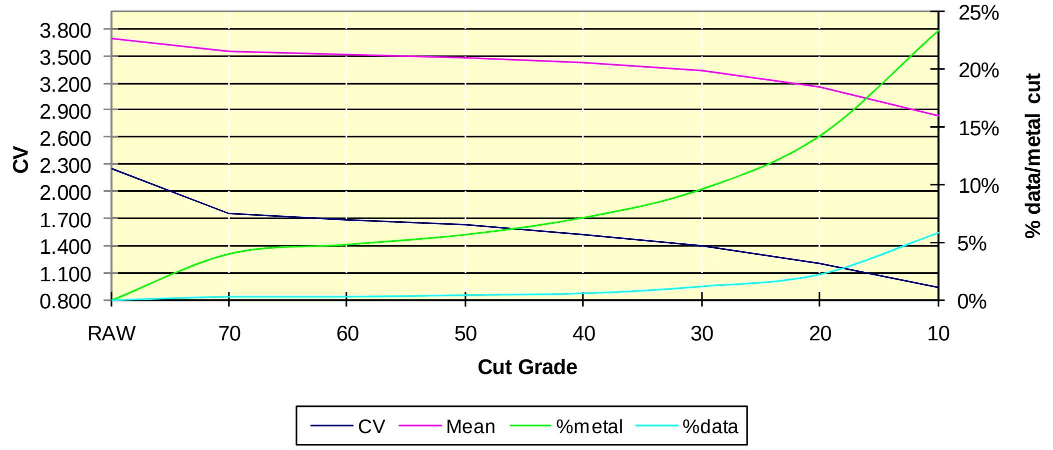


Figure 17.1.3_5
The Effect of Top Cutting at Various Grades



17.1.4 Variography

<u>Introduction</u>

Geostatistics has two primary objectives:

- To mathematically determine the variability relationship between two points in space, measuring the zone of influence and the degree of variability compared to a homogeneous field.
- To establish spatial modelling of a regional variable distribution.

Variography is used to describe the spatial variability or correlation of an attribute (e.g. Au). The spatial variability is traditionally measured by means of a variogram, which is generated by determining the averaged squared difference of data points at a nominated distance (h), or lag. The averaged squared difference (variogram or γ(h)) for each lag distance is plotted on a bivariate plot, where the X-axis is the lag distance and the Y-axis represents the average squared differences (γ(h)) for the nominated lag distance. The term variogram will be used as a generic term to describe all spatial measures in this document.

Variography

The variography was generated by Coffey Mining using Gemcom Surpac mining software. The final variograms and variogram models used for nugget variance and major, semi-major and minor axis calculation, for the seven mineralized zones are displayed in Figure 17.1.4_1.

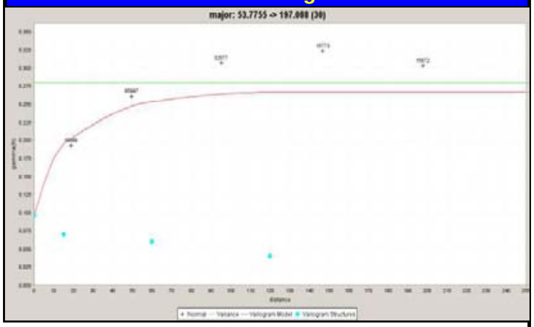
Figure 17.1.4_1
Ollachea Mineralized Zones Variograms

Target	Ollachea
Ore	Mineralization
Variable	Au

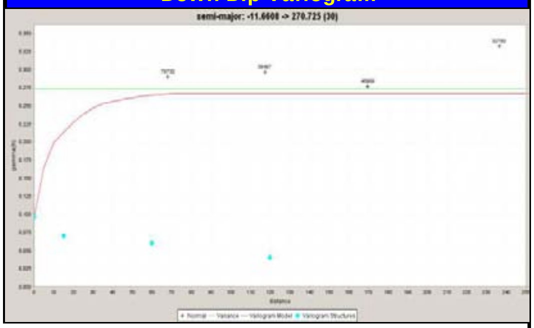
Variogram Structure		
Model	Spherical	
Structure	Sill	Range
Nugget	0.096	-
1	0.070	15
2	0.060	60
3	0.040	120

Elipsoide Orientation	
Bearing	197
Plunge	54
Dip	10
Maior/Semi-maior	1.7
Maior/Menor	4.6

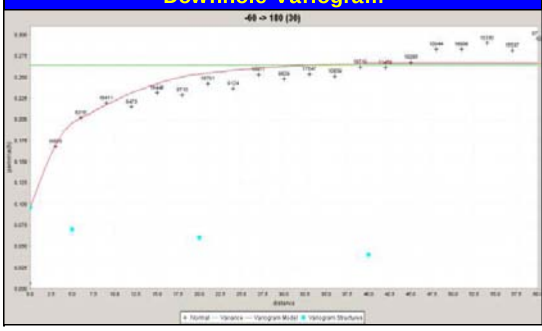
Strike Variogram



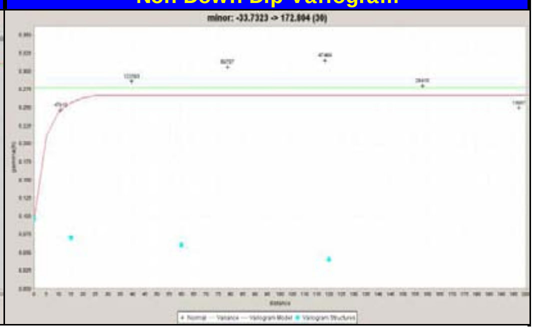
Down Dip Variogram



Downhole Variogram



Non Down Dip Variogram



A summary of the findings by Coffey Mining are listed below; Table 17.1.4_1 summarizes the variogram parameters used for the mineralized zones.

- Downhole variography showed a nugget of approximately 36% of the total sill.
- Major continuity was determined to be towards 197º with 10º dip and 54º plunge (Gemcom Surpac Software Rotation).
- Three spherical schemes were used to model the experimental directional variograms; overall range was 15m in the major direction, 8.8m in the semi-major direction and 3.3m in the minor direction.

Table 17.1.4_1
Ollachea Project
Summary Variogram Parameters for Mineralized Zones

Area	Orientation*			Co	C1	Range 1 (m)			C2	Range 2 (m)			C3	Range 3 (m)		
	Bearing	Plunge	Dip			Major	Semi-Major	Minor		Major	Semi-Major	Minor		Major	Semi-Major	Minor
Mineralized Zones	197º	54º	10º	0.096	0.07	15.0	8.8	3.3	0.06	60.0	35.3	13.0	0.04	120.0	70.6	26.1

*Gemcom Surpac Rotation Method

17.1.5 Cross Validation

The technique of cross validation was used to validate modelled variograms and to elaborate a Kriging plan for the estimation.

The cross validation technique consists of an estimation of the samples of the composite using self batch of samples. During the estimate of a sample, its analytical value is not considered in the estimate of the self value.

After the self values estimation, the technique of OK compares the estimated value to the analytical sample values (“real”).

Four tests for each two passes were realised varying the minimum number of samples. The objective was to select the test that represented better values between the estimated and real. The numbers of samples used in the tests T1, T2, T3 and T4 are presented in Table 17.1.5_1.

Table 17.1.5_1
Ollachea Project
Cross Validation Summary for Ollachea Mineralized Zones

Pass	Minimum Samples	Maximum Samples	Research Type
1	3	10	Ellipsoid
2	5	16	Ellipsoid

Test	Horizontal Range	Vertical Range
T1	30	15
T2	45	15
T3	60	15
T4	120	15

The Au (g/t) variable was submitted to cross validation, considering 4 search ellipses with different ranges. For each search ellipses, the three tests were completed above. The use of search ellipses was used to test a possible plan of Kriging to estimate the resources model of the target (Table 17.1.5_2 - multiple search ellipses used in the crossed validation, as the definitions of ellipsoid of search). Figure 17.1.5_1 presents the result of the validation for Pass 1 and Test 1, verified as better result of the Kriging test.

Table 17.1.5_2
Ollachea Project
Au (g/t) Cross Validation Results

	Class	Test 1	Test 2	Test 3	Test 4
Pass 1 Results	Linear Correlation	0.1232	0.2493	0.2444	0.2438
	Interceptation Constant	3.4288	3.0207	3.0211	2.9856
	Correlation Coefficient	0.0754	0.1255	0.121	0.1178
	R2	0.0057	0.0157	0.0146	0.0139
Pass 2 Results	Linear Correlation	0.3975	0.1374	0.2197	0.2298
	Interceptation Constant	3.1457	3.4752	3.2148	3.1336
	Correlation Coefficient	0.1292	0.0658	0.1044	0.1071
	R2	0.0167	0.0043	0.0109	0.0115

17.1.6 Block Model Development and Estimation

Block models were generated using the Gemcom mining software package. A parent block size of 20mE x 30mN x 4mRL was selected with sub-blocking to a 2.5mE x 3.75mN x 0.5mRL cell size to improve volume representation of the interpreted wireframe models. The model is defined in Table 17.1.6_1.

Each block was characterized by a series of attributes, as described in the Table 17.1.6_2.

For the grade estimates, Au was interpolated using OK techniques. For the purposes of this report, only the OK Au estimation is considered appropriate and suitable.

OK is one of the more common geostatistical methods for grade estimation of the block. In this interpolation technique, the contributing composited samples are identified through a research applied from the centre of each block. The weights are determined to minimise the variance error, considering the space localization of the selected composites and the modelled variogram. Variography describes the correlation between samples composited in function of distance and the direction. The grade of the weighted composited sample is combined to generate the estimative of the block and the variance.

The established Kriging plan considered 4 steps of estimate, each one relative to the precision degree of the same one, resulted from the Cross Validation, as presented in the Table 17.1.6_3.

Figure 17.1.5_
Cross Validation Summary – Pass 1, Test 1

Table 17.1.6_1
Ollachea Project
Block Model Parameters – Ollachea Deposit

	East	North	Elevation
Minimum Coordinates	338,740	8,474,285	2,400
Maximum Coordinates	339,980	8,474,795	3,600
Parent Block size (m)	20	30	4
Sub-Block Size (m)	2.5	3.75	0.5

Table 17.1.6_2 Ollachea Project Block Model Attribute List		
Attribute Name	**Type**	**Description**
au_da	Real	Average anisotropic distance to samples
au_dn	Real	Anisotropic distance to nearest sample
au_kv	Real	Kriging variance
au_nn	Real	Blocks of au ppm nearest neighbour
au_ns	Integer	Number of samples
grade_au	Real	Blocks of au ppm grade kriged
ore	Integer	0=air, 1=waste, 2 to 8=mineralized zones
resource	Integer	1=measured, 2=indicated, 3=inferred
sg	Real	Specific gravity = 2.80g/cm³ (mineralized zone)
step_au	Integer	Pass of au estimation

Table 17.1.6_3 Ollachea Project Ordinary Kriging Strategy					
Mineralization Type	**Step**	**Minimum Search Distance (m)**	**Search**	**Minimum Sample Numbers**	**Maximum Sample Numbers**
Mineralized Zone	1	30	Ellipsoid	3	10
	2	60	Ellipsoid	3	10
	3	120	Ellipsoid	3	10
	4	10,000	Ellipsoid	1	10
Ellipsoid Orientation (Gemcom Surpac Rotation):			Bearing 197º; Plunge 54º; Dip 10º		
Anisotropy Factors:			Major/Semi-Major 1.7; Major/Minor 4.6		

The estimated blocks in the Project were restricted to parent cells and sub-cells which were both below the topographic surface and within the mineralized zones wireframes. Figure 17.1.6_1 shows the comparison between the ellipsoid used to run the mineral resource estimation and the seven mineralized zones modelled.

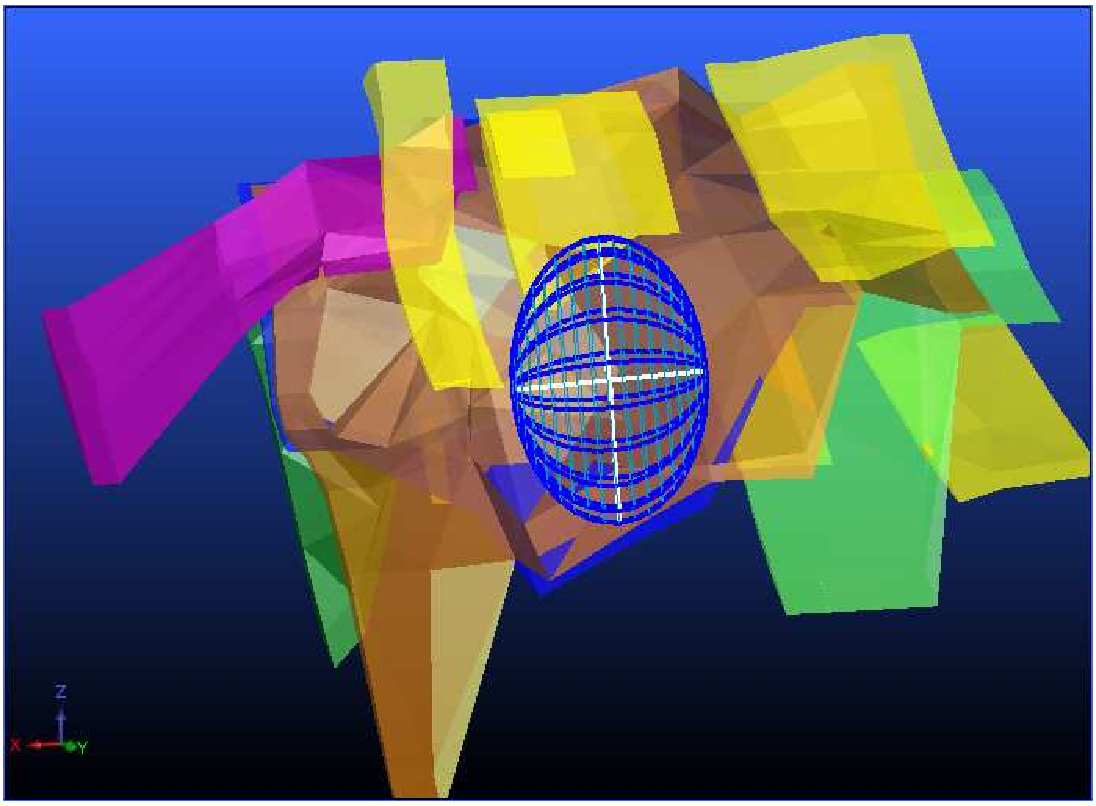
17.1.7 Model Validation

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. Table 17.1.7_1 summarizes this comparison.

Table 17.1.7_1 indicates that the adherence of the block model to solids is very good.

Figure 17.1.6_1
Comparison Between the Ellipsoid and the Mineralized Zones Modelled



(view to southwest)

Table 17.1.7_1
Ollachea Project
Volume Comparison
Geological Model x Block Model

Mineralized Zone	Solids Vol. (m³)	Block M. Vol. (m³)	Solids/Blocks Vol. (%)
2	2,117,663	2,126,319	99.59%
3	1,035,263	1,029,781	100.53%
4	342,188	341,639	100.16%
5	130,463	132,564	98.42%
6	375,863	377,994	99.44%
7	563,213	569,736	98.86%
8	291,450	289,594	100.64%
Total	**4,856,100**	**4,867,627**	**99.76%**

Block Model Sections

Sections for visual validation of the compatibility of solids with the block model had been generated. This validation mainly aims to verify the adherence to the volumes of solids and the block model. It is possible to notice from sections, a good adherence to the block model and the modelled sections.

Nearest Neighbour Check

The technique of the “Nearest Neighbour” was used to validate the Ordinary Kriging estimate.

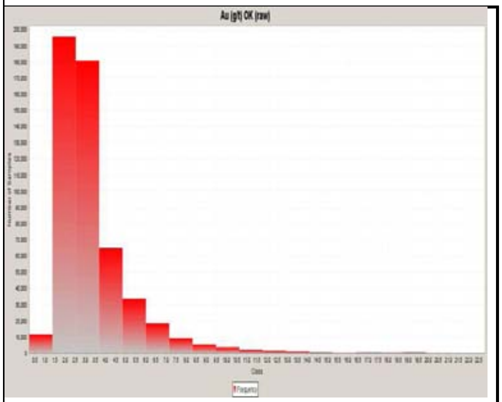
The grade comparison was completed for the total resource. The Au (g/t) variable was analyzed.

The validation was carried out by histogram comparison of the results for “Nearest Neighbour” estimation technique. The dispersion and Quantil-Quantil graphs had been created to verify the occurrence of bias and the softening of the estimate. Figure 17.1.7_1 presents the comparison of Au grades (g/t). The Q-Q Plot shows an accepted correlation.

Figure 17.1.7_1
NN-Check Analysis

Target	Total
Lithology	Ore
Variable	Au (g/t) Top Cut
Resource	Total

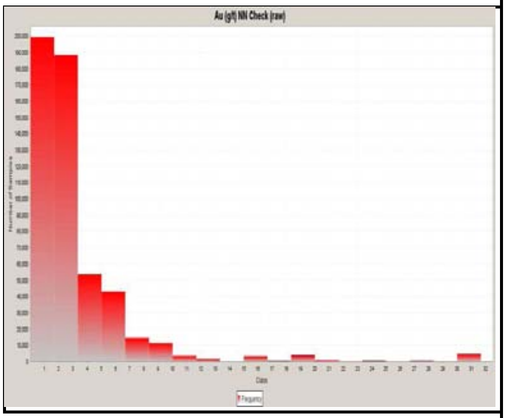
Ordinary Kriging



Samples:	**529552**
Minimum:	**0.02**
Maximum:	**30.00**
Mean:	**3.27**
Variance:	**15.56**
Std. Deviation:	**3.94**

Quantiles	
2.5%	**0.32**
1°Quartil:	**1.34**
Median:	**2.12**
3°Quartil:	**3.53**
97.5%:	**16.04**

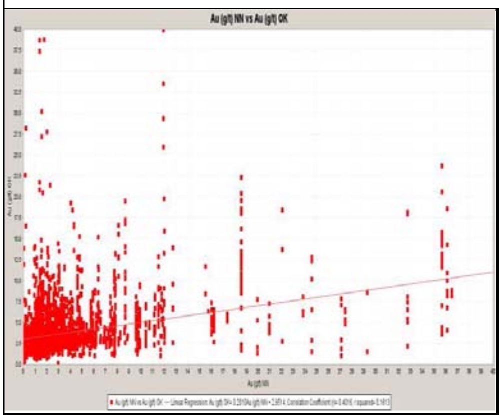
NN-Check



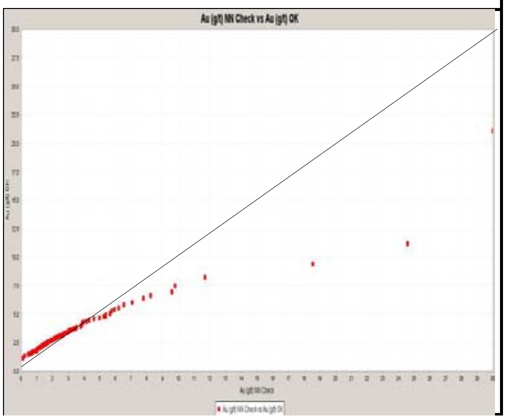
Samples:	**529552**
Minimum:	**0.21**
Maximum:	**21.06**
Mean:	**3.38**
Variance:	**3.89**
Std. Deviation:	**1.97**

Quantiles	
2.5%	**1.42**
1°Quartil:	**2.22**
Median:	**2.82**
3°Quartil:	**3.83**
97.5%:	**8.78**

Correlation



Q-Q Plot



To complete the NN-Check validation, Coffey Mining plots some graphics with the OK and NN-Check Au (g/t) grade spatial comparison (x, y and z). Figure 17.1.7_2 to Figure 17.1.7_4 below show a good global correlation.

