EXECUTIVE SUMMARY

Project Overview

This scoping study report, prepared by Coffey Mining, presents a preliminary evaluation for the development of the Ollachea Gold Project (the Project) in the Puno region of Peru. The scoping study is based on the use of underground mining by sub-level long-hole stoping and a carbon-in-leach processing plant to produce an average of over 100,000 ounces of gold per year for a mine life of nine years.

Minera Kuri Kullu S.A. (MKK), a fully owned subsidiary of Minera IRL S.A. (Minera), has the right to acquire 100% of the Project (the Project) from Rio Tinto plc (Rio) pursuant to an option agreement dated 1 September 2006. Minera IRL S.A. is a wholly owned subsidiary of Minera IRL Limited (MIRL).

Coffey Mining has completed a scoping study of the Project including an NI43-101 compliant resource estimate and has been able to indicate the following key outcomes:

- This scoping study is based on the central Minapampa Zone of the Project. Mineralisation has not been closed off along strike nor down dip and this is subject to ongoing exploration drilling.
- Based on the results of 15km of diamond drilling carried out to 11 September 2009, the resource, of inferred category, has been estimated to be contained in 7 lenses totalling 8.9Mt at an average of 4.5g/t of gold using a 2.5g/t cut-off grade. Contained gold totals 1.3M ounces.
- Based on this resource, a mining inventory of 8.2Mt at 4.0g/t containing 1.1M ounces of gold has been estimated.
- An underground mining method termed “long-hole, sub-level stoping with fill” has been selected as the most appropriate method.
- The geotechnical assessment of the deposit waste and mineralised rock estimates the rock mass to be weak. The use of waste rock in combination with paste fill from filtered tailings has therefore been incorporated into the mining method to allow for greater ore recovery.
- The mine will be accessed through a 1.3km 5m by 5m drive. The operation will be fully mechanised and has been design to produce 1.0Mt per year.
- Metallurgical testwork has indicated that conventional gravity and CIL techniques are able to recover approximately 91% of the gold. The testwork also indicated that a simple 3 stage crushing, single stage milling, gravity and CIL circuit could be utilised to treat 1Mtpa of feed material.
- Three options for tailings deposition were studied. It was established that dry stacking of filtered process plant tailings is likely to be the most cost and environmentally effective method to be used for the Project. It is estimated that approximately half the tailings generated will be used for underground backfill.

- The operation will provide direct employment for approximately 380 people.
- Financial analysis of the Project has established the outcomes shown in Table 1 based upon a gold price of US$850 per ounce.

Table 1
Ollachea Gold Project
Project Financial Highlights

Capex (2009$)	US$156.8M
Opex (2009$)	US$46.02/t
LOM Cash flow pre tax (real)	US$221.0M
LOM Cash flow post tax (real)	US$147.7.0M
NPV @ 8% Real (pre tax)	US$102.5M
NPV @ 8% Real (post tax)	US$58.7M
IRR pre tax (real)	22.4%
IRR post tax (real)	17.4%
Payback (pre tax)	3.7 years
Payback (post tax)	4.0 years

Introduction

The Project is located in the Ollachea District of Carabaya Province in the Puno Region of south-eastern Peru. The Ollachea village, 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 and most of the workforce, more than 100 employees during the exploration phase, is sourced from Ollachea.

Ollachea is close to major infrastructure such as the airport in Juliaca, 240km south-southwest of the Project, and the San Gaban hydroelectric complex, 43km north-northwest of the Project. This complex connects directly to the national grid, which passes directly across the Project. Fresh water is available from the Ollachea River, a major melt-water drainage that flows immediately east of the Ollachea village, anticipated to provide an adequate water supply for any future mining and processing activities. The new Southern Interoceanic Highway between the Atlantic and the Pacific Oceans passes through the Ollachea village.

Resource

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 mineralised interpretations that were created with the assistance of MKK geologists.