Figure 17.1.7_2
X Coordinate Au (g/t) Validation

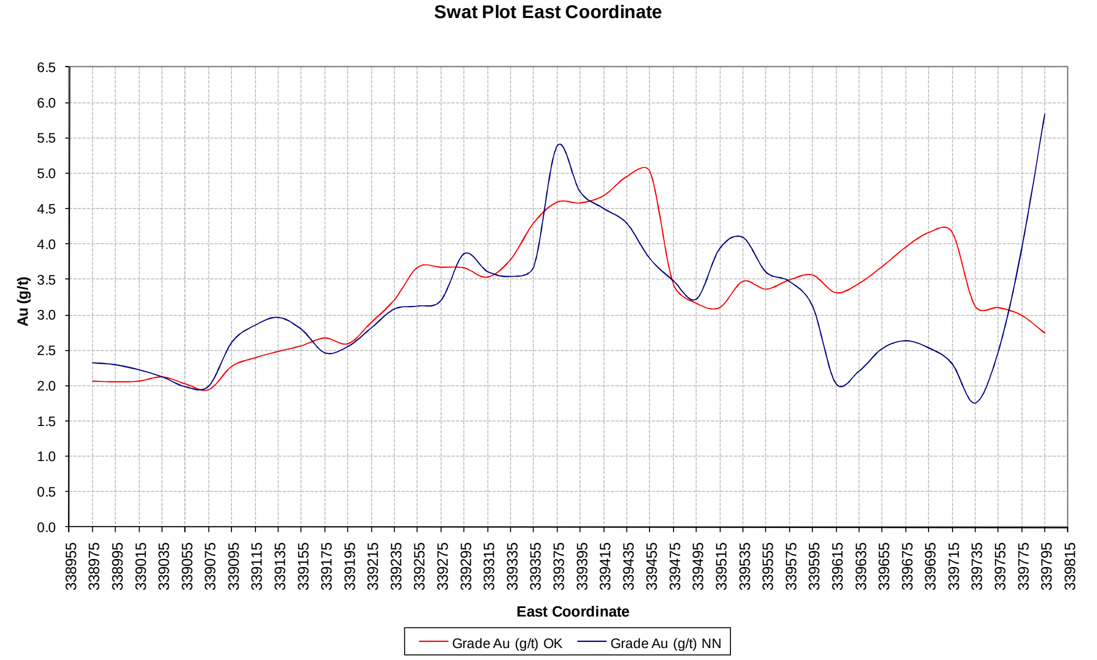


Figure 17.1.7_3
Y Coordinate Au (g/t) Validation

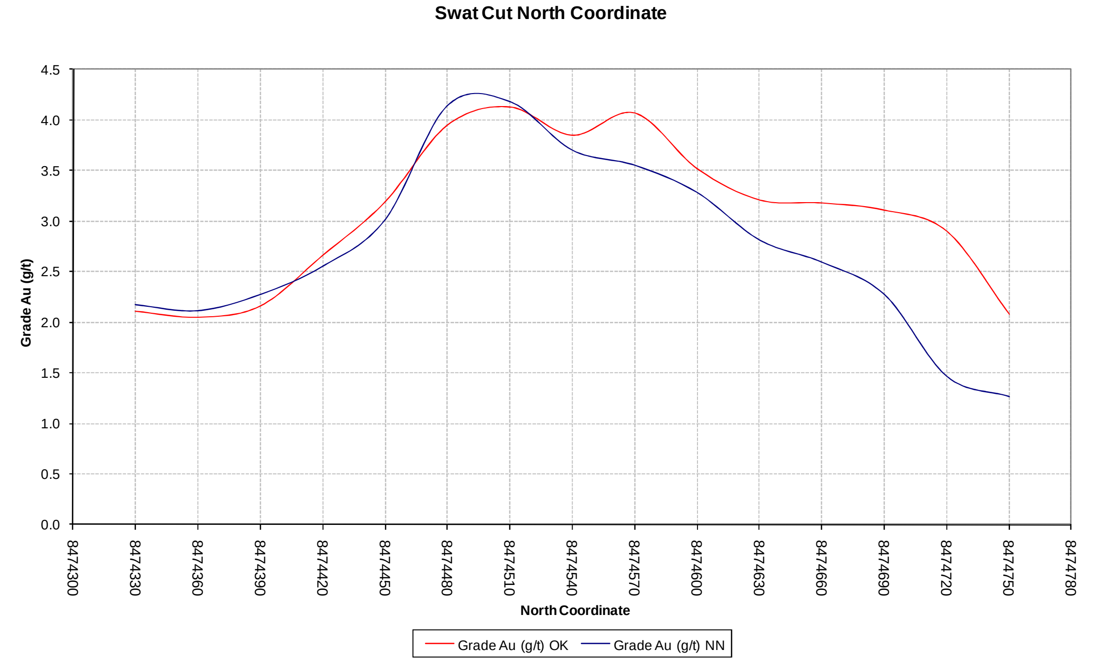
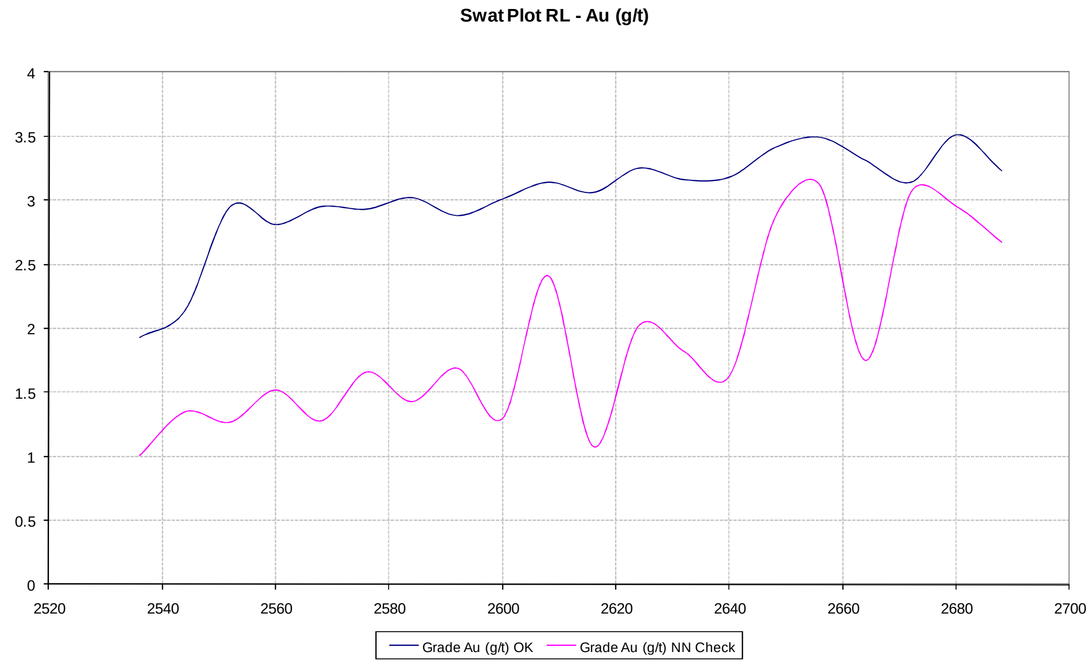


Figure 17.1.7_4
Z Coordinate Au (g/t) Validation



17.1.8 Resource Classification

The grade estimates for the Project has been classified as an Inferred Mineral Resource, in accordance with NI 43-101 and the CIM standard, based on the confidence levels of the key criteria that were considered during the resource estimation. Key criteria are tabulated in Table 17.1.8_1.

A summary of the estimated resource for the Ollachea deposit is provided in Table 17.1.8_2 below.

Figure 17.1.8_1 shows the gold grade distribution of the block model.

Figure 17.1.8_2 shows the grade-tonnage curve for the Inferred Resources.

This resource estimate was prepared by Bernardo Viana, Resource Consultant with 8 years of Resource Estimation and Exploration geological experience. Bernardo Viana is a member of the Australian Institute of Geoscientists ("MAIG"). The certificate of qualified person under the NI 43-101 is presented in Section 23 of this report.

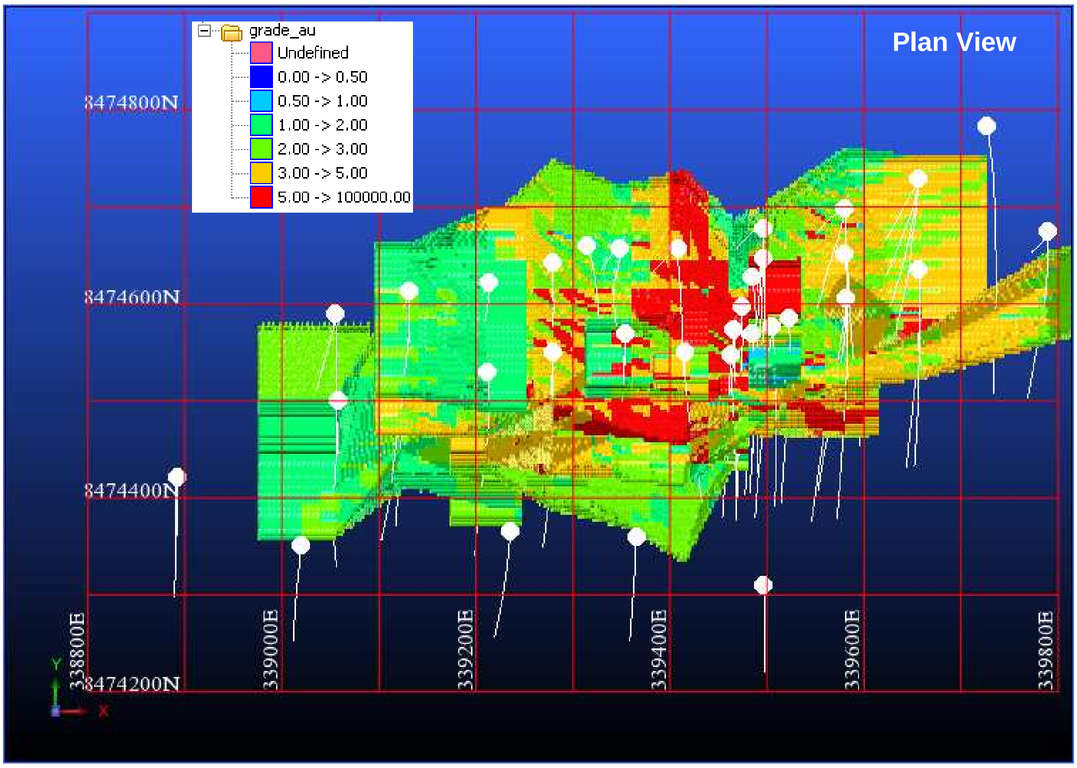
Table 17.1.8_1
Ollachea Project
Confidence Levels of Key Criteria

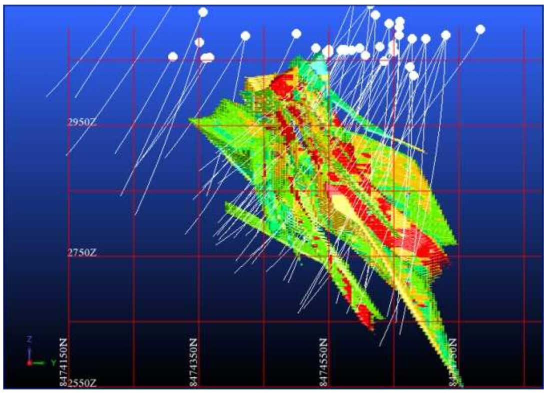
Items	Discussion	Confidence
Drilling Techniques	Diamond drilling is Industry standard approach.	High
Logging	Standard nomenclature and apparent high quality.	High
Drill Sample Recovery	Good recovery recorded except in shear/fault zones.	High
Sub-sampling Techniques & Sample Preparation	The 2m sampling method is not recommended for this style of gold. The poor precision of field duplicates reflects this. Meter by meter sampling is a more accepted practice. This has been implemented	Moderate
Quality of Assay Data	Available field duplicate data shows that the precision of assaying is inconclusive. Standards have failed badly in earlier drilling also	Moderate to Low
Verification of Sampling and Assaying	Umpire samples have been taken but not available at the time of this report	N/A
Location of Sampling Points	Survey of all collars with downhole survey completed for most holes.	Moderate to high
Data Density and Distribution	Approximately 60m x 60m spaced drilling in central zone has provided adequate data for an inferred resource. Infill to 30 X 30m will be required to increase the confidence of the current interpretation.	Moderate
Audits or Reviews	Barry Smee report has indicated issues with assay precision as noted above	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. Additional zones are expected but additional drilling is required to improve the confidence. The current 3D model is restricted to the high grade zones. The low grade zones must be further defined.	Moderate-Low
Estimation and Modelling Techniques	Ordinary Kriging has been used to obtain estimates of Au g/t grade. Coffey Mining used 4 steps for all blocks: • Step 1 – 30m range ellipsoid; • Step 2 – 60m range ellipsoid; • Step 3 – 120m range using ellipsoid; • Step 4 – 10,000m range using ellipsoid.	High
Cutoff Grades	A Cutoff Grade of 1g/t Au was used to define the high grade envelopes. There is no low grade cutoff or interpretation.	Moderate
Mining Factors or Assumptions	20mE by 30mN by 4mRL	High

Table 17.1.8_2
Ollachea Project
Grade Tonnage Report – Mineral Resource (as at 6th October 2009)
Ordinary Kriging Estimate
20mE x 30mN x 4mRL Selective Mining Unit

	Lower Cutoff Grade (g/t Au)	Million Tonnes	Average Grade (g/t Au)	Contained Gold (Kozs)
Inferred Mineral Resource	0.0	13.6	3.6	1,574
	0.5	13.6	3.59	1,574
	1.0	13.5	3.62	1,571
	2.0	11.4	3.98	1,456
	2.5	8.9	4.50	1,277
	3.0	6.5	5.06	1,067
	5.0	2.1	7.81	531

Figure 17.1.8_1
Block Model Gold Grade Distribution





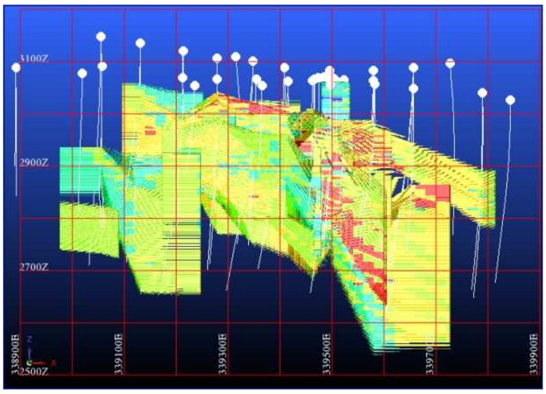
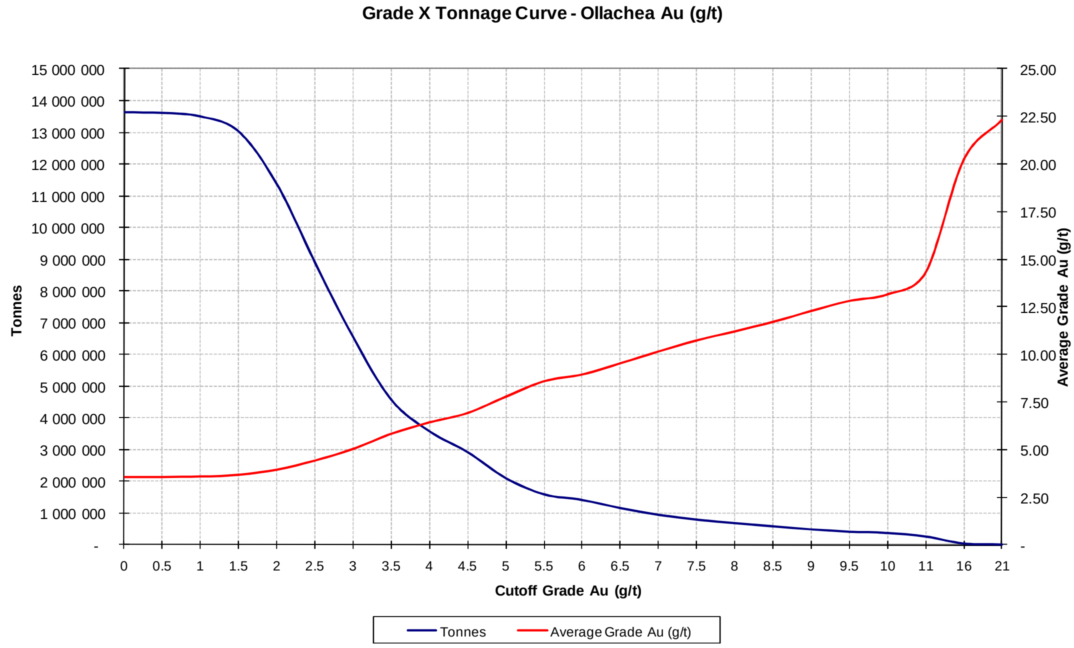


Figure 17.1.8_3
Grade-Tonnage Curve for Ollachea Project Inferred Resources



17.2 Mineral Reserves – Mining Inventory

There are no Mineral Reserves which can be disclosed from the Inferred Resources presented in Section 17.1.8. Nonetheless, as part of a preliminary assessment, a scoping study (the Study) was completed by Coffey Mining and the mining inventory was estimated to be 8.2Mt at 4g/t Au head grade for a possible recoverable production of approximately 1.0Moz as presented in Table 17.2_1.

Table 17.2_1
Ollachea Gold Project
Mining Inventory Estimate

Items	Value
Inferred Resource tonnes	8.9M
Inferred Resource grade @ 2.5g/t Cutoff Grade (Au)	4.5
Inferred Resources Ounces	1.3M
Mining Recovery	80%
Dilution	15%
Mining Inventory	8.2M
Dilution grade	0.9
Head grade (Au)	4.0
Ounces supplied to plant (Au)	1.1M
Recovery	91%
Recoverable Ounces (Au)	1.0M

It must be noted that the Study is preliminary in nature, it includes solely Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary assessment as estimated in the Study will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

18 OTHER RELEVANT DATA AND INFORMATION

18.1 Introduction

A preliminary economic assessment was undertaken by Coffey Mining on behalf of MKK. As part of the Study the following items were assessed:

- Geotechnical conditions
- Mining method and backfill system;
- Metallurgy and processing flow sheet;
- Tailings facilities;
- Cost estimations and financial analysis;

It must be noted that the Study is preliminary in nature, it includes solely Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary assessment as estimated in the Study will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

18.2 Geotechnical Input and Conditions

18.2.1 Introduction

As part of the Study, it was required to provide an assessment of the geotechnical aspects of the project. This preliminary assessment included:

- A brief discussion of the geological setting;
- An assessment of the probable mining methods and underground access arrangements;
- An assessment of rock mass conditions;
- Determination of the parameters relevant to the underground stope design, i.e. the stable stope dimensions, both supported and unsupported;
- The conditions and support requirements of the access drive; and
- Recommendations for data collection to support the ongoing mining studies.

The information utilised in the formulation of this report has been obtained from the following sources:

- Core photos and logs, geological maps, and sections as obtained by Coffey Mining during a brief site visit undertaken in 2009;
- The Structural Study of the Ollachea District, Puno, Peru, Field report 2009 by Telluris Consulting;
- Primary resource model of the Ollachea mineralized zone with topographic map; and
- Further information supplied by MKK during the study.

18.2.2 Geological Setting

Gold mineralization occurs within the intensely foliated black slates as quartz vein/ veinlet-hosted auriferous mineralization.

The observed black slates were weak to medium in strength. There is no distinct difference between the mineralized zone and non-mineralized zones as illustrated in Figure 18.2.1_1.

Figure 18.2.1_1
Mineralized and Un-Mineralized Core Photos

Un-mineralized Hanging all

Mineralized

Un-mineralized Footwall

18.2.3 Objective

Given the mineralized zone location and probable dimensions, the aim of the Study was to evaluate the geotechnical aspects relevant to underground extraction at a scoping study level.

Due to the location, the deposit dimensions, and the deposit orientation, the assessment is based on:

- Assumed extraction by open stopes with filling and possible use of cut and fill extraction as required by the deposit shape or ground conditions; and
- Access arrangements to suit such an approach and the interaction with the topography and the processing plant location.

18.2.4 Rock Mass Conditions

The rock mass classification has been conducted by applying the Modified Tunnelling Quality Index (Q'). Q' has been calculated by applying logging information and the information collected by Coffey Mining during the site visit as outlined in the following sections of this report.

<u>Rock Quality Designation</u>

At the time of the geotechnical preliminary assessment, Rock Quality Designation ("RQD") was available from 46 drillholes drilled within the deposit. The average RQD values for each borehole are given in Figure 18.2.4_1 and the RQD distribution from these boreholes is presented in Figure 18.2.4_2.

As the main structures are parallel or sub parallel to the mineralized zone and as not all of the holes were drilled perpendicular to it, the drillholes will be striking the structures at an oblique angle, resulting in apparent spacing of structures being much wider and RQD values being much higher than the actual occurrence. For this reason, the initial assessment of ground conditions was based on the lower quartile (nominal conservative design value) of the RQD values.

A summary of the statistical parameters are presented in Table 18.2.4_1.

The weighted RQD distribution by the core length, as shown in Figure 18.2.4_2, indicates that about 25% of the measured core length has RQD value less than 10 percent - a 'very poor' quality rock. The low RQD values are assumed to be due to intensely foliated and weakly convoluted rock structure.

Figure 18.2.4_1
Average RQD of Holes

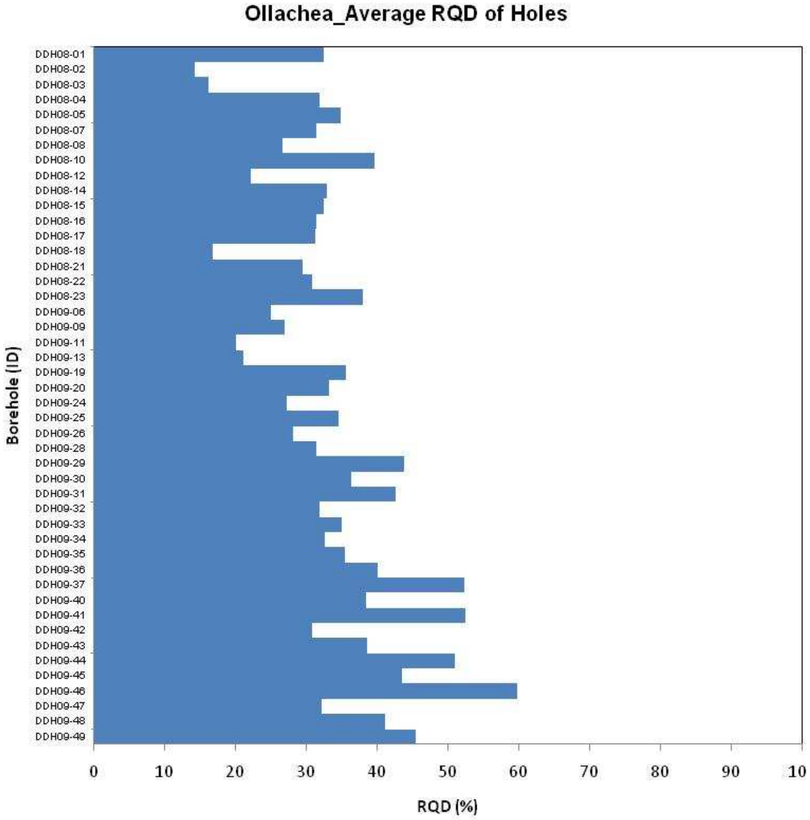


Figure 18.2.4_2
RQD Distribution Graph Weighted by Length Holes

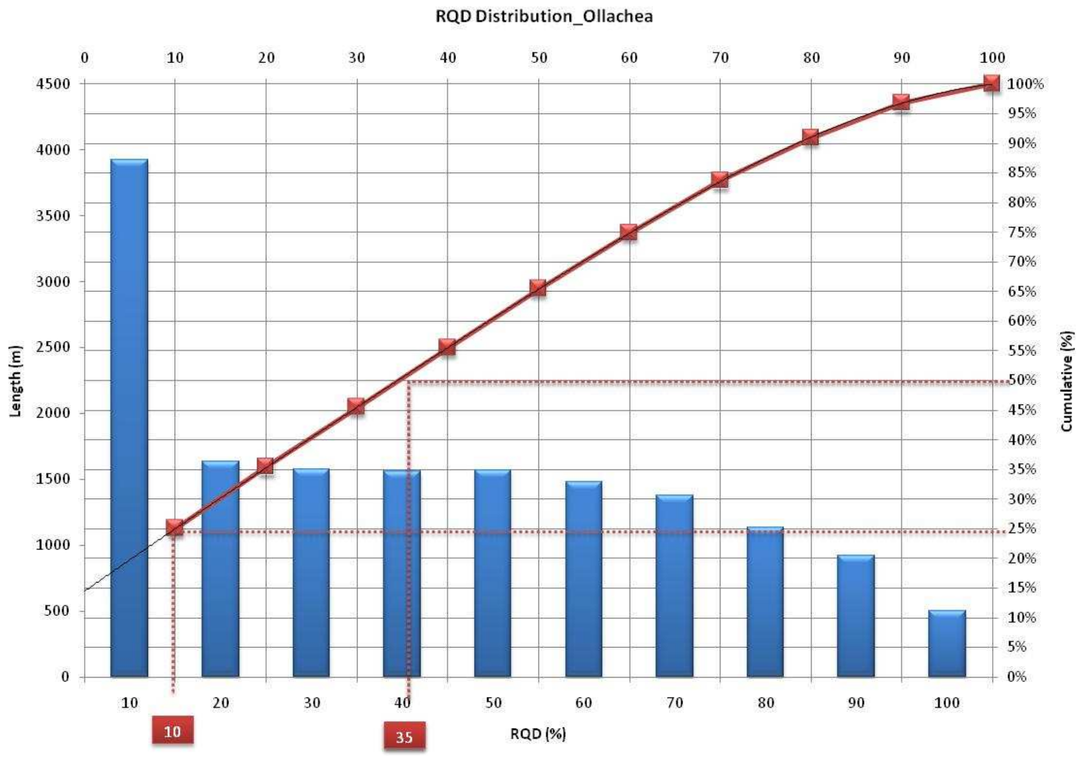


Table 18.2.4_1
Ollachea Gold Project
Statistical Summary of RQD (weighted by length)

RQD (25%ile)	10
RQD (Mean,50%ile)	35
RQD (75%ile)	60
Mode	0
Median	29
Std Dev	23

Structural Components

As there was no core logging information on the measured oriented structures, the main joint sets in the vicinity of the mineralized zones were estimated from the geological section in Figure 18.2.4_1, from the overall arrangement as shown in Figure 18.2.4_3, and from the observations made during the site investigations. The core is illustrated in surface joint exposures in Figure 18.2.1_1 and in underground exposures as shown in Figure 18.2.4_5. For the purpose of the Study, the relevant groups of the structures that were identified include:

- Foliation (Joint Set 1)

  Foliation is considered to be the main limitation on open stope stability due to its orientation being parallel to ore with average 40 degrees dip.

- Sub-Vertical Joints (Joint set 2)

  Joint set 2 is also considered to have some impact on the stability of the stopes, due to the apparent orientation of its dip and its strike that is about perpendicular to the foliation plane.

- Random Joints

  Additionally, random joints are also apparent.

Table 18.2.4_2 summarizes the estimated orientation of the recorded discontinuity sets.

Table 18.2.4_2
Ollachea Gold Project
Average Plane Orientation of Discontinuity Sets

Structure Type	Strike	Dip
Foliation (Joint Set 1)	270	40
Sub-Vertical (Joint Set 2)	000	70
Random Fractures	180	40

Major Structures

Based on the available geological section as shown in Figure 18.2.4_3, major faults appear to be following the foliation orientation, which is considered in its own right, as one of the main structural components for the purpose of this study.

Figure 18.2.4_3
Geological Cross Section through Ollachea Area Showing Structures Parallel to Mineralization

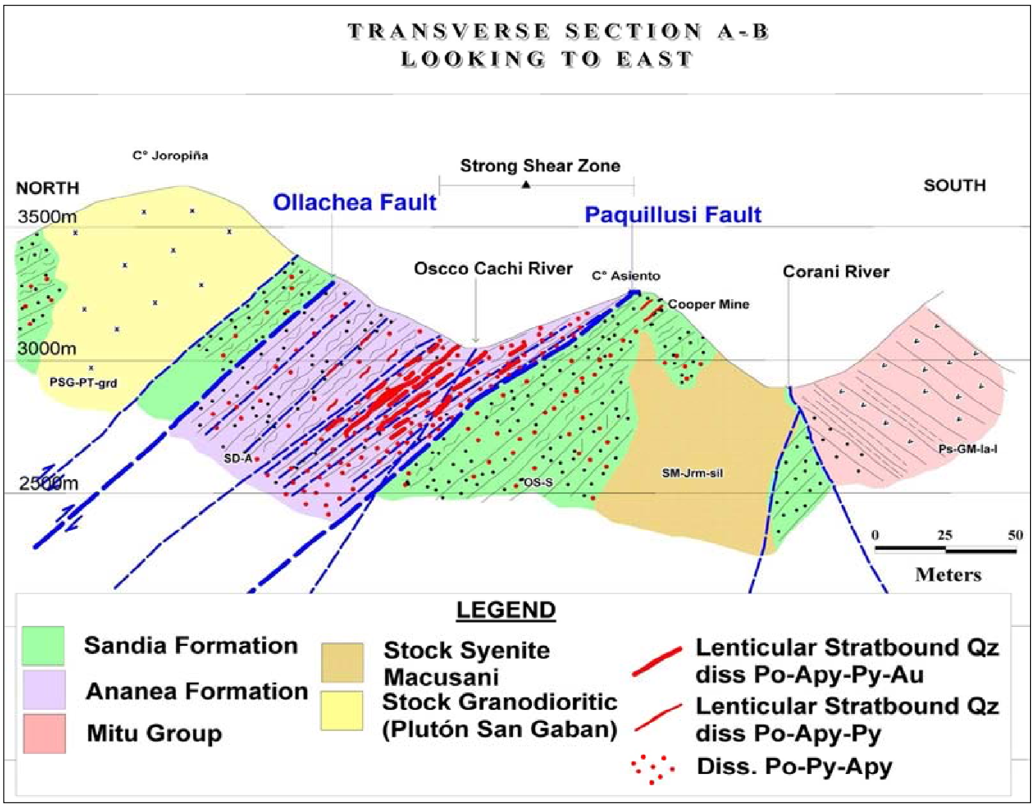


Figure 18.2.4_4
Observed Joints on the Borehole Logs

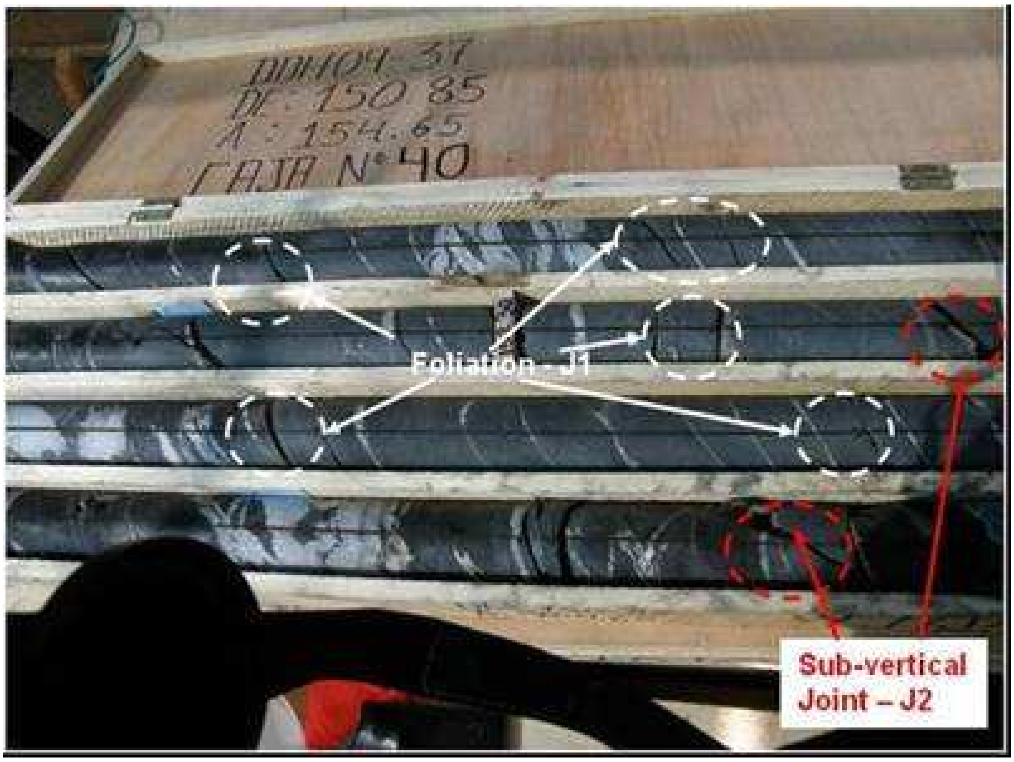
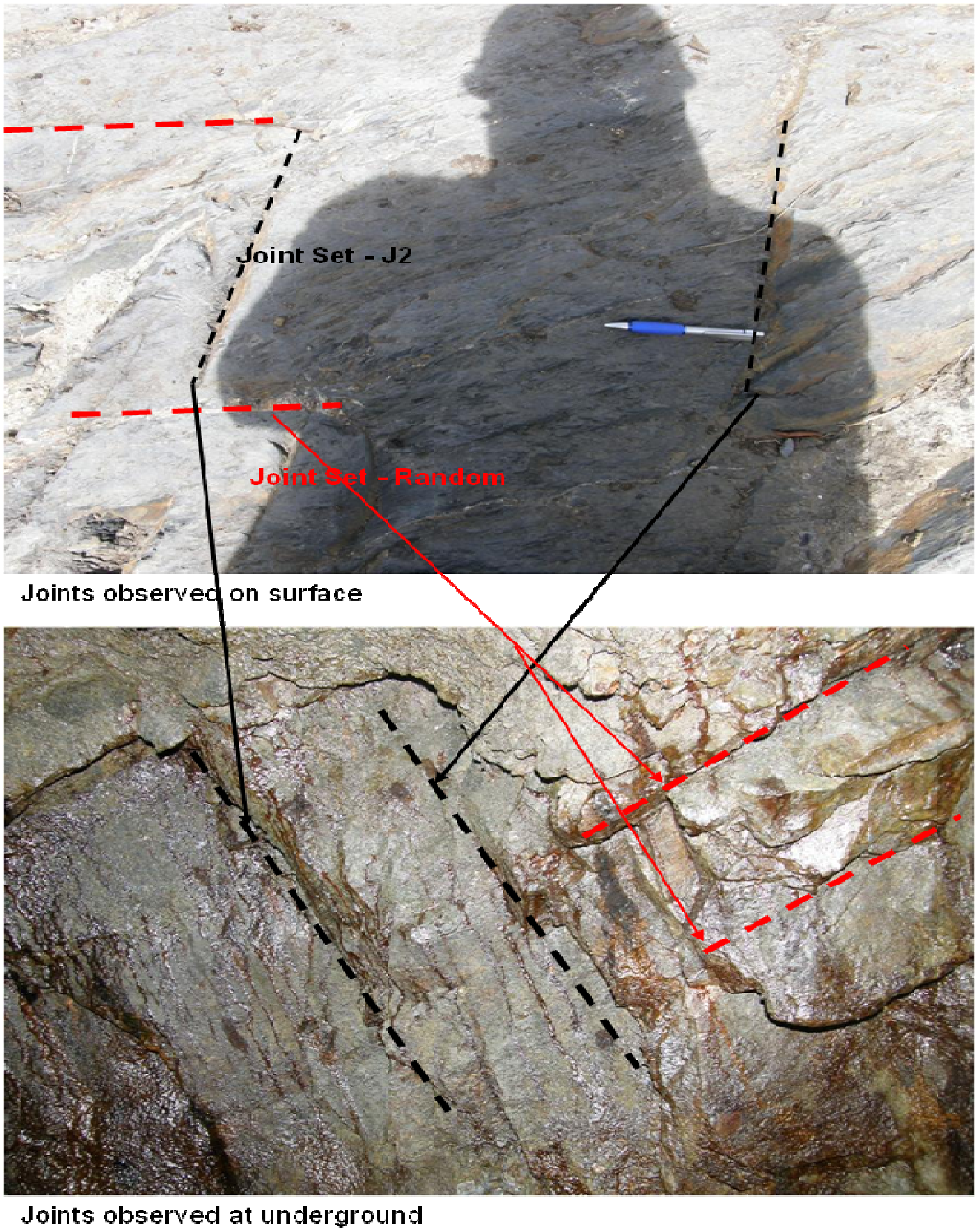


Figure 18.2.4_5
Observed Joints during the Site Visit



Intact Rock Properties

There is no specific information on the intact rock properties. Based on the literature review, the Uniaxial Compressive Strength (UCS) of slate was assumed at 50MPa (UCS for slate varies from 20 to 120MPa, as presented in 'Foundations of Engineering Geology, 2nd ed. Tony Waltham, 2002).

In-situ Stress Field

No direct information of the stress environment at Ollachea is currently available. It is in steep and variable terrain with major decreases in elevation to its north. Hence, for the purpose of this level of study, a stress ratio of one was considered. It can be noted that due to the relatively shallow depth of the mineralization, it is not likely that stress will be an issue

Rock Mass Classification

Coffey Mining utilised the Modified Rock Quality Index (Q') to classify the in-situ rock mass at Ollachea. The Q index is:

$$Q' = \frac{\mathrm{RQD}}{\mathrm{J_n}} \mathrm{x} \frac{\mathrm{J_r}}{\mathrm{J_a}}$$

where:

- RQD – Rock Quality Designation;
- Jn – Joint Number;
- Jr – Joint Roughness Condition; and
- Ja – Joint Alteration Condition.

The rock mass parameters were estimated from the available logging data and by examining the core photos. Due to the relatively scattered and inadequate geotechnical data available on the rock mass and discontinuity features, the parameters Jr, Ja, etc had to be assumed. The assumptions made on the derivation of these factors are as follows:

- **Jn** – due to the absence of measured oriented structures, the number of the effective joint sets in vicinity of the mineralized zone was estimated by investigating core photos and information collected from the site visit by Coffey Mining personnel.
  - As per the structural analysis, there are two dominant joint sets with some random features. Hence a Jn value of 6 was chosen as is shown in Table 18.2.4_3.
- The joint condition values of Jr and Ja were assessed based on available logs and core photographs.
  - **Jr** – A value of 2 was assumed, corresponding to an undulating smooth surface as shown in Table 18.2.4_4.
  - **Ja** – A value of 6 was assumed, corresponding to 1mm to 5mm joint separation and non-softening, slightly clayey non-cohesive filling as shown in Table 18.2.4_5.
- **Calculation of Q'**
  - Using the parameters described above in relation to joint conditions within Ollachea, Coffey Mining calculated the following Q' value based on the 25th and 50th percentiles RQD value (Refer to Table 18.2.4_1). The calculated Q' values are tabulated in Table 18.2.4_4.

The values used for the study have been formatted in bold.