The Ollachea resource estimate is based on diamond core (DC) drilling. The estimate contains 49 DC holes totalling 15km.

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

The resource estimate for Ollachea has been classified as an NI43-101 compliant Inferred Mineral Resource based on the confidence levels of the key criteria that were considered during the resource estimation. Table 2 presents the grade tonnage report estimated as of the 6th of October 2009.

For the purpose of the scoping study, the mining inventory was estimated to be 8.2Mt at 4g/t head grade as presented in Table 3.

Table 2

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 Cut-off 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

Table 3

Ollachea Gold Project

Minable Tonnes Estimate

Items	Units
Resources tonnes	8.9M
Resource grade @ 2.5g/t Au COG	4.5
Resource Oz Au	1.3M
Mining Recovery	80%
Dilution	15%
Mining Inventory tonnes	8.2M
Dilution grade g/t Au	0.9
Head grade g/t Au	4.0
Gold ounces feed to plant	1.1M
Gold Recovery	91%
Recovered Ounces Au	1.0M

Geotechnical

The weighted 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 (L) and 26m in height (H) along the dip could be considered to be stable subject to the application of cable bolting to the exposures.

Mining

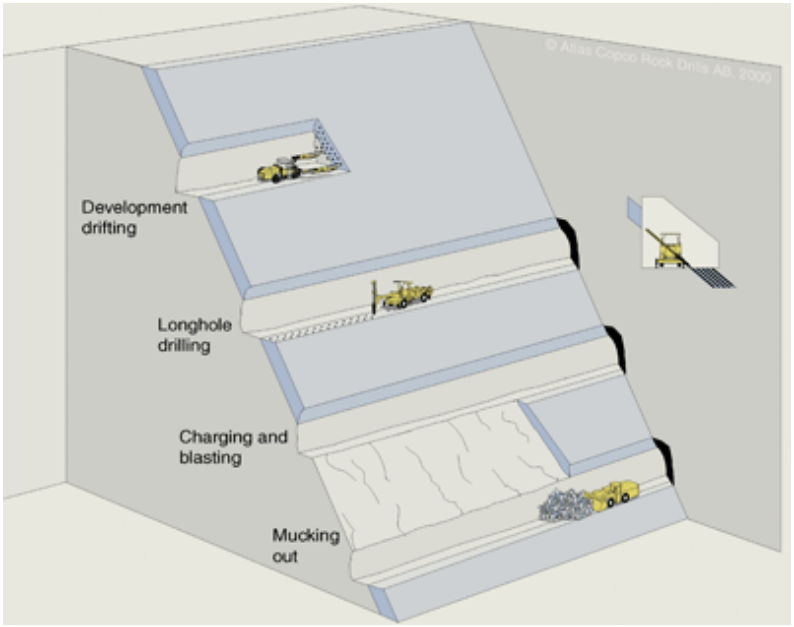
As the basis of the shapes for selecting the mining method, the mineralisation envelopes created at a cut-off grade of 1.0g/t of gold were used.

The Resource extends in the east-west direction about 600m and is still open at both ends. 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. The lenses dip at an average of 50° to 55° to the north. The thickness is irregular and varies from 2m to more than 25m, in some areas; the average thickness is estimated to be 7m.

The mining method selected for the current study is sublevel stoping in a narrow vein setting as presented in Figure 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 geotechnical aspect is the current limiting factor for stope size.