Table 18.2.4_3
Ollachea Gold Project
Jn Value

Number of Joint Sets	**Joint Set No. J_n**
Intact, no or few joints	0.5 — 1.0
One joint set	2
One joint set plus random joints	3
Two joint sets	4
Two joint sets plus random joints	**6**
Three joint sets	9
Three joint sets plus random joints	12
Four or more joint sets, random, heavily jointed, sugar cube, etc.	15
Crushed rock, earthlike	20

Table 18.2.4_4
Ollachea Gold Project
Jr Value

Description of Joint Surface Roughness	**Discontinuous**	**Undulating**	**Planar**
Rough	4	3	1.5
Smooth	3.0*	**2**	1
Slickensided	2.0*	1.5	0.5
Planes containing gouge thick enough to prevent rockwall contact	1.5*	1	1

Table 18.2.4_5
Ollachea Gold Project
Ja Value

Description of Gouge	**Joint Alteration Number Ja for Joint Separation (mm)**		
	<1.0[1]	**1.0-5.0[2]**	**>5.0[3]**
Tightly healed, hard, non-softening impermeable rock mineral filling	0.75	-	-
Unaltered joint walls, surface staining only	1	-	-
Slightly altered, non-softening, non-cohesive rock mineral or crushed rock filling	2	4	6
Non-softening, slightly clayey non-cohesive filling	3	**6.0***	10.0*
Non-softening strongly over-consolidated clay mineral filling, with or without crushed rock	3.0*	6.04	10
Softening or low friction clay mineral coatings and small quantities of swelling clays	4	8.0*	13.0*
Softening moderately over-consolidated clay mineral filling, with or without crushed rock	4.0*	8.04	13
Shattered or micro-shattered (swelling) clay gouge, with or without crushed rock	5.0*	10.04	18

Table 18.2.4_6
Ollachea Gold Project
Calculated Q' Values

	RQD @ 25%	**RQD @ 50%**
RQD	10	35
Jn	6	6
Jr	2	2
Ja	6	6
Q'	0.55	1.94

18.2.5 Mining Method

Stable Span Methodology

The specific underground mining method selected for the Study was sublevel stoping.

Coffey Mining used the stability graph method, after Potvin 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 dataset of self caving and stable underground mines.

N' = Q' x A x B x C

Where:

- A – Rock Stress Factor;
- B – Joint Orientation Factor; and
- C – Gravity Adjustment Factor.

Application

A – Rock Stress Factor

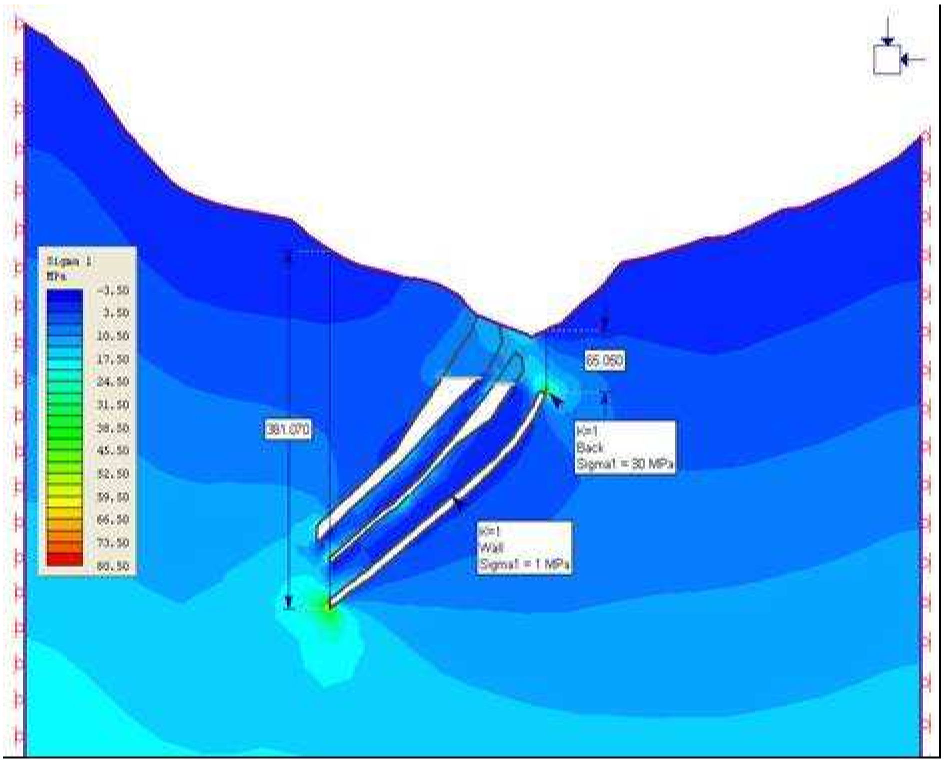
This factor is determined by calculating the ratio of the UCS to the maximum induced compressive stress. Its determination, in order to assess the extraction approach for this deposit, was based on 2D numerical modelling conducted using the software package *Phase*^{2} as is shown in Figure 18.2.5_1 and incorporating a design UCS of 50MPa.

The design values of the A Factor are estimated at 0.1 and 1.0 for the stope back and wall respectively as outlined in Table 18.2.5_1.

Table 18.2.5_1
Ollachea Gold Project
Evaluation of Rock Stress Factor A (UCS=50MPa)

Depth (m)	k=1					
	Back			Hanging wall		
	Sig1 (MPa)	UCS/Sig1	A	Sig1 (MPa)	UCS/Sig1	A
65m	30	1.67	**0.10**	1	50.00	**1.00**
380m	13	3.85	**0.31**	5	10.00	**1.00**

Figure 18.2.5_1
Phase 2D Modelling of Ollachea Mining



B – Joint Orientation Factor

The joint orientation factor is calculated by evaluating the relative differences in orientation between the major joint sets and their intersection with the back, the hanging wall, and the ends of the stopes. The B Factor values for different stope faces are summarized in Table 18.2.5_2.

Table 18.2.5_2
Ollachea Gold Project
Evaluation of Joint Orientation Adjustment Factor B

	Critical Joint Set	Difference in Strike (degree)	Difference in Dip (degree)	Factor B
Back	Fol	40	0	**0.4**
Hanging wall	Fol	0	0	**0.3**
Ends	J2	20	0	**0.2**

C – Gravity Adjustment Factor

The gravity adjustment factor 'C' is based on the most likely structural failure mechanisms i.e.:

- a gravity or slabbing fall; or
- by sliding.

Based on the mining method information, a "C" Factor was determined. The outcome of the assessment is as shown in Table 18.2.5_3.

Table 18.2.5_3
Ollachea Gold Project
Evaluation of Gravity Adjustment Factor C

	Critical Joint Set	Failure Mode	Inclination (degree)	Factor C
Back	Fol	Gravity	0	**2**
Hanging wall	Fol	Gravity/Slabbing	40	**3.4**
Ends	J2	Sliding	70	**4**

Calculation of Stability Number for Ollachea

Based on the calculated values of Q' and the A, B and C Factors, the stability number N' can be calculated for the stope faces for the Ollachea deposit.

The calculated N' values are given in Table 18.2.5_4 for the back, hanging walls and ends

N' = Q' x A x B x C

Table 18.2.5_4
Ollachea Gold Project
Calculated Stability Numbers for Stope Faces

	Depth (m)	Q'	B	C	A	N'
Back	65-380	0.55	0.4	2.0	0.1	0.04
Hanging wall	65-380	0.55	0.3	3.4	1	0.56
Ends	65-380	0.55	0.2	4.0	1	0.44

Maximum Unsupported and Supported Hydraulic Radius for Open Stoping

Coffey Mining used the Stability Graph Method, as presented by Potvin and Nickson (1992), for the Ollachea deposit to determine the hydraulic radius for this purpose for open stoping.

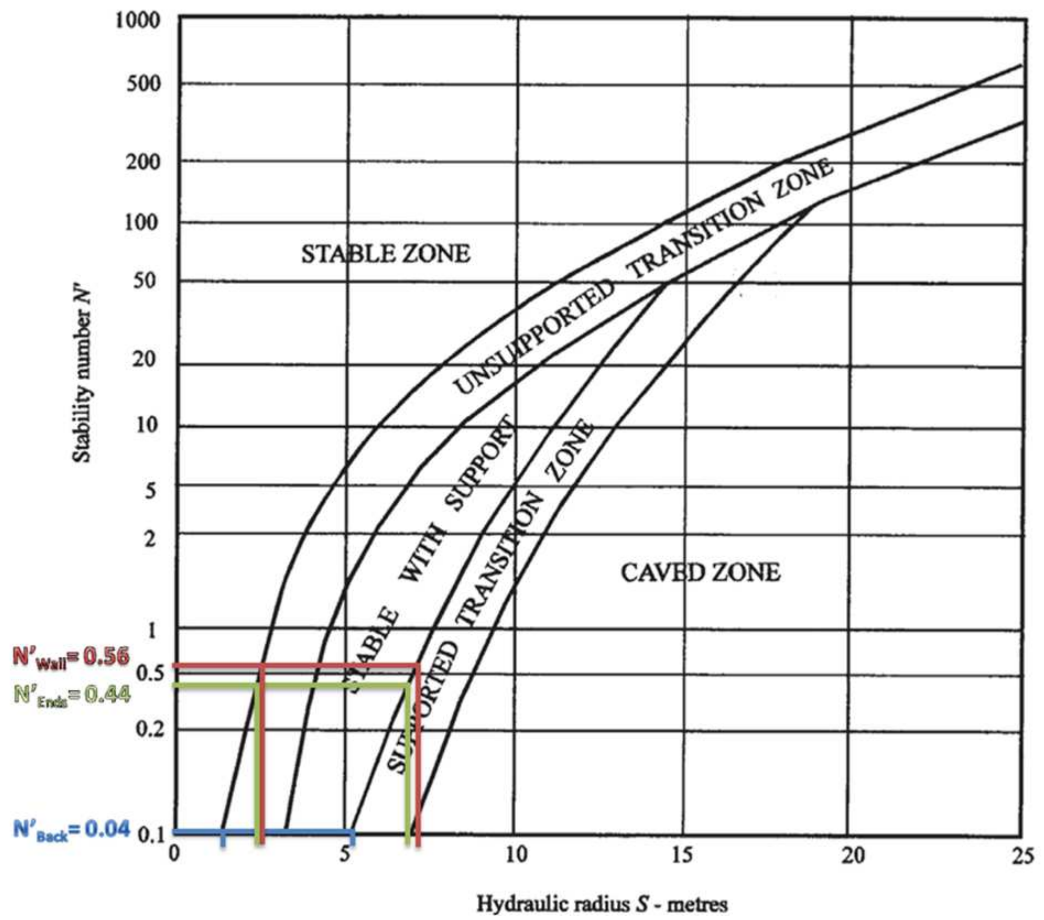
The hydraulic radii of stable spans, with and without support, were calculated using the N' values as shown in Table 18.2.5_4 for different sectors of the stope. These were also tabulated in Table 18.2.5_5.

Table 18.2.5_5
Ollachea Gold Project
Stable Hydraulic Radii for Stope Faces

	Depth (m)	N'	Max Stable HR	Max Stable HR with Cable Bolt
Back	65-380	0.04	1.30	5.20
Hanging wall	65-380	0.56	2.50	7.10
Ends	65-380	0.44	2.40	6.80

Note that the estimates were based on the information summarized in Figure 18.2.5_2 per Potvin and Nickson.

Figure 18.2.5_2
Achievable Hydraulic Radius at Ollachea for Open Stoping



Based on the above hydraulic radii, the stope dimensions for unsupported and supported stopes are given in Table 18.2.5_6 and Table 18.2.5_7 respectively.

Table 18.2.5_6
Ollachea Gold Project
Unsupported Stope Dimensions for Ollachea Deposit

Stope Face	HR	Stope Width (m)		
		5	10	15
Back	1.3	5.4 (L)	3.5 (L)	3.1 (L)
Hanging wall	2.5	5.4 x 67.5 (L x H)	3.5 x ∝ (L x H)	3.1 x ∝ (L x H)
Ends	2.4	120.0 (H)	9.2 (H)	7.0 (H)

Table 18.2.5_7
Ollachea Gold Project
Supported Stope Dimensions for Ollachea Deposit

Stope Face	**HR**	**Stope Width (m)**		
		5	**10**	**15**
Back	5.2	∝ (L)	∝ (L)	33.0 (L)
Hanging wall	7.1	30x26 (L x H)		
Ends	6.8	∝ (H)	∝ (H)	145.0 (H)

18.2.6 Access Drive Conditions and Support

The Ollachea access drive's ground conditions and support requirements have been analysed using worldwide accepted empirical guidelines for support selection.

The two most widely used rock mass classifications are Bieniawski's RMR (1976, 1989) and Barton et al's Q approach (1974).

<u>Q Rating</u>

The Q values for 25 percentile and 50 percentile RQD values have been estimate to be 0.22 and 0.78, respectively.

<u>RMR</u>

The RMR values for 25 percentile and 50 percentile RQD values are calculated to be 30 and 42 respectively using the following relationship:

RMR= 9lnQ + 44

<u>Support Requirements based on the Q-system</u>

Based on Q-system the excavation will be developed in a very poor rock mass. The following reinforcement categories will apply:

- At a Q value of 0.22, for a 5.0m by 5.0m tunnel, reinforcement category of 5 (fibre reinforced shotcrete and bolting, 5 – 9cm) will be required.
- At a Q value of 0.78, for a 5.0m by 5.0m tunnel, reinforcement category of 4 (systematic bolting and unreinforced shotcrete and bolting, 4 – 10cm) will be required.

<u>Support Requirements based on RMR</u>

Based on the calculated RMR values of 30 and 42, the tunnel will be developed in a poor to fair rock mass conditions.

<u>Access Drive Support Recommendations</u>

Using the guidelines for both the Q and RMR methods the recommended support for the tunnel is presented in Table 18.2.6_1

Table 18.2.6_1
Ollachea Gold Project
Support Recommendations for the Ollachea Access Tunnel

RQD %	Bieniawski's RMR			Q - System			Recommended Support for Ollachea Access Tunnel
	RMR	Rock Mass Class	Support Based on Bieniawski's Guideline	Q	Reinforcement Category	Support Based on Q Chart	
10	30	Class: IV Poor Rock RMR: 21-40	Systematic bolts 4-5m long, spaced 1-1.5m in crown and walls with wire mesh. Shotcrete 100-150mm in crown and 100mm in sides. Light to medium ribs spaced 1.5m where required.	0.22	5	Systematic bolting and fibre reinforced shotcrete 50mm to 90mm thick	Systematic bolts 3m long spaced 1m in crown and walls. Steel Fibre Reinforced Shotcrete 75mm in crown and in sides.
35	42	Class: III Fair Rock RMR: 41-60	Systematic bolts 4m long, spaced 1.5-2m in crown and walls with wire mesh in crown. Shotcrete 50-100mm in crown and 30mm in sides.	0.78	4	Systematic bolting and unreinforced shotcrete 40mm to 100mm thick.	Systematic bolts 3m long spaced 1.5m in crown and walls. Where required Steel Fibre Reinforced Shotcrete 50mm in crown and in sides.

The current recommendations for the underground access arrangements were based on limited data and Coffey Mining makes the following observations:

- It should be noted that the rock mass ratings used in the support recommendations were based on the information obtained from the mineralized zone. The rock mass conditions along the tunnel may vary.
- Additional information of the targeted location; the access point; the presence of any major structures; and of the access's intersection with the mineralized zone; should be gained in stages through the PFS and Feasibility studies.
- The access arrives on the hanging wall side, as indicated earlier. More will be required as to its interaction with mining production in the later stages of any study.
- In addition, it is strongly recommended that ongoing rock mass rating should be carried out during the development of the tunnel, and relative adjustments to the support requirements should be made based on the principles set out in this report and agreed with the contractor.

18.3 Mining

18.3.1 Mining Method Selection

<u>Introduction</u>

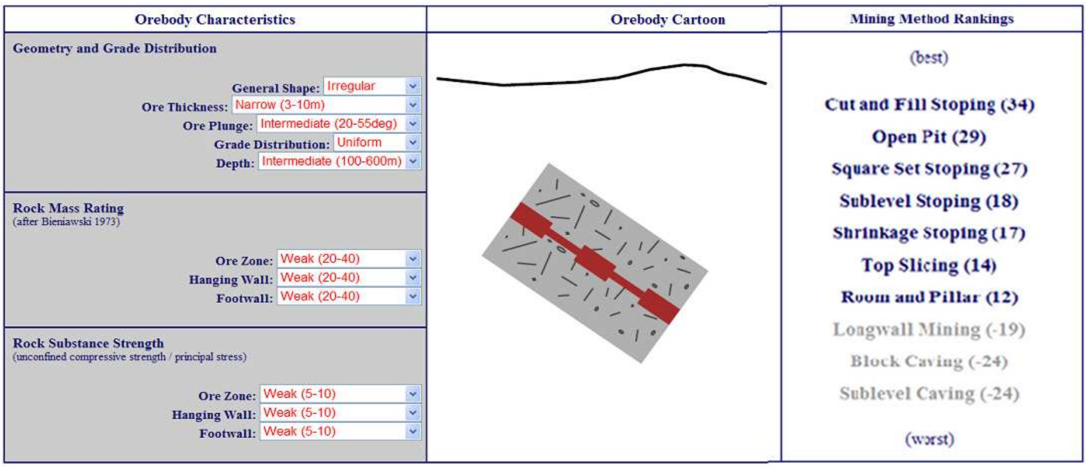
The lenses dip at an average of 50° to 55° to the n orth. The thickness is irregular and varies from 2m to more than 25m in some areas. The average thickness is estimated to be 7m. The Resource extends in the eastwest direction about 800m and is still open along strike. It is about 530m vertically, with over 90% of the tonnes in the upper 325m section. In the north-south direction, the deposit covers about 350m.

<u>Mining Method Selection</u>

To help with the mining method selection, an assessment was made using the Nicholas method. The Nicholas approach to method selection is semi-quantitative and works by applying a ranking to all of the physical characteristics of the deposit. The method was modified by Miller-Tait et Al. (1995) to include relative depth and rock mass rating rather than fracture spacing and fracture strength along with the initial parameters set by Nicholas (1992). The summary of the known Project characteristics and method rankings are as shown in Figure 18.3.1_1.

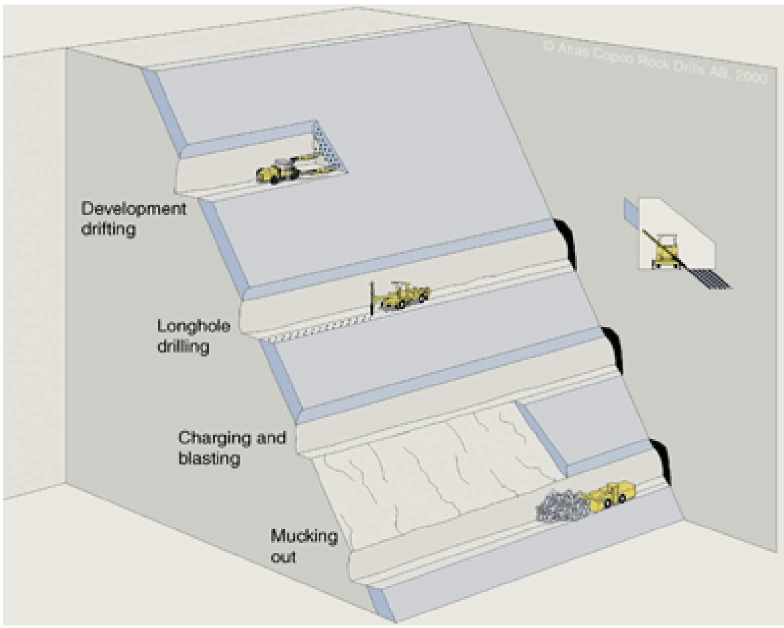
The analysis identifies cut and fill stoping as the best suited mining method for the deposit, as the ground conditions are weak and the plunge is classified as intermediate. The method rated second best is Open Pit, as the mineralization is relatively shallow. This possibility has been evaluated and has not been found economical for this project, some of the major constraints being the limited waste disposal capacity available nearby and the diversion of the stream crossing the deposit. The Sublevel Stoping method comes after Square Set Stoping because of the weak rock mass rating. Square Set Stoping was not selected because of its low production rates.

Figure 18.3.1_1
Nicholas Modified Method Summary



The method selected for the current study is sublevel stoping in a narrow vein setting as presented in Figure 18.3.1_2. The weak ground conditions will be offset by reducing the height of the stopes and the use of tailing based backfill. Some shallow dipping areas may have to be mined using the cut-and-fill method. The stopes are designed to be mined with longitudinal accesses and do not extend high vertically, with sublevels kept at only 15m distance from floor to floor in the vertical axis. Stopes are 30m long in the horizontal axis. The geotechnical aspect is the current limiting factor for stope size.

Figure 18.3.1_2
Typical View of the Selected Mining Method (Sourced from Atlas Copco)

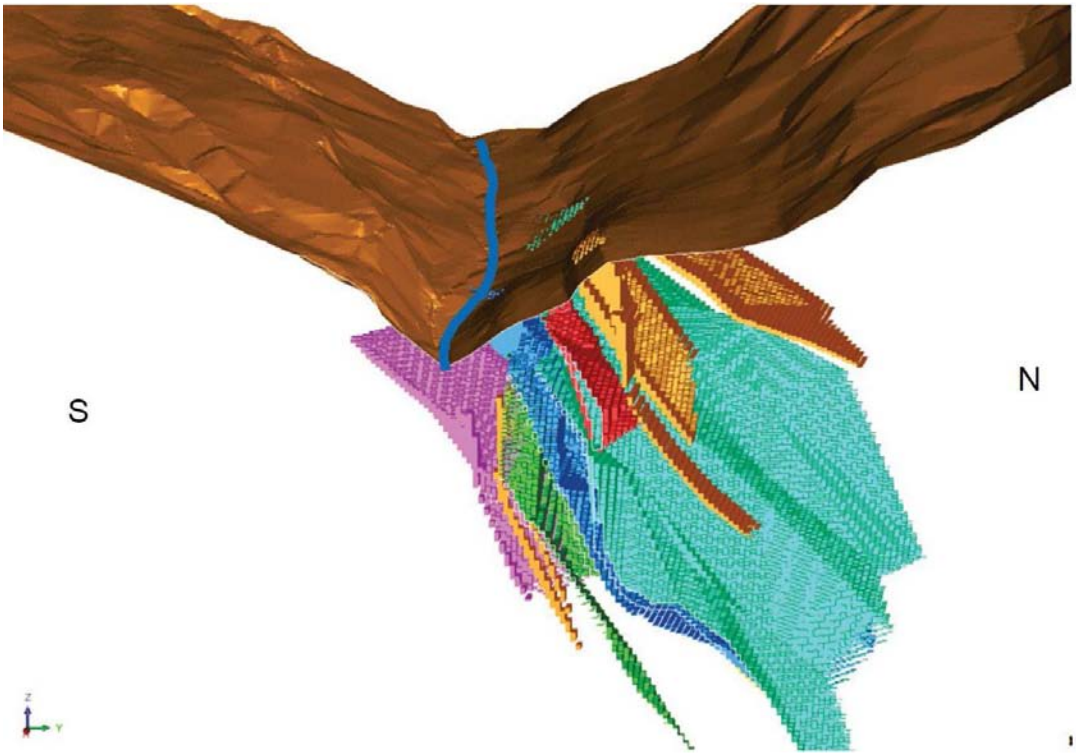


Alternate Mining Methods

Selection Constraints

The highly mountainous area of the Project brings many constraints to the mining method selection. The disposal of waste rock and tailings, for instance, is a critical aspect in the mining method selection. The deposit sits near the bottom of a valley between two mountains with a stream flowing above it as shown in Figure 18.3.1_3.

Figure 18.3.1_3
Deposit vs Stream's Position



18.3.2 Mine Design

Introduction

The mountainous area of Ollachea is very limiting with regards to any major mining infrastructure. The actual road access to the mineralized zone and locating suitable waste rock and tailings disposal areas will be challenging. However, it does provide the opportunity to access the mine by means of a relatively short, near-horizontal drive that would provide access through the mountain to the plant site located towards the northeast. Consequently, for the Study, it has been assumed that no shaft infrastructure would be required, with only an access drive, about 1.3km long, would need to be excavated from the potential plant site. This access drive will be developed during the exploration period to serve as an exploration drive, which will allow drilling of deep down-plunge extensions of the mineralized bodies that are currently not easily accessible from the mountain side. The drive will then be converted to a tramming drive for ore production and transportation of personnel and materials.

Design Constraint

The quickest access to the mineralized zone would normally be from directly above. However, the terrain at the Project, comprising steep mountain walls and narrow plateaus is not favourable to the installation of major infrastructure.

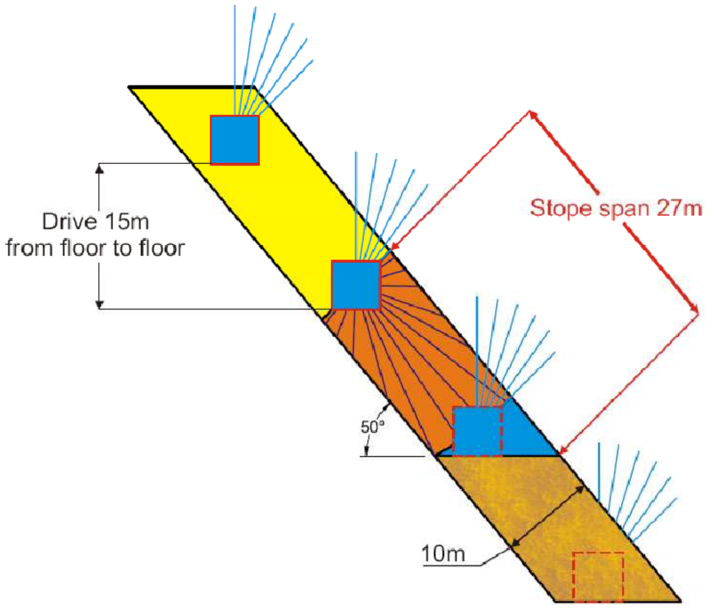
The intermediate plunge of the mineralized zone is on average at 50° to 55°, however, when the dip of the lenses is below 45° it may limit the free flow of rock in the stopes.

The geotechnical analysis gives a generally poor rock mass rating. This factor influences negatively the mining costs and productivity rates.

Level Interval

The interval between levels has been set at 15m for geotechnical reasons. Figure 18.3.2_1 shows a typical section of the mineralized zone plunging at 50°. With a lens thickness of 7m, when using a 5.0m by 5.0m drive, a 15m vertical distance from floor to floor gives a stope span of 26m and for a 10m lens thickness with the same parameter the span becomes 27m.

Figure 18.3.2_1
Typical Sections of the Mineralized Zone at 50°



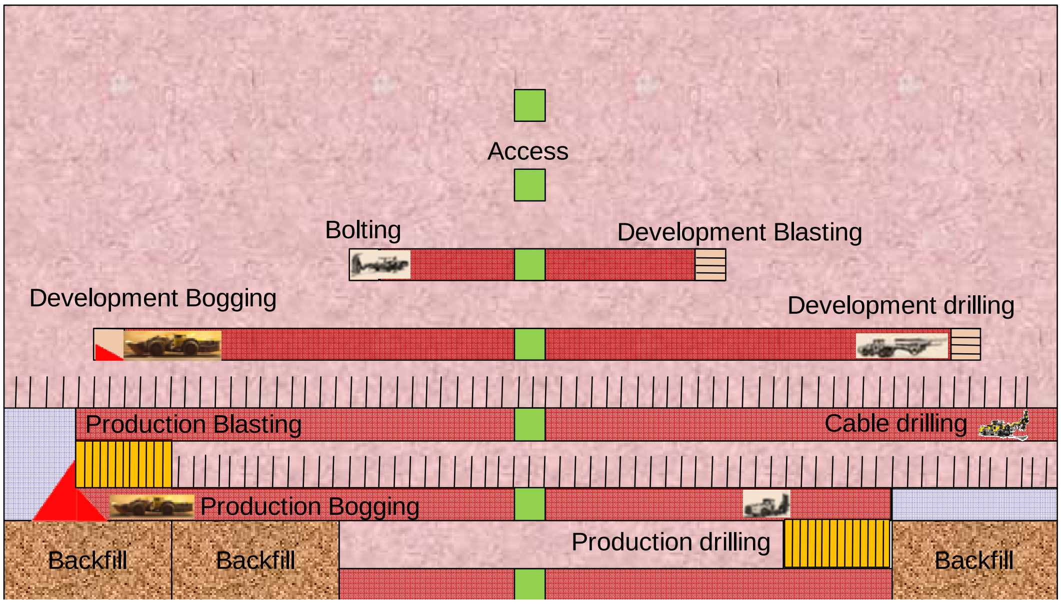
The geotechnical estimation of the maximum stable stope span is currently of 26m. This estimate is based on empirical data: it is therefore acceptable to use the 27m span described by the geometry of the stope configuration of 10m width or more which is less than 4% higher than the estimated maximum span.

It is technically possible to efficiently drill and blast a 76mm diameter hole with a good precision up to 20m; longer holes are usually drilled with a larger diameter to increase precision. The proposed level interval will maintain a maximum hole length of about 20m, allowing for good performance and precision.

Stope Design

The stopes have been design to be 30m in length by the width of the lens and 15m vertically as previously stated, for geotechnical reasons. Figure 18.3.2_2 shows a typical longitudinal view of the mining method.

Figure 18.3.2_2
Typical Longitudinal View of the Mining Method



Development Layout

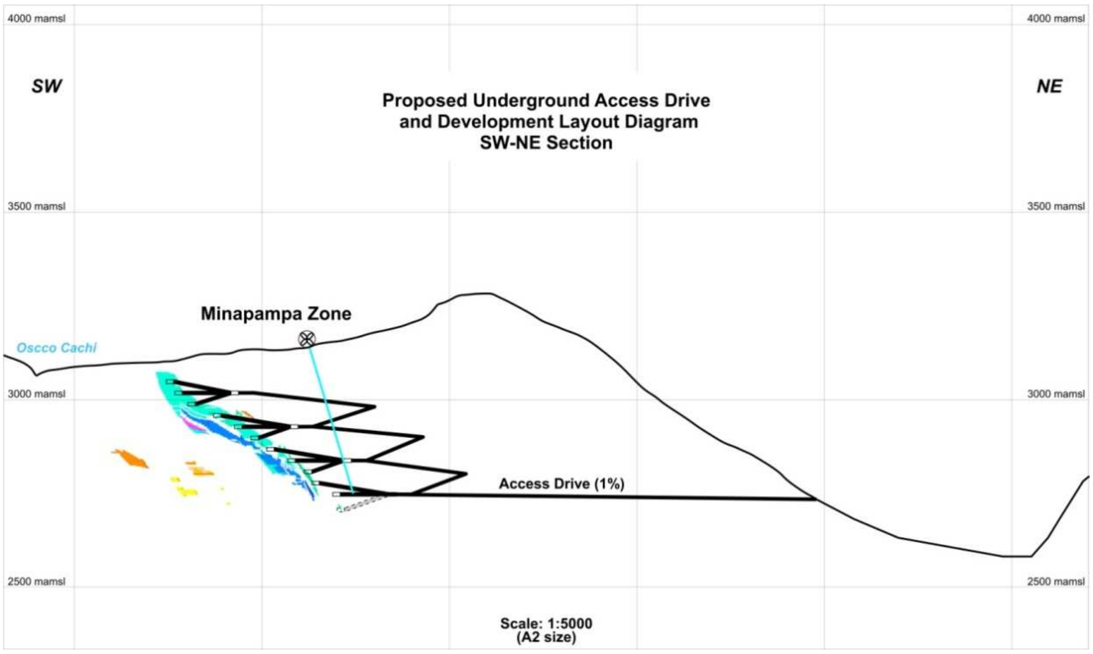
Portal and Access Drive

The current plan is to access the mineralized zone through an adit from the northeast side of the Project. This access would permit run of mine (ROM) transport directly to the proposed plant site. Figure 18.3.2_3 presents a sketch of the access drive. For adequate transport and ventilation, this drive is designed at a 5m by 5m section. This size will easily allow for trucks of 30t to 50t to be used depending on the final equipment requirements. For the purpose of the current study, the truck size chosen is 45t. This access is estimated to be about 1,300m at approximately 1% gradient. No passing bays have currently been included in the design as every 100m a 10m cross-cut can be used for vehicle parking while traffic is passing through. For costing purposes, the drive has been assumed to require heavy support including shotcrete. As the drive will be part of the exploration budget, it has not been included in the scoping study financial model.

Decline

The decline is designed to be roughly 120m from the mineralise zone with level access extending from it every 45 vertical metres. All the development is positioned on the hanging wall to get to the main lens (Lens Number 2) more efficiently as well as keeping away from the stream bed when in the upper part of the mine. The cross-section of the decline is 4.5m by 4.5m as for most waste development drives.

Figure 18.3.2_3
Access Drive



Waste Development

The cross-section of the waste development is generally 4.5m by 4.5m with the exception of loading point in the ore access drives where the backs (roofs) will need to be about a metre higher to assist with loading. For scheduling and costing purposes, all the waste development, apart from the access drive, has been summarized to their equivalent metres of 4.5m by 4.5m section.

Ore Development

For longitudinal stoping, the long-hole mining method requires the development of the full length of the mineralized zone on at least the bottom level. The upper level access mainly serves as a drill and blast level for the mining of the bottom lift and becomes the loading level as the mining progresses up. It is also possible to drill and blast up from the bottom loading level when no further mining upward is required.

Development Summary

A cross-sectional area of 4.5m by 4.5m formed the basis for scheduling and costing purposes, and all horizontal waste development with a different cross-sectional area was back calculated to the 4.5m by 4.5m area equivalent on a cubic metre ratio basis. Table 18.3.2_1 summarizes the development physicals for waste and ore as well as horizontally and vertically.

Table 18.3.2_1
Ollachea Gold Project
Development Physical

Items	Horizontal Waste Development	Dimensions	Length (m)	Equivalent Dimensions (4.5m x 4.5m) (m)
1	Access drive and Drill cut-out	5m x 5m x 1500m	1,500	1,650
5	Longitudinal Main upper level drive	4.5m x 4.5m x 400m	2,000	2,000
1	Decline 2550 to surface	4.5m x 4.5m x 3,300	3,300	3,300
5	Longitudinal Main lower level drive	4.5m x 4.5m x 100m	500	500
40	Stope Access Above 2700mRL	4.5m x 4.5m x 100m	4,000	4,000
16	Stope Access Below 2700mRL	4.5m x 4.5m x 100m	1,600	1,600
50	Lenses crossing access	4.5m x 4.5m x 25m	1,250	1,250
10	Vent Raise Access	4.5m x 4.5m x 150m	1,500	1,500
5	Crib Rooms & Refuge Chamber	4m x 4m x 8m	40	32
10	Sumps	4.5m x 4.5m x 6m	60	60
2	Pump station	5m x 5m x 15m	30	37
4	Underground storage area	4.5m x 4.5m x 6m	24	24
1	Others	4.5m x 4.5m x 500m	500	500
Total Horizontal Waste Development			**16,354**	**16,453**
Items	**Horizontal Waste Development**	**Dimensions**	**Length (m)**	
2	Main Vent Raise	4mϕ x 300m	600	
1	Main Vent Raise	4mϕ x 300m	200	
Total Vertical Waste Development			**800**	
Items	**Ore Development**	**Dimensions**	**Length (m)**	
30	Lens 2 (19*400m+11*150m)	5m x 5m x 310m	9,300m	
29	Lens 3	5m x 5m x 150m	4,350m	
12	Lens 4	5m x 5m x 75m	900m	
21	Lens 5	5m x 5m x 50m	1,050m	
9	Lens 6	5m x 5m x 25m	225m	
12	Lens 7	5m x 5m x 75m	900m	
14	Lens 8	5m x 5m x 150m	2,100m	
1	Others	5m x 5m	500m	
Total Ore Development			**19,325m**	

18.3.3 Development and Mining Schedule

Development Schedule

Table 18.3.3_1 presents the schedule for both capital and operational development. The schedule has been developed for a rate of 120m per month per jumbo, using three jumbos. Bolting is done with two mechanised bolters and two scissor lift teams with air legs and stopers as backup. The equipment used for loading and transport will be the same as for production. It has been assumed for costing, that about 25% of the development will be shotcreted, both in ore and waste.

Mining Schedule

Table 18.3.3_2 presents the schedule for the mining and processing ore feed. The mining production will be achieved with three loaders with a rated payload of 17.2t at 80% fill factor and five trucks with a rated payload at 45t. The number of units required takes into account the requirement for development loading and trucking also.

Table 18.3.3_1
Ollachea Gold Project
Development Schedule

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Horizontal Development											
Longitudinal Main upper level drive	500m	500m	300m	300m	300m	100m					2,000m
Decline 2550 to surface	950m	700m	600m	600m	450m						3,300m
Longitudinal Main lower level drive				100m	100m	100m	100m	100m			500m
Stope Access Above 2700mRL	800m	800m	600m	600m	600m	400m	200m				4,000m
Stope Access Below 2700mRL				300m	600m	300m	300m	100m			1,600m
Lenses crossing access		200m	200m	150m	150m	150m	150m	150m	150m		1,300m
Vent Raise Access	325m	400m	175m	150m	150m	150m	150m	0m	0m		1,500m
Crib Rooms & Refuge Chamber	6m	13m	6m	6m							32m
Sumps	6m	12m	12m	12m	12m	6m					60m
Pump station	19m			19m							37m
Underground storage area		6m	6m		6m	6m					24m
Others		100m	100m	100m	50m	50m	50m	50m			500m
Total Horizontal Waste Development	**2,606m**	**2,731m**	**1,999m**	**2,337m**	**2,418m**	**1,262m**	**950m**	**400m**	**150m**		**16,503m**
Vertical Development											
Vent Raise (2 x 300m+1 x 200m)	600m				200m						800m
Total Vertical Waste Development	**600m**				**200m**						**800m**
Ore Development											
Lens 2 (19*400m+11*150m)	440m	800m	1,240m	1,240m	1,240m	1,240m	1,240m	930m	930m	620m	9,920m
Lens 3	300m	550m	550m	550m	550m	550m	575m	475m	150m	100m	4,350m
Lens 4			150m	150m	100m	125m	125m	125m	125m		900m
Lens 5		100m	200m	100m	100m	200m	250m	100m			1,050m
Lens 6						75m	75m	75m			225m
Lens 7			150m			250m	300m	200m			900m
Lens 8		150m	150m	125m		500m	650m	425m	100m		2,100m
Others						167m	167m	167m			500m
Total Ore Development	**740m**	**1,600m**	**2,440m**	**2,165m**	**1,990m**	**3,107m**	**3,382m**	**2,497m**	**1,305m**	**720m**	**19,945m**

Table 18.3.3_2
Ollachea Gold Project
Mining Schedule

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Mill feed (t)		700,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	1,000,000	497,749	8,197,749
Ore devel (t)		163,800	170,800	151,550	139,300	217,467	236,717	174,767	91,350	50,400	1,254,400
Ore Stope (t)		536,200	829,200	848,450	860,700	782,533	763,283	825,233	908,650	447,349	5,445,600
Waste tonnes (t)	168,863	154,827	113,362	132,499	144,138	71,555	53,865	22,680	8,505		963,848
Total material moved (t)	168,863	854,827	1,113,362	1,132,499	1,144,138	1,071,555	1,053,865	1,022,680	1,008,505	497,749	9,161,598
Stope/year		67 s/a	104 s/a	106 s/a	108 s/a	98 s/a	95 s/a	103 s/a	114 s/a	56 s/a	
tonnes/day		1,918 t	2,740 t	2,740 t	2,740 t	2,740 t	2,740 t	2,740 t	2,740 t	2,740 t	
Avg Stope/month		6 s/mth	9 s/mth	9 s/mth	9 s/mth	8 s/mth	8 s/mth	9 s/mth	9 s/mth	5 s/mth	

The average size of the stopes is 8,000t which make the average number of stopes required per month about eight stopes. The number of stopes required varies with the amount of ore development executed in a month. With such a high number of small stopes, work will have to be carried on from multiple levels and lenses in a sequence that allows a minimum of 14 days of cure for the backfill prior to mining a stope adjacent to a fill face. A good balance between the thicker lenses and the narrow lenses will need to be maintained to ensure a stable production rate.