Figure 1
Typical View of the Selected Mining Method

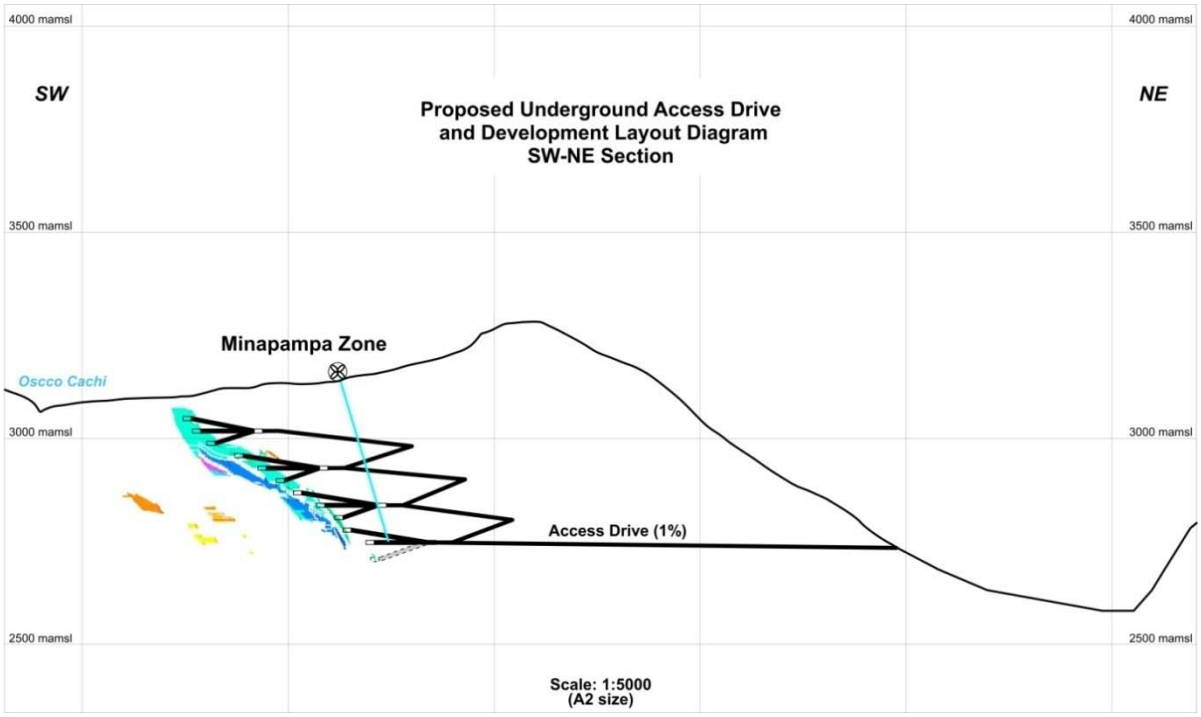


The mountainous area of Ollachea provides the opportunity to access the mine by means of an access drive about 1.3km long from the proposed plant site situated in an adjoining valley, through the mountain located towards the north. Figure 2 presents a sketch of the access drive. This access drive will 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.

The development schedule has been based on the use of 3 jumbos at an advance rate of 120m/month. Bolting is carried out with 2 mechanised bolters and 2 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 production will be achieved with 3 loaders with a rated payload of 17t and 5 trucks with a rated payload at 45t. The number of units takes into account the requirement for development loading and trucking. The average size of the stopes is 8,000t which make the average number of stopes required per month, about 8 stopes.

Figure 2
Access Drive



Other significant aspects related to mining include:

- The ventilation estimation methods used indicate airflow between 264m³/s and 313m³/s with an average of 290m³/s. A simple average of the 4 methods has been used for the scoping study. Ventilation will be supplied through 3 main fans in a push-pull arrangement, one pushing and two pulling air from opposite ends of the deposit. The main access drive will also serve as an intake source.
- Water inflow at the Project is expected to be high. The mineralised zone lies below a stream that crosses the property and groundwater for the Ollachea village is sourced from near the mine site. This indicates high potential for groundwater inflow to the mining area. The mine design includes two pumping stations, one at the bottom of the mineralised zone to lift the water to 2,700mRL where the second pumping station will serve to transport the water out of the mine through the access drive.
- Underground infrastructure will be minimised, as ground conditions are not favourable to the excavation of large workshops or other rooms. 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; most vehicles will be driven to surface at the end of each shift to be serviced and the less mobile equipment, such as drilling equipment, will be supplied via a service truck during the shifts.