Equipment

Table 18.3.3_3 presents the list of the mobile equipment required to achieve both the development and mining schedule. The 4.8m³ loader is not part of the production fleet and will be used for services such as barricade construction, it may be used as a backup for production.

Table 18.3.3_3
Ollachea Gold Project
List of Mobile Equipment

Items		**Units**
Drilling Equipment	Jumbo drill	3
	Production drill	2
	Cable support drill	2
	Total Drilling Equipment	**7**
Ground Support & Services	Bolter	2
	Scissor lift	2
	Scissor lift	2
	Total Ground Support & Services	**6**
Load & Haul	7.2m³ LHD	3
	4.8m³ LHD	1
	45t truck	5
	Total Load & Haul	**9**
Ancillary Equipment	Grader	1
	Utility vehicle	14
	Charging gear and vehicle	1
	Pallet handler	1
	Total Ancillary Equipment	**17**
Total Mobile Equipment		**39**

The number of jumbo drills was derived from a development rate of 120m per month per jumbo. This rate should be easy to maintain even in difficult ground conditions with multiple drive faces available as will be the case in this Project.

Workforce

Table 18.3.3_4 presents the list of the total site workforce required to operate every aspect of the operation. Of the 376 employees, a total of 147 people will be working on direct mining activities. Mine maintenance, mine engineering, geology and mine supervision represent 52 staff of the 126 people in G&A.

Table 18.3.3_4
Ollachea Gold Project
Site Workforce

Activity		**Workforce**
Development	Jumbo drilling	18
	Bolting	12
	Scissor Bolting and services	12
	Extra Support, Shotcrete and other material	6
	Total Development	**48**
Mining Production	LHD	16
	Truck rated payload	20
	Production drilling	8
	Blasting	14
	Ground Support (cables in stopes)	14
	Backfill	16
	Services	11
	Total Mining Production	**99**
Maintenance	Mechanic	12
	Electric	8
	Total Maintenance	**20**
Processing Plant and Tailing		**83**
G&A includes Mining eng, geo and U/G maintenance		**126**
Grand Total Site Labour and Staff		**376**

18.3.4 Ventilation

Introduction

Ventilation requirements are driven by the following factors:

- The size of the diesel fleet, usually specified in cubic metres of ventilating airflow per kilowatt of diesel engine power.
- The need to dilute the products of combustion to a safe level and then remove them from the workplace. This is the combination of blasting fumes and engine exhaust.
- The need to remove heat and humidity from the mine environment. This is of particular concern in mines at depth or with a very high geothermal gradient.

In most situations, the most significant determinant of ventilating airflow for mechanised mining is the size of the diesel fleet.

Estimated Requirement

The four methods used to estimate the required airflow indicate airflow between 264m³/s and 313m³/s with an average of 290m³/s. A simple average of the 4 methods will therefore be used for the scoping study.

Minimum Airflow for Each Working Area

As well as providing the required total airflow, the network must provide sufficient airflow for each working area. This has been estimated based on the expected fleet working in each area.

The estimate in Table 18.3.4_1 shows that a minimum of over 50m³/s is required in each of the production areas where a loader and trucks will work together. The mine configuration is such that only the loader will access the stope and the truck will be waiting to be loaded near the decline in the stope access drive. The level access area will require the airflow to be above 50m³/s, whereas the stope and stope access will only require 30m³/s. Having 30m³/s in the stoping area will be sufficient for all the other mining activities in the stope and stope access area as no other equipment or combination of equipment require as much airflow as the loaders. The other working areas should therefore be suited to feed the minimum of 30m³/s per stoping area.

Table 18.3.4_1
Ollachea Gold Project
Estimation of the Airflow Requirements for a Single Work Area

Item		kW Output	Number or Units	Utility	m³/s @ 0.067m³/s per kW
Load & Haul	Truck	330kW	1	100%	22m³/s
	LHD	440kW	1	100%	30m³/s
Load & Haul		Avg 385kW	2	100%	52m³/s

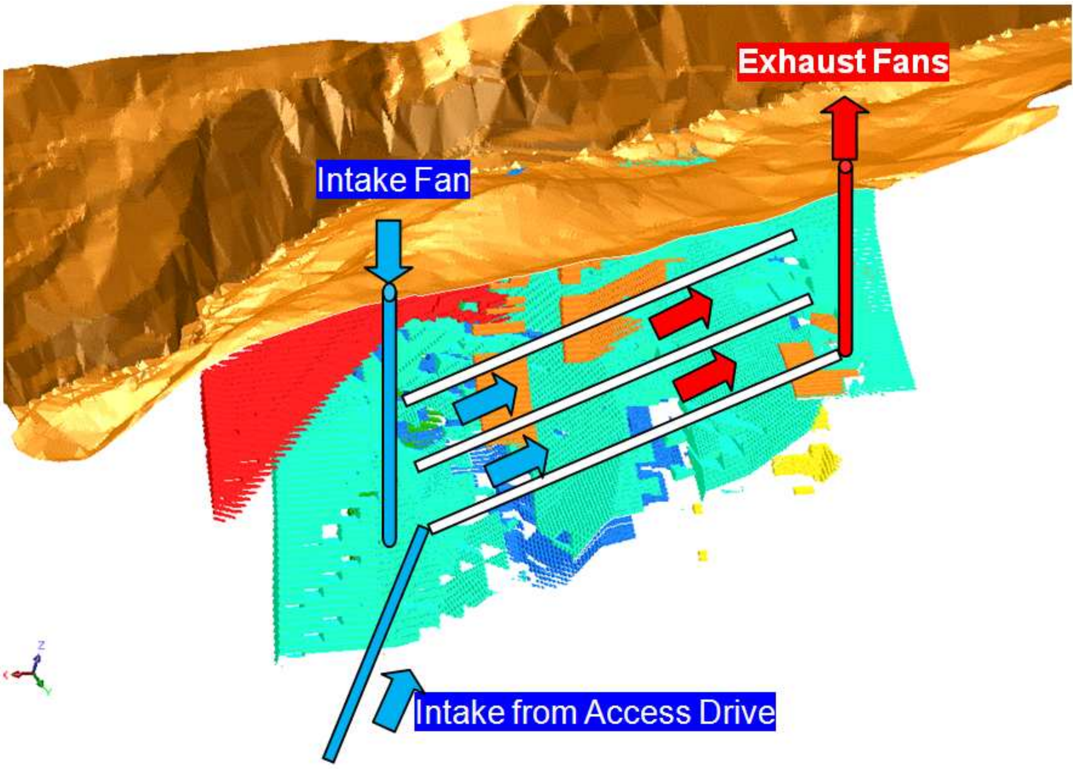
Proposed Ventilation Layout

It is proposed that the main access drive be used as one of the intake air supplies along with one of the two 4m diameter raises to be placed on the east and west extensions of the deposit. The second 4m diameter raise will be used as the exhaust. The system will pull the air from the exhaust raise with two main fans: one fan will be used to push air in the intake raise so that no obstructions are placed in the access drive portal. The weather at Ollachea is mild and should not require either heating or cooling of the air prior to its entering the mine. As mining progresses upward, accesses to the vent raises will be excavated and control walls will be used to adjust the required airflow on each level. Figure 18.3.4_1 shows a sketch of the proposed ventilation circuit air flow.

Proposed Fans

It is assumed that three large axial fans will be needed to provide the required duty in this high altitude environment. A combination of smaller units in series or parallel could also be evaluated to optimise cost or pressure requirements at higher altitude.

Figure 18.3.4_1
Proposed Ventilation Circuit



18.3.5 Pumping and Drainage

Water Inflow

Water inflow at the Project is expected to be high. The inflow to the mine is generally from a combination of rainfall, water used in mining and groundwater. Based on the planned level of activity, an accurate estimate of the expected inflow from mining can be made. Water is used for flushing, hydraulic cooling and dust suppression in the drilling operations and for dust suppression during loading.

The proximity of the mineralized zone to the stream and the fact that potable water for the Ollachea village is being sourced from underground near the mine site indicates high potential for ground water inflow to the mining area. It is also noted that some diamond drillholes encountered artesian water, albeit at low pressure. No measurements of flow were taken from these holes.

For the fleet of equipment proposed, the water consumption listed in Table 18.3.5_1 is expected.

Table 18.3.5_1
Ollachea Gold Project
Equipment Water Consumption

Mining Uses	Number	Utility	Flow (l/s)	Average Flow (l/s)
Jumbo	3	40%	3.0	3.6
Production and Support drilling	4	60%	2.0	4.8
Diamond Drilling	2	70%	1.0	1.4
Dust suppression	3	100%	0.5	1.5
Backfill	1	100%	3	3
Other uses and leaks				0.6
Total				**14.9**

Rainfall will not directly affect the inflow of ground water as the mine does not offer any possible catchment for rain. However, it will be affected indirectly as rain will recharge the aquifers and hence, potentially increase the underground inflow. As no hydro-geological studies have been performed to date, the underground water inflow is assumed to be 15l/s - equal to the equipment consumption. However, detailed hydro-geological studies are required to determine the degree of ground water and the possible flow rates that may be experienced once the mine is fully developed. The presence of large aquifers could increase the drainage system requirements significantly.

Drainage and Pumping

As the access to the mine is not through a shaft but rather by an adit which will lie at about 2,700mRL, most of the water above should be drained by gravity to this elevation rather than being pumped. The mine design includes two pumping stations, one at the bottom of the mineralized zone to lift the water to 2,700mRL from where the second pumping station will pump the water out of the mine through the access drive. This will help to prevent damage to the haulage way by taking as much water as possible away from the road surface. The 2,700mRL pump station will require low-head pressure type pumps and mainly serve to feed the 1.3km dewatering line from the mine to the entrance portal as the gradient in the access drive will be favourable for gravity flow. The 2,550mRL pumping station will be required to pump up approximately 150m head plus the piping friction losses which are about 15m of head in a 120mm inside diameter pipe installed vertically in a borehole. This will be a relatively simple system and should be easy to operate.

18.3.6 Underground Infrastructure

Underground infrastructure will be minimised as ground conditions are not favourable to the excavation of large workshops or other rooms. It is also less cost effective to make underground excavation than it is to construct surface buildings. As the equipment can easily be transported to surface via the access drive, all the required workshops and other facilities should be kept in the vicinity of the plant site. The mine will provide only crib rooms/refuge chambers, pump stations and sumps, and storage areas for ground support supplies and a limited amount of small consumable items. No fuel station will be constructed underground as most vehicles will be serviced at surface at the end of each shift to be serviced. The less mobile equipment, such as drilling equipment, will be supplied via a service truck during the shifts.

18.4 Backfill

18.4.1 Backfill Information

Terrain

The location of the relatively shallow-dipping mineralized zone at the base of a steep sided valley makes the positioning of large infrastructure facilities, such as the processing plant, waste dumps, roads and tailings storage facilities, complex. The conceptual layout of the site assumes that the main plant and backfill facility will be situated on a relatively flat area to the north-east of the mineralized zone over a ridge or spur line. The proposed backfill plant site is about 300m lower than the top of the mineralized zone. The access portal is situated to the north east with the main decline access drive cutting about 1300m through the ridge to intersect the zone.

This configuration means that the backfill will need to be pumped along the access drive to the orebody and then up to the relevant stopes.

Mining Method

The proposed mining method is sublevel mining in a narrow vein setting. The poor ground conditions mean that the individual stope sizes are relatively small, in the order of 26mE x 30mN x 7mRL. The access decline will intersect the mineralization in the bottom third. Over 90% of the mining inventory will be sourced from above this intersection. This means that the need to undercut the backfill is limited to only isolated instances and reduces the backfill strength demand. In addition the relatively shallow orebody angle means that when backfilled stopes are undercut they are only partially undercut reducing the backfill strength requirement. As the mining sequence will start at 2,700mRL some stopes at this elevation will need to be undercut. It may also be chosen during mining to create a new mining front to adjust for operational constraints, in which case, undercutting backfill may be required. An initial assessment of the backfill strength requirements indicates that a minimum unconfined compressive strength of about 1MPa is required to undercut the backfill and an unconfined compressive strength of about 0.35MPa may be needed for vertical stope exposures. Therefore, Coffey Mining estimates that an average of 4.5% w/w cement will be required in the backfill.

Material Properties

The backfill strategy will be largely determined by the materials available at the site, which in this case are the tailings and waste rock produced by the mine itself. Currently, there is only a limited amount of information available on the potential backfill products and therefore the discussion below is in general terms only.

Waste Rock

It is likely that the bulk of the development will be within ore zones and, therefore, waste rock will not significantly contribute to the backfilling strategy. Nevertheless, waste rock can be disposed of in non exposure stopes to limit the amount of waste haulage to a surface waste dump.

Tailings

Design criteria for milling will need to be confirmed as only one particle size distribution (PSD) of the tailings was made available. It can be expected that the final PSD will show some variation to the current data. Table 18.4.1_1 below presents the available PSD data. It is recommended that further testwork is carried out to establish the PSD and variability of the tails product.

Table 18.4.1_1
Ollachea Gold Project
Available Tailings PSD

Particle Size (μm)	Percent Passing
65-113	80
38	63
12	20

There is no specific mineralogy testing currently available on the tailings; however, based on the rock descriptions of the ore zones, it can be expected that the tailings will be relatively inert, with the presence of minor amounts of sulphides.

18.4.2 Backfill Options

The mountainous terrain makes the location of surface tailings disposal facilities challenging and ultimately the tailings storage methodology will dictate the product prepared for the backfill system. The preliminary assessment on tailing disposal presents the most feasible and cost effective surface tailings storage options as being Dry Stacking and indicate that all tailings will be filtered to a solids concentration of about 85%.

The resulting filter cake can be repulped and cement added to produce a suitable backfill using simple and well established technologies. The type of backfill (pastefill or hydraulic fill) will depend largely on the particle size distribution (PSD) of the tailings. If the material has enough fines, generally greater than 15% less than 20μm, then the total tailings can be used to produce pastefill. However, if the PSD is substantially coarser than this, then hydraulic fill will be suitable. In this case, the tailings filter cake will need to be repulped and hydrocycloned, with the overflow returned to the filtration plant and the underflow diverted to the backfill plant. For the purpose of capital and operating cost estimation pastefill has been selected as part of the base case scenario.

If the tailings PSD is suitable, pastefill can provide an appropriate backfill method. The relative advantages and disadvantages of pastefill are presented in Table 18.4.2_1 below:

Table 18.4.2_1
Ollachea Gold Project
Relative Advantages and Disadvantages of Pastefill

Advantages	Disadvantages
Uses the total tailings resulting in a simpler process. Has a higher placement density therefore able to place more tailings underground. Retains the water within the structure and therefore there is no need to wait for the water to drain from the stope during or after placement. This is particularly important because of the rapid cycle time required for the selected mining method.	Reticulation of a paste backfill is the most significant challenge at this operation. Pumping paste uphill is expensive and an alternative arrangement results in double handling.

The most significant challenge to the use of pastefill at this site is reticulation. The process plant is located on the only suitably sized, relatively flat area over the ridge and to the north-east of the valley in which the deposit is situated. The elevation of the process plant and paste plant is similar to the lower levels of the mine. This means that the paste fill will need to be pump approximately 1300m horizontally along the access development and vertically though a height of about 300m. Only appropriately sized positive displacement pumps are suitable for this reticulation. This technology is relatively expensive but proven. Alternatively, it may be possible to truck the filtered tailings to a location situated above the mine and repulp the material to produce paste that can utilise gravity to help reticulate the paste around the mine. A cost benefit analysis should be carried out once it is established where the processing plant will be located and where a suitable tailings repulping facility could be placed above the mine.

18.5 Tailings

18.5.1 Introduction

This section of the report presents the scoping level design for the Tailings Storage Facility (TSF) required as part of the Study. A desk top study was undertaken to assess possible sites for tailings storage. Following selection of a preferred site, design concepts were developed for three possible tailings options.

18.5.2 Document Review

The following information was provided by MKK:

- A general arrangement plan of the Project area number O-04, scale 1:70,000.
- Climatic data in the form of an Excel spreadsheet and PDF documents covering some of the settlements near the Project site.
- Jpeg file of the geology of the exploration areas providing a map of the bedrock geology.
- Topographical information in the form of a DXF file of the Project area near the mine (2D file, contours 50m intervals, levels adjusted to their true elevation in the file by Coffey Mining).
- A map of Peru showing contours of ground acceleration for earthquakes with a 10% exceedance in 50 years.

The results of the document review indicate that the Project area has a climate with similar annual evaporation and precipitation. The annual average precipitation is approximately 950mm and the annual pan evaporation is approximately 1200mm. A TSF in such an environment is likely to have a neutral or slightly positive water gain during operations and at closure. This implies that provision for water diversion and spillways will be required in the design.

The seismic information indicates that the Project area has moderate seismicity, with a ground acceleration of 0.2g for an earthquake of approximately 1:475 year annual exceedance probability (AEP).

Reference to the geological information for the area indicates the geology at a ‘local’ tailings storage site is likely to be complex with sandstone, granodiorite, diorite and syenite at the site. The presence of sandstone within a TSF basin could indicate a potential for ‘high’ seepage from a TSF at such a site depending on the structure of the sandstones.

18.5.3 Siting Study

A desk top study was carried out to identify possible TSF sites. Seven sites were identified. Four of these sites are typified by steep/precipitous terrain and the construction of any confining embankments to form a TSF at these sites would be difficult with high risk. Three other sites were identified between 15km to over 30km from the mine area and can be expected to have lower construction risk; however, the transport corridor between the mine and these remote sites has a high geotechnical risk due to landslides. Based on discussions between Coffey Mining and MKK, it was decided that a conceptual design for a TSF located close to the mine would be presented for the purpose of the Study. The site selected was a site identified by MKK during recent site visits.

18.5.4 TSF Design Concept

<u>Design Criteria</u>

The conceptual design for the TSF is based on the following parameters:

- Total tailings production of 7.5Mt (revised to 8.2Mt*).
- Tailings production rate 1 to 1.5Mtpa (final 1Mtpa*).
- The final split between underground backfill and surface storage was assessed to be 45%:55%.
- Storage facility design capacity nominally 3.75Mt (final 4.5Mt*).

** The construction quantities/cost estimates for the storage of 3.75Mt of tailings were pro-rated to 4.5Mt.*

It should be noted that no physical or geochemical tailings testwork, or geotechnical investigations have been performed as part of the Study. Consequently the conceptual tailings storage design work has been performed based on assumed parameters.

The conceptual design options examined in the Study were as follows:

- Dry Stacking – Base Case:

  Dry Stacking of filtered tailings in an adjacent valley site which was subsequently changed to a site approximately 1.5km north of the plant site.

- HD Thickened TSF:

  Disposal of thickened tailings in a valley storage, involving the construction of a dam embankment(s).

- Valley Storage TSF:

  Storage of unthickened tailings in a valley storage, involving the construction of a dam embankment(s).

Tailings Treatment and Disposal

The Dry Stacking chosen as the base case consists of two 150m² horizontal belt filters complete with all associated vacuum filtration and product handling systems. No rheology, settling or filtration testwork was carried out as part of the completed test program. Assumptions were made as to the amount of filtration based on the type of ore, slurry and rarefied atmosphere under which the plant would need to operate. It was assumed that the belt filter product would be able to be reasonably free of moisture (~15% moisture) and that it would have sufficient strength to be able to form peaks when conveyed rather than flowing. It was also assumed that the filter cake would be sufficiently dry to be able to be easily transported by open top dump trucks from the process facility tails storage pad to either the mine back fill plant or the Dry Stacking TSF.

Following placement, the tailings will be spread using a small dozer and compacted utilising a vibratory roller. It was assumed that the tailings will be a silty sand/sandy silt (50-80% passing 75 micron) in order to allow for ease of compaction.

The perceived benefits of Dry Stacking tailings for the Project can be summarized as follows:

- Recovery of metals from solution through the filtration process prior to stacking.
- Dry Stacks have a low probably of catastrophic failure and can be designed to withstand static and seismic forces.
- The footprint is smaller, when compared to other forms of tailings storage, because of the low moisture content and higher density of the stacked tailings.
- Allows progressive covering and rehabilitation for closure.

A possible site for the Dry Stack is a location identified by MKK approximately 1km to 1.5km north of the proposed plant in an unutilised side valley. Another site was initially considered adjacent to the valley TSF; however, there is no reason why this facility cannot be located closer to the plant. The design concept for the Dry Stack will include an initial starter containment embankment. As the stack is constructed over the life of the mine there will be a requirement for erosion protection of the downstream stack batter and for drainage diversion works to divert runoff upslope of the stack around and downstream of the stack. The landform for the Dry Stack could be potentially terraced to provide useful agricultural land at closure.

In addition the design concept for the tailings stack will include an underdrainage system under the stack in order to collect tailings leachate. The requirement for the artificial lining of the stack area would be based on the geotechnical site conditions and environmental risk. An increased contingency allowance has been made in the cost estimate to allow for liner construction.

The total quantity of mine waste materials forming the starter containment embankment of the Dry Stack was estimated at 65,000m^3. An estimated 120,000m^3 of capping material comprising clayey material to reduce oxygen ingress and erosion protection (non acid mine waste) will be required above the starter embankment level to the final stack height.

The Dry Stack will have an ultimate storage volume of approximately 2.1Mm^3 or a storage capacity of 3.8Mt of tailings assuming a tailings dry density of 1.8t/m^3.

An outline of the capital associated with the Dry Stacking option is shown in Tables 18.6.1_3.

Dry Stacking appears to be the most appropriate route for tailings disposal as the capital cost is the lowest and best deals with the challenging terrain in the area. While this needs to be confirmed in future studies, this option was adopted as the base case for the purposes of the scoping study.

18.6 Costs

All costs as shown in the following sections are denominated in US dollars.

18.6.1 Capital Costs

Mining

The mining capital cost includes the first year of waste development and pre-production ore development. As the access drive will have been developed during the exploration period, it has not been included in the capital costs inputs. Ongoing waste development is included in sustaining capital. A summary of the mining capital costs are presented in Table 18.6.1_1.

Table 18.6.1_1 **Ollachea Gold Project** **Capital Cost Mining**			
Project Capital Cost	**Amount US$M**	**Contingency (20%)**	**Total**
Mining Development (Pre-Prod)	8.0	1.6	9.6
Mining Equipment	41.5	8.3	49.8
Backfill System	5.8	1.2	6.9
Total	**55.3**	**11.1**	**66.4**

Process Plant

A summary of the process plant capital costs is presented in Table 18.6.1_2. These costs have been derived from Coffey Mining's cost database and experience.

Table 18.6.1_2 **Ollachea Gold Project** **Process Plant Capital Cost Estimate**				
Item		**Amount M$US**	**Contingency (20%)**	**Total**
Total Direct Costs		48.4	9.7	58.1
Indirect costs		14.0	2.8	16.8
Total Plant Costs		62.4	12.5	74.9
Owners Costs	Tailings dam	2.0	0.4	2.4
	Communications	0.2	0.0	0.2
	Infrastructure	3.0	0.6	3.6
	Power lines & Ancillary	1.5	0.3	1.8
	Raw water supply	0.3	0.1	0.4
	Spares	2.4	0.5	2.9
	First fill	3.6	0.7	4.3
Total Owners Costs		13.0	13.0	2.6
Overall Costs		**75.4**	**15.1**	**90.5**

Direct Costs

Major equipment items for the plant were sized based on the testwork. These items were costed from Coffey Mining's database for this equipment. Direct Costs include:

- Mechanical equipment and installation.
- Earthworks.
- Concrete supply and installation.
- Structural steel supply and installation.
- Platework supply and install.
- Piping supply and installation.
- Electrical supply and installation.

- Instrumentation supply and installation.
- Buildings supply and installation.
- Painting
- Heavy lift cranes.
- Scaffolding.
- Mobilisation, Demobilisation & Miscellaneous.
- Freight allowance.

These costs were factored from the major equipment supply cost. The factors applied for the estimate are typical for this style of plant; however, extra allowances were made for the remote location and mountainous terrain in the area.

Indirect Costs

Indirect costs (EPCM) such as design, project and construction management, commissioning, Project and construction expenses been factored from the direct costs. Typical factors have been applied.

Owners Costs

Allowances have been made for the following:

- Tailings dam.
- Communications.
- Infrastructure.
- Raw water supply.
- Power supply.
- Spares (5% of direct costs).
- First fill (7.5% of direct costs).

<u>TSF Cost Estimate</u>

The TSF cost estimate was compiled utilising client supplied earthworks and other rates obtained from recent projects in Peru. Construction quantities have been estimated using Surpac / Autocad packages.

Table 18.6.1_3 presents the capital cost for the Dry Stacking TSF.

Table 18.6.1_3
Dry Stacking TSF Capital Cost

Stage	**Dry Stacking**	**Contingency (20%)**	**Total**
Total	$2.0M	$0.4M	$2.4M

The following is excluded from the TSF cost estimates:

- Cost of investigation, design and documentation and design of TSF.
- Cost of government approval documentation.
- Costs associated with management of the approvals process.
- Cost of slurry and return water pipework and associated pumps (costs allowed elsewhere in the study report).
- Cost of mechanical and electrical (including thickeners, filters etc) associated with tailings disposal (costs allowed elsewhere in the study report).
- Operational and closure costs.

18.6.2 Operating Costs

<u>Mining</u>

Development Costs

Table 18.6.2_1 presents the development cost and quantities used in this study. The ore development has been increased by 5% to account for size variation and varying lens directions.

Table 18.6.2_1
Ollachea Gold Project
Development Cost

Items	Horizontal Waste Development	Unit Cost (US$/m)	Quantities (m)	Cost (US$)
5	Longitudinal Main upper level drive	1,475	2,000	2,941
1	Decline 2550 to surface	1,475	3,300	4,853
5	Longitudinal Main lower level drive	1,475	500	735
40	Stope Access Above 2700mRL	1,475	4,000	5,882
16	Stope Access Below 2700mRL	1,475	1,600	2,353
50	Lenses crossing access	1,475	1,300	1,912
10	Vent Raise Access	1,475	1,500	2,206
5	Crib Rooms & Refuge Chamber	1,475	32	46
10	Sumps	1,475	60	88
2	Pump station	1,475	37	54
4	Underground storage area	1,475	24	35
1	Others	1,475	500	735,260
Total			**14,853**	**21,871**
Items	**Horizontal Waste Development**			
2	Main Vent Raise	3,500	600	2,100
1	Main Vent Raise	3,500	200	700
Total Vertical Waste Development			**800**	**2,800**
Items	**Ore Development**			
30	Lens 2 (19*400m+11*150m)	1,550	9,300	15,345
29	Lens 3	1,550	4,350	6,729
12	Lens 4	1,550	900	1,392
21	Lens 5	1,550	1,050	1,624
9	Lens 6	1,550	225	348
12	Lens 7	1,550	900	1,392
14	Lens 8	1,550	2,100	3,249
1	Others	1,550	500	773
Total Ore Development			**19,325**	**30,853**

The development costs are split between capital and operating cost depending on the purpose of the drives or excavation. All ore development is normally put in as an operational cost except during the pre-production period. Conversely, waste development, is generally capitalised unless it is used for direct access to the stopes.

Mining Operation Costs

Table 18.6.2_2 present a summary of the mining unit operating cost.

Table 18.6.2_2
Ollachea Gold Project
Mining Unit Operating Cost Summary

Items		**Unit Cost**
Development Operational Cost	Operation cost Development Waste	1.09 $/t
	Operation cost Development ORE	3.62 $/t
	Total Operation Cost Development	4.71 $/t
Mining	Loading	1.75 $/t
	Trucking	3.95 $/t
	Production drilling	1.68 $/t
	Blasting	1.16 $/t
	Ground Support (cables in stopes)	2.68 $/t
	Backfill	4.90 $/t
	Services	1.25 $/t
	Total Mining Cost	17.37 $/t
	Total Mining and Development	22.08 $/t

For the financial analysis the cost are separated in fixed and variable cost.

Process Plant

Table 18.6.2_3 shows a summary of the plant operating costs derived for the base case. These costs have been based on the testwork and information from Coffey Mining's database for plants with similar operations.

Table 18.6.2_3
Ollachea Gold Project
Plant Operating Cost Estimate

Item	**M$US/a**	**$US/t milled**
Labour	1.0	1.0
Power	2.4	2.4
Reagents	9.3	9.3
Consumables	2.6	2.6
Maintenance Materials	1.7	1.7
Tailings	1.6	1.6
Miscellaneous	0.9	0.9
Total Cost	**19.5**	**19.5**

Labour

The number and type of personnel allocated to run the process plant is based on experience in Minera's Peruvian operation at Corihuarmi.

Power

Based on the testwork, major equipment items were sized for the study. The power draw for the major equipment was used to calculate the overall power requirements. Ancillary equipment power draws were derived from other studies with similar equipment. The processing operation is expected to consume 37kWh per tonne of ore processed.

It is assumed the grid will supply power to the mine at a unit power cost of 0.06US$/kWh.

Reagents and Consumables

Reagent consumptions are based on the KCA testwork. Typical oxygen addition consumptions for primary ores similar to Ollachea have been used in the absence of testwork. Consumables, such as crusher liners, mill liners and mill balls are based on an assumed medium abrasion index.

Unit costs for reagents and consumables are derived from MKK's database.

A cyanide consumption rate of 1.5kg/t of whole ore was used. This accounts for 46% of the total reagent costs and 22% of the total processing operating costs.

Maintenance Materials

Maintenance materials are factored from the direct capital cost estimate of the plant.

Miscellaneous

Allowances have been made for the following:

- ROM pad maintenance and crusher feed.
- Laboratory costs.
- Site vehicle fuel and maintenance.
- Liquid oxygen equipment hire.
- Maintenance contractors.
- Equipment hire.
- Environmental and technical consultants.
- Water supply and TSF operation.

18.7 Financial Analysis

18.7.1 Financial Model

The financial model ("Model") was supplied by MKK to facilitate the compatibility of the Study with the financial structure of the company. The model presents the economics of the project on an annual basis and has been constructed on a real 2009-dollar basis. For purpose of the discount cash flow analysis, the cash flows are assumed to be received mid-year, based on the assumptions that the revenues and costs are spread evenly over the year. It is also assumed that the net present values are at the commencement of year -1.

The model was review by Coffey Mining and was found to satisfy the requirements of the Study.

The following preliminary assessment is preliminary in nature, it includes solely Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary assessment as estimated in the Study will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

18.7.2 Financial Model Inputs

<u>Input Description</u>

The inputs to the financial model include dates and periods, metal prices, mining inventory, mining and processing throughputs, realisation costs, operating costs, capital costs, royalties and taxation parameters. The Model inputs are summarized in the following sections.

<u>Physicals</u>

The mining inventory is 8.2Mt at 4g/t Au head grade containing 1.1M ounces of gold. The mining and processing rate has been set to 1.0Mtpa with a ramp-up period of 70% during the first year. The overall processing recovery is estimated at 91.2% for the life of mine. Table 18.7.2_1 summarizes the mining, processing and production schedule.

<u>Metal Prices and Realisation Costs</u>

The metal prices used in the Model are summarized in the Table 18.7.2_2.

The realisation costs used in the Model are summarized in the Table 18.7.2_3.

Table 18.7.2_1
Ollachea Gold Project
Mining, Processing and Production Schedule

	Year 1	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**Year 8**	**Year 9**	**Total / Average**
Ore Mine/Process(t)	0.7	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.5	8.2
Gold grade (g/t)	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0	4.0
Silver grade (g/t)	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2
Contained gold (oz)	89.9	128.5	128.5	128.5	128.5	128.5	128.5	128.5	63.5	1.1
Contained silver (oz)	27,007	38,581	38,581	38,581	38,581	38,581	38,581	38,581	19,204	316,277
Gold recovery (%)	92%	92%	92%	92%	92%	92%	92%	92%	92%	92%
Silver recovery (%)	10%	10%	10%	10%	10%	10%	10%	10%	10%	10%
Gold production (oz)	82,0	117.1	117.1	117.1	117.1	117.1	117.1	117.1	58.3	960.4
Silver production (oz)	2.7	3.9	3.9	3.9	3.9	3.9	3.9	3.9	1.9	31.6

Table 18.7.2_2 Ollachea Gold Project Metal Prices (2009$)	
Metal	**Price**
Gold (US$/oz)	850
Silver (US$/oz)	12.0

Table 18.7.2_3 Ollachea Gold Project Realisation Costs (2009$)		
Metal		**Cost**
Gold	Payable metal (%)	100%
	Refinery charge (US$/oz)	6.0
Silver	Payable metal (%)	100%
	Refinery charge (US$/oz)	0.5

Operational Costs

Table 18.7.2_4 presents a summary of the operating costs incorporated in the Model.

Table 18.7.2_4 Ollachea Gold Project Operating Costs Summary (2009$)				
Site Operating Cost	**Fixed (US$M/a)**	**Variable (US$/t)**	**Total at Steady State (US$/t)**	**LOM Average (US$/t)**
Mining	2.31	19.77	22.08	22.20
Processing	4.87	14.63	19.50	19.75
G&A	3.87	0.0	3.87	4.07
Total	**11.05**	**34.40**	**45.45**	**46.02**

The life of Project unit production cost per ounce is summarized in Table 18.7.2_5.

Table 18.7.2_5 Ollachea Gold Project Unit Cost of Production per Ounce (2009$)	
Parameter	**LOM Average Cost (US$/oz) Au**
Mining	190
Processing	169
G&A	35
Total Site Operating Costs	**393**
Refinery Charge	6
Silver credit	(0.4)
Mine Cash Operating Cost	**399**
Royalties	20
Total Production Costs	**419**

<u>Capital Costs</u>

The capital costs for the Project are summarized in Table 18.7.2_6.

Table 18.7.2_6
Ollachea Gold Project
Capital Cost Summary (2009$)

Project Capital Cost	**Amount US$M**	**Contingency (20%)**	**Total**
Mining	8.0	1.6	9.6
Mining Equipment	41.5	8.3	49.8
Processing Plant	62.4	12.5	74.9
Infrastructure	11.0	2.2	13.2
Tailings	2.0	0.4	2.4
Backfill	5.8	1.2	7.0
Total	**130.7**	**26.1**	**156.8**
Ongoing Capital Cost	**Amount US$M per a**	**Contingency (0%)**	**Total**
Mine Development	1.4		1.4
Mining Equipment	2.6		2.6
Total	**4.0**		**4.0**
Closure Cost	**Amount US$M per a**	**Contingency (0%)**	**Total**
Closure/Rehabilitation Costs	5.0		5.0
Total	**5.0**		**5.0**

<u>Royalties</u>

The following royalties are included in the Model.

a. <u>Peru Government Royalty</u>

The Peru Government Royalty is based on the following:

- Companies with sales of up to the first US$60 million per year – has a royalty of 1% for that portion of sales;
- With the portion above US$60 million of sales from US$60 million to US$120 million per year – the royalty increases to 2% for that portion of sales; and
- Any sales over US$120 million per year – has a royalty of 3% for that portion of sales.

b. <u>Vendor Royalty</u>

A vendor royalty of 1% net smelter revenue (NSR) is included in the Model.

Peru Taxes and Employee Profit Share

The tax calculation and workers' profit have been supplied by MKK as part of the financial model.

a. Income tax

Peruvian corporate income tax is levied at a rate of 30%.

The method of tax depreciating fixed assets usually allowable is the straight-line method. An average tax depreciation rate of 20% straight-line (useful life 5 years) has been included in the Model.

Tax losses can be carried forward for a period of up to four years. It is estimated the MKK will gross tax losses of US$15 million from previous activities in Peru that can be used to offset income tax payable.

b. Workers' Profit Participation

The workers share in the company's profits through the company's distribution of a percentage of the annual taxable income before Income Tax. Such share percentage is determined according to the activity of the company. Mining companies are obliged to distribute 8%.

Profit sharing only applies to companies having more than 20 workers. In order to determine the annual income to which this benefit refers, the carry forward of tax losses generated in prior years is allowed. The Model has included a distribution rate of 8% after offsetting profits by carried forward gross losses, from previous activities in Peru, of US$15 million. Workers' Profit Participation is deductible for Income tax.

c. General Sales Tax - IGV

The Peruvian Taxation System incorporates a general sales tax called IGV (a value added tax). Because the activity of the Project is the export of goods, the IGV is able to be recovered. The capital and operating costs exclude IGV as it is assumed that the IGV is recoverable immediately.

18.7.3 Financial Model Results

The cash flow model, after tax, shows a US$157M capital investment with an internal rate of return of 17.4% pa real and a net present value (NPV) US$59M when discounted at 8% pa real.

Table 18.7.3_1 and Figure 18.7.3_1 presents the annual cash flows from the Model.

The pre-tax (including pre Workers' Profit Participation) and post-tax Internal Return of Return (IRR); Net Present Value (NPV) at a discount rate of 7% pa real and 8% pa real and payback period are summarized in Table 18.7.3_2. The financial analysis shows promising returns for the Project.

Table 18.7.3_1
Ollachea Gold Project
Cash Flows (2009$)

Parameter	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total / Average
Revenue (US$M)		69.2	98.9	98.9	98.9	98.9	98.9	98.9	98.9	49.2	**811.0**
Operating Costs (US$M)		(35,1)	(45.4)	(45.4)	(45.4)	(45.4)	(45.4)	(45.4)	(45.4)	(24.0)	**(377.3)**
Royalties (US$M)		(1.5)	(2.4)	(2.4)	(2.4)	(2.4)	(2.4)	(2.4)	(2.4)	(2.4)	**(19.0)**
Capital Costs (US$M)	(156.8)	(4.0)	(4.0)	(4.0)	(4.0)	(4.0)	(4.0)	(4.0)	(4.0)	(5.0)	**(16.5)**
Workers' Profit Share (US$M)			(0.3)	(1.4)	(1.3)	(1.3)	(3.8)	(3.7)	(3.6)	(1.1)	**(193.6)**
Income Tax (US$M)			(1.0)	(4.8)	(4.6)	(4.4)	(13.0)	(12.8)	(12.5)	(3.9)	**(56.9)**
Net Cash flow (US$M)	**(156.8)**	**28.7**	**45.8**	**40.9**	**41.2**	**41.5**	**30.4**	**30.6**	**31.0**	**14.2**	**147.7**

Figure 18.7.3_1
Project Cash Flow After Tax

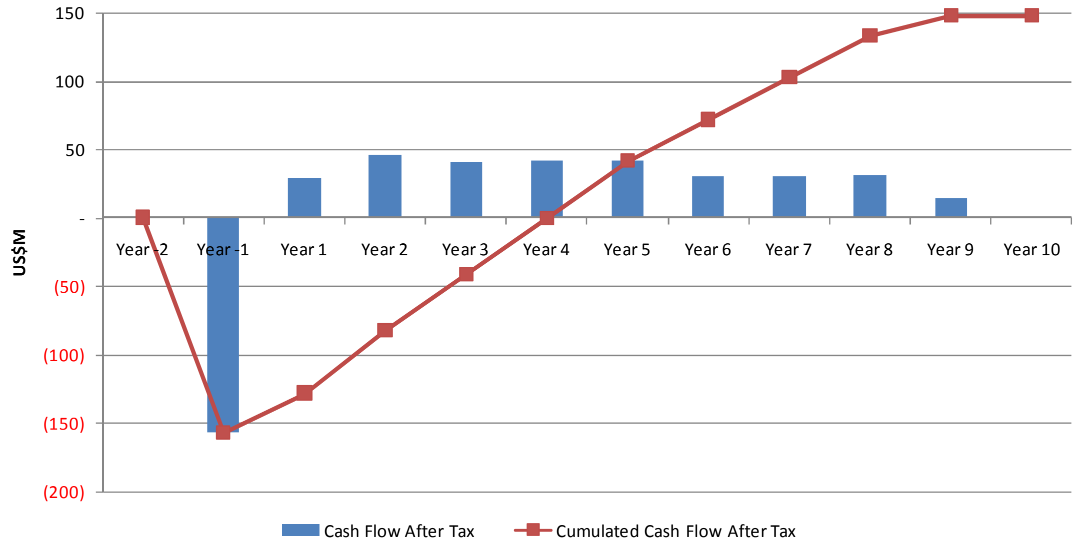


Table 18.7.3_2
Ollachea Gold Project
Project IRR, NPV and Payback

Parameter	Pre-Tax	Post-Tax
Cash flow (real)	US$221.0m	US$147.7m
IRR (real)	22.4%	17.4%
NPV at 7% real	US$113.9m	US$67.3m
NPV at 8% real	US$102.5m	US$58.7m
Payback period from commencement of production	3.7 years	4.0 years

18.7.4 Financial Model Sensitivity

Introduction

The sensitivity analysis was carried out on gold price and gold head grade, operational and capital cost as well as minable tonnes and throughput. The effect of the changes to these parameters was measured against the IRR and the NPV @ 8%. Figure 18.7.4_1 and 18.7.4_2 present the sensitivity of IRR and NPV @ 8% against the variation of the above mentioned variables.

Revenue

Gold price and gold grade have the same effect on the model as they both only affect the revenue in the same manner and do not affect the cost significantly. The sensitivity applied to these items is plus or minus 15% making the price of gold vary between US$722.5/oz and US$977.5/oz and the head grade between 3.4g/t and 4.6g/t. Table 18.7.4_1 presents the sensitivity for these gold price variations.

Figure 18.7.4_1
Sensitivity of the IRR

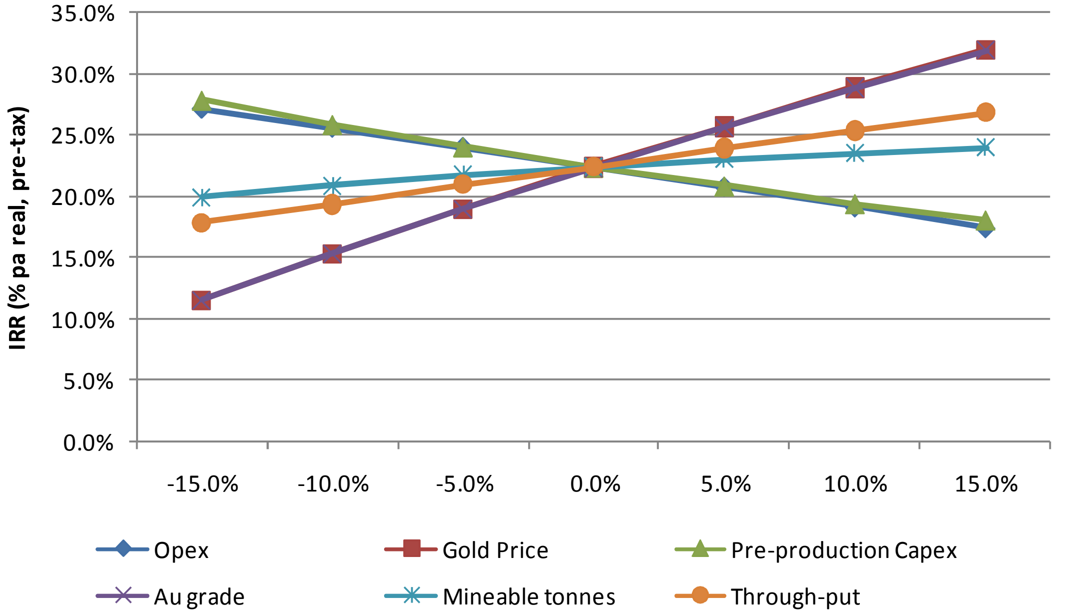


Figure 18.7.4_2
Sensitivity of the NPV @ 8%

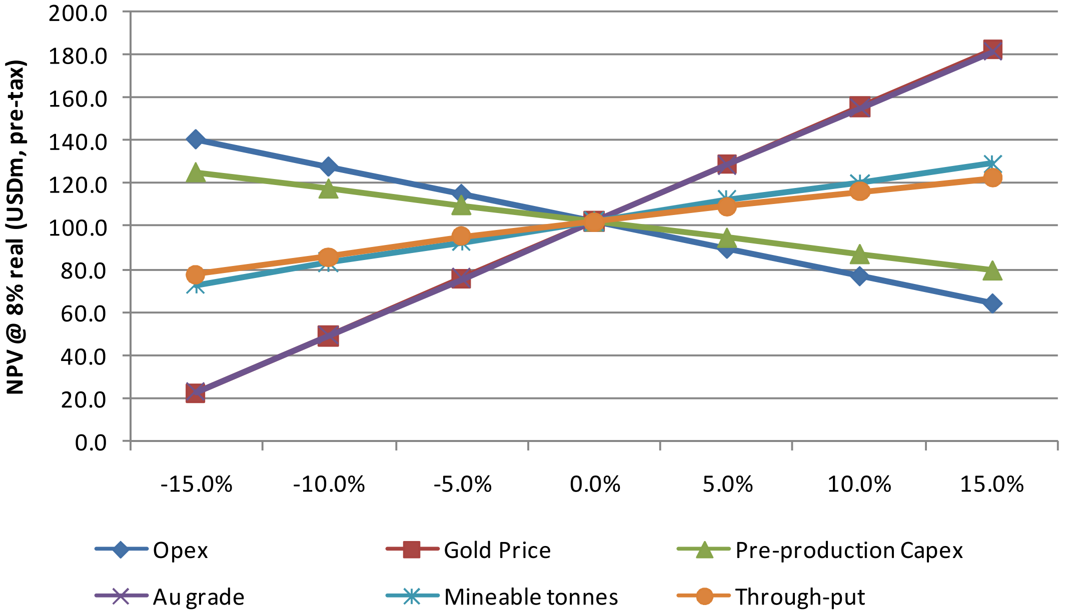


Table 18.7.4_2 Ollachea Gold Project Gold Price Sensitivity						
Gold Price US$/oz	**Pre-Tax**			**Post-Tax**		
	IRR	**NPV @ 8% Real**	**NPV @ 10% Real**	**IRR**	**NPV @ 8% Real**	**NPV @ 10% Real**
700	9.4%	8.7	81.2	7.3%	-4.0	57.6
800	18.3%	71.2	174.4	14.3%	38.3	117.7
850	**22.4%**	**102.5**	**221.0**	**17.4%**	**58.7**	**147.7**
900	26.2%	133.8	267.7	20.3%	78.9	177.7
1000	33.5%	196.3	360.9	25.8%	119.4	237.7
1100	40.4%	258.5	453.5	31.0%	159.5	297.4
1200	46.9%	320.4	545.8	35.8%	199.4	356.8

As with most projects, variables that affect revenue have the largest impact on the project economics. The breakeven point of the gold price for the NPV @ 8% is US$710/oz whereas the IRR reaches zero when the price of gold is US$614/oz. Once in operation, the mine is cash flow positive at gold prices above $400 per ounce.

Operating and Capital Costs

The effect of operating cost on the Project's financial outcomes is the next most important Project driver after gold price and head grade, which is to be expected with a Project where the cutoff grade is close to head grade. Although the capital cost has a significant influence, its impact is less than operating cost.

The breakeven point of the operating cost is US$61.9/t or nearly 35% more than the base case price to bring the NPV @ 8% to zero and capital cost would need to increase to US$238M including contingencies. The Project capital cost would need an increase of over 50% to reach this value.

Some of the major operating cost items such as fuel, power and cyanide consumption were analysed independently to verify their impact on the total operating cost. None of these items showed more than 2% influence on the total operating cost.

Tonnes and Throughput

Minable tonnes and throughput have similar curves but minable tonnes has an effect on total revenue and variable cost whereas throughput only affects the variable cost and extends or shortens the mine life thus contributing more or less a fixed cost to the model.

The effect of either minable inventory or processing throughput is less significant The breakeven point of both minable tonnes and, the throughput is about 45% less of the current tonnes or throughput to make the NPV @ 8% equal to zero.

Current drilling by MKK outside the limits of the Minapampa mineralized zone has indicated the potential for additional tonnes. Table 18.7.4_2 shows the Project returns based on an additional 2.0Mt at a gold grade of 4.0g/t, containing 257,000 ounces.

Table 18.7.4_2 Ollachea Gold Project Project IRR and NPV with additional 2Mt at 4.0g/t Au		
Parameter	**Pre-tax**	**Post-tax**
LOM Cash flow (real)	US$322.4M	US$213.0M
IRR (real)	24.8%	19.7%
NPV at 7% real	US$163.8M	US$99.0M
NPV at 8% real	US$147.8M	US$87.4M
Payback period from commencement of production	3.7 years	4.0 years

It will be important to limit the impact of the more sensitive items whilst the Project is in the early stages of exploration. For instance, expanding the resource east, west and at depth would reduce the impact of an increase in cost or lower revenue. On this basis, it is recommended to continue exploration efforts to the east and to the west of the current resource. It is also recommended to invest in the access drive as early as possible to allow the exploration program to reach the deepest area of the deposit as well as permit geotechnical hydro-geological and structural data collection which will be essential for further study work.

18.8 Project Risk

The current significant risks to the Project are considered to be:

- The Resource risk has the potential to have the greatest effect on the viability of the Project. Although the mineralization appears to have reasonable continuity, the interpretation of the lenses can affect the dip of the stopes which has an impact on the choice of the mining method. On the other hand, the mineralization is open in 3 directions (east, west and depth) and this represents significant upside to the Project.
- Geotechnical aspects of the design, in particular the rock mass rating evaluation, is based on limited data. The visit to the underground workings of the local artisanal miners tended to present a more positive outlook of the rock mass. However, for the purpose of the study the geotechnical aspect is conservative.
- The operational risks for underground mining are reduced by the simplicity of the type of operation, the main concern is the geological ability to follow the lenses in the development phase or grade control.
- The Project has moderate to slight cost risk. A 20% increase in operating costs would reduce the Project cash flow by approximately 30%.
- The Project has moderate revenue risk. A reduction of revenue by 15%, which could be due to either a grade or metal price shortfall, indicates over 50% reduction in total Project cash flow.
- Adequate surface area, whether for infrastructure construction or disposal of tailing and waste, is critical to the Project.

A formal and thorough risk assessment should be conducted as part of subsequent detailed studies.

19 INTERPRETATION AND CONCLUSIONS

The pertinent observations and interpretations which have been developed in producing this report are detailed in the sections above.

20 RECOMMENDATIONS

The following recommendations are made for the next phases of the Project:

20.1 Studies

- As the resource is only of Inferred category, it will need to be brought to a higher level of confidence, i.e. Measured or Indicated, before an Ore Reserve can be reported.
- It is recommended that a future study optimises the mining method selection with more detailed geotechnical input. Geotechnical considerations will also influence the development cost as ground support is an important part of the cost and the decisive factor for the rate of development.
- A more thorough study for the tailings storage facility (TSF) including preliminary water balance, hydrogeological, geotechnical and geo-chemical reviews should be undertaken. Closure issues will need to be examined as part of any further studies. This is particularly important as the tailings could be Potentially Acid Forming (PAF).

20.2 Testwork

- Undertake slurry characterisation, waste and tailings testing. Based on the results evaluate the suitability of the tailings for use as pastefill or hydraulic fill.
- Carryout metallurgical comminution testwork to establish the relationship between grind size, gravity recovery and overall circuit recovery.
- Determine the amount of gravity recoverable gold so that improved CIL modelling can be carried out.
- Conduct flotation testwork with and without gravity recovery and regrind to try to maximise gold recovery and minimise capital expenditure.
- Determine the settling and filtration rate parameters of appropriate slurry streams.

20.3 Budget and Schedule

MIRL has total budget of $12.3M in 2010 and $10.0M in 2011 excluding vendor payments for the Project. Incorporated in this budget is expenditure on studies of $6.8M in 2010 and $4.8M in 2011, which includes drilling to increase resource confidence, all the required test work and the completion of an access drive. This budget will allow MIRL to complete a Prefeasibility Study in 2010 and finalise a Bankable Feasibility Study by the end of year 2011. Also included in the total budget is expenditure of $2.8M in 2010 and $2.7M in 2011 on exploration and associated drilling. This exploration is well justified considering the exploration potential of the Project. Coffey Mining believes that the level of funding budgeted and schedule proposed by MIRL are appropriate to reach these objectives.

21 REFERENCES

Coffey Mining (RSG Global) - Yeates, R. et al., 2007, Competent Persons Report – April 2007, Project Number PINV01, 1162 Hay Street, West Perth 6005 Australia.

Coffey Mining(November 2009) - Technical Report, Project Number MineWPer00466AC, 1162 Hay Street, West Perth 6005 Australia.

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

Telluris Consulting Ltd. (September 2009) – Structural Field Study of the Ollachea District

22 DATE AND SIGNATURE PAGE

The effective date of this Report is 6th April 2010

[signed]

B Nicholls B.Sc Geol. MAIG
Geology Manager - Brazil
Coffey Mining Pty Ltd

6th April 2010

[signed]

Jean-François St-Onge eng. B.Sc.A. (Mining), MAusIMM
Specialist Mining Engineer
Coffey Mining Pty Ltd

6th April 2010

[signed]

B Viana BSc (Geol), MAIG
Resource Geologist
Coffey Mining Pty Ltd

6th April 2010

[signed]

Barry Cloutt BAppSc (Eng Met) MAusIMM
Chief Metallurgist
Coffey Mining Pty Ltd

6th April 2010

23 CERTIFICATES OF AUTHORS

Certificate of Qualified Person

As the primary author of the report entitled "Ollachea Gold Project, Technical Report" (the Report), dated 6th April 2010, I hereby state:

1. I, Beau Nicholls, Consulting Geologist of Coffey Mining Pty Ltd, 1162 Hay Street, West Perth, Western Australia, Australia, do hereby certify that:
2. I am a practising geologist with 15 years of mining and exploration experience. I have worked in Australia, Eastern Europe, West Africa and currently Brazil. I am a member of the Australian Institute of Geoscientists ("MAIG").
3. I am a graduate of Western Australian School of Mines – Kalgoorlie and hold a Bachelor of Science Degree in Mineral Exploration and Mining Geology (1994). I have practiced my profession continuously since 1995.
4. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
5. I visited the property that is the subject of this Report between the 7th and 10th May 2009.
6. I am responsible for all sections of this Report (excluding Sections 16, 17 and 18).
7. I hereby consent to the use of this Report and my name in the preparation of documents for a public filing including a prospectus, an annual information filing,, brokered or non-brokered financing(s), or for the submission to any Provincial or Federal regulatory authority.
8. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, or the omission to disclose which makes the Report misleading and that as of the date of this certificate, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.
9. I have read and understand National Instrument 43-101 and am independent of the issuer as defined in Section 1.4 and prior to visiting Ollachea I had no involvement in or knowledge of the property that is the subject of this Report.
10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Report has been prepared in compliance with the Instrument and the Form.
11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Property that is the subject of this report and do not hold nor expect to receive securities of Minera IRL Limited.

Dated at Perth, Western Australia, Australia, on 6th April 2010.

[signed]

Beau Nicholls B.Sc Geol MAIG
Geology Manager - Brazil
Coffey Mining Pty Ltd

Certificate of Qualified Person

As co-author of the report entitled "Ollachea Gold Project, Technical Report" (the Report), dated 6th April 2010, I hereby state:

1. I, Jean-Francois St-Onge eng., Employee and Specialist Mining Engineer of Coffey Mining Pty Ltd, 1162 Hay Street, West Perth, Western Australia, Australia, do hereby certify that:
2. I am a non-resident member of the OIQ (Ordre des Ingénieurs du Québec - 111717) as well as a member of the AusIMM (Australasian Institute of Mining and Metallurgy), and a 'Qualified Person' in relation to the subject matter of this report. .
3. I graduated from the University Laval, Québec, Qc, Canada with a B.Sc.A. (Mining) Degree in 1992. I have practiced my profession continuously since then.
4. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
5. I visited the property that is the subject of this Report between the 7th and 10th May 2009.
6. I am responsible for Sections 1, 17.2 and 18 of this report.
7. I hereby consent to the use of this Report and my name in the preparation of documents for a public filing including a prospectus, an annual information filing,, brokered or non-brokered financing(s), or for the submission to any Provincial or Federal regulatory authority.
8. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, or the omission to disclose which makes the Report misleading and that as of the date of this certificate, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.
9. I have read and understand National Instrument 43-101 and am considered independent of the issuer as defined in Section 1.4.
10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Report has been prepared in compliance with the Instrument and the Form.
11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Property that is the subject of this Report and do not hold nor expect to receive securities of Minera IRL Limited.

Dated at Perth, Western Australia, Australia, on 6th April 2010.

[signed]

Jean-François St-Onge eng. B.Sc.A. (Mining), MAusIMM
Specialist Mining Engineer
Coffey Mining Pty Ltd

Certificate of Qualified Person

As co-author of the report entitled "Ollachea Gold Project, Technical Report" (the Report), dated 6th April 2010, I hereby state:

1. I, Barry Cloutt, am the Chief Metallurgist with the firm Coffey Mining Pty Ltd, 1162 Hay Street, West Perth, Western Australia, Australia, do hereby certify that:
2. I am a practising metallurgist and I am a Member of AusIMM (Australasian Institute of Mining and Metallurgy).
3. I am a graduate of the Western Australian Institute of Technology (Curtin University and hold a Bachelor of Applied Science (Engineering Metallurgy) degree 1981. I have practiced my profession continuously since 1982.
4. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
5. I have not visited the property that is the subject of this Report.
6. I am responsible for Section and 16 of this report.
7. I am co responsible for Section 1 of this report.
8. I hereby consent to the use of this Report and my name in the preparation of documents for a public filing including a prospectus, an annual information filing, brokered or non-brokered financing(s), or for the submission to any Provincial or Federal regulatory authority.
9. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, or the omission to disclose which makes the Report misleading and that as of the date of this certificate, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.
10. I have read and understand National Instrument 43-101 and am considered independent of the issuer as defined in Section 1.4.
11. I have read the National Instrument and Form 43-101F1 (the "Form") and the Report has been prepared in compliance with the Instrument and the Form.
12. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Properties that are the subject of this Report and do not hold nor expect to receive securities of Minera IRL Limited.

Dated at Perth, Western Australia, Australia, on 6th April 2010.

[signed]

Barry Cloutt BAppSc (Eng Met) MAusIMM
Chief Metallurgist
Coffey Mining Pty Ltd

Certificate of Qualified Person

As author of the report entitled "Ollachea Gold Project, Technical Report" (the Report), dated 6th April 2010, I hereby state:

1. I, Bernardo Viana, am a Resource Consultant with the firm Coffey Mining Pty Ltd, 1162 Hay Street, West Perth, Western Australia, Australia, do hereby certify that:
2. I am a practising geologist with 8 years of Resource Estimation and Exploration geological experience. I am a member of the Australian Institute of Geoscientists ("MAIG")
3. I am a graduate of Federal University of Minas Gerais – Belo Horizonte and hold a Bachelor of Science Degree in Geology (2002). I have practiced my profession continuously since 2002.
4. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
5. I have not visited the property that is the subject of this Report.
6. I have prepared and take responsibility for Section 17.1 of this report.
7. I hereby consent to the use of this Report and my name in the preparation of documents for a public filing including a prospectus, an annual information filing,, brokered or non-brokered financing(s), or for the submission to any Provincial or Federal regulatory authority.
8. I am not aware of any material fact or material change with respect to the subject matter of the Report that is not reflected in the Report, or the omission to disclose which makes the Report misleading and that as of the date of this certificate, to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.
9. I have read and understand National Instrument 43-101 and am considered independent of the issuer as defined in Section 1.4.
10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Report has been prepared in compliance with the Instrument and the Form.
11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Properties that are the subject of this report and do not hold nor expect to receive securities of Minera IRL Limited.

Dated at Perth, Western Australia, Australia, on 6th April 2010.

[signed]

Bernardo Viana BSc Geology MAIG
Resource Manager - Brazil
Coffey Mining Pty Ltd