Backfill

An initial assessment of the backfill strength requirements indicated 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.

The most significant challenge to the use of pastefill at this site is reticulation. 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 pumped approximately 1300m horizontally along the access development and vertically though a height of about 300m. An appropriately sized positive displacement pump is necessary for this duty. This technology is relatively expensive but proven.

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.

Most samples leached poorly in cyanide without the addition of activated carbon in the leach. Recoveries ranged from 15% to 79% extraction after leach times varying from 36 to 72 hours. Recoveries were significantly improved when leaching occurred in the presence of activated carbon (CIL). Recoveries ranged from 81% to 95% extraction after 36 hours. The cyanide consumption is moderate to high - 1.3 to 3.0kg/t. Lime consumption is moderate at 0.9 to 1.5kg/t.

One sample with a sulphide sulphur content of 2.7% was floated to gauge its response. The mineralised material floated reasonably well within a range of simple reagent regimes for periods between 13 to 20 minutes of laboratory float time. Gold recoveries varied from 20 to 96% in a mass recovery of 35%. Flotation will be further investigated in the future.

Processing

The chosen base case processing flowsheet consists of 3 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 7 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 4 below:

Table 4
Ollachea Gold Project
Major Design Criteria

Criteria	Unit	
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

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 1.5 km 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 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.

Costs

The estimated capital costs for the Project are summarised in Table 5. Initial capital totals US$156.8M including a contingency of US$26.1M. 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 is included on a yearly basis as well as a US$5.0M closure plan allowance at the end of the mine life. No contingencies have been added to the sustaining capital cost and closure cost in the financial model.

Table 5
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**

The operational costs are divided into fixed and variable costs, and include mining, processing and G&A. Table 6 presents a summary of the operational costs.

Table 6
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**

Financial Analysis

The input parameters for the analysis are as follow:

- The mining inventory is estimated to be 8.2Mt at 4g/t head grade for 1.1M contained ounces. The mining and processing rate has been set to 1.0Mtpa with a ramp-up period of 70% during the first year. The processing recovery is estimated at 91.2% for the life of mine.
- Base case metal prices used in the model are US$850 per ounce of gold and US$12 per ounce of silver.
- The life of Project and steady state unit production cost per ounce are summarised in Table 7.
- The financial model includes Peru Government Royalty, a Vendor Royalty, Income tax and Workers' Profit Participation. The Peruvian Taxation System IGV (sales tax) has been excluded due to the activity of the Project. Being export of goods, IGV is assumed to be immediately recoverable, consistent with Peruvian established practice.

Table 7
Ollachea Gold Project
Unit Cost of Production per Ounce
(2009$)

Parameter	Steady State Cost (US$/oz Au)	LOM Average Cost (US$/oz Au)
Mining	188	190
Processing	167	169
G&A	33	35
Total Site Operating Costs	**388**	**393**
Refinery Charge	6	6
Silver credit	(0.4)	(0.4)
Mine Cash Operating Cost	**394**	**399**
Royalties	20	20
Total Production Costs	**414**	**419**

Results of financial analysis include:

- The pre-tax (including Workers' Profit Participation) and post-tax results of the financial analysis are summarised in Table 8 and Figure 3. The financial analysis shows promising returns for the Project.
- Sensitivity analysis was carried out on gold price and gold head grade, operating cost and capital cost as well as minable tonnes and throughput.
- 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 up-and-running, the Project is cash cost positive above $400 per ounce. Table 10 illustrates the effect on cash flow, NPV and IRR for a range of gold prices from $700/oz to $1,200 per ounce.

- 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 minable inventory and throughput is the least important of the items analysed. Current drilling by MKK outside the limits of the Minapampa mineralised zone has indicated the potential for additional resources. Table 9 shows the Project returns based on an additional 2.0Mt at a gold grade of 4.0g/t, containing 257,000 ounces or nearly 25% increase in resource.

Table 8
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 3
Project Cash Flow After Tax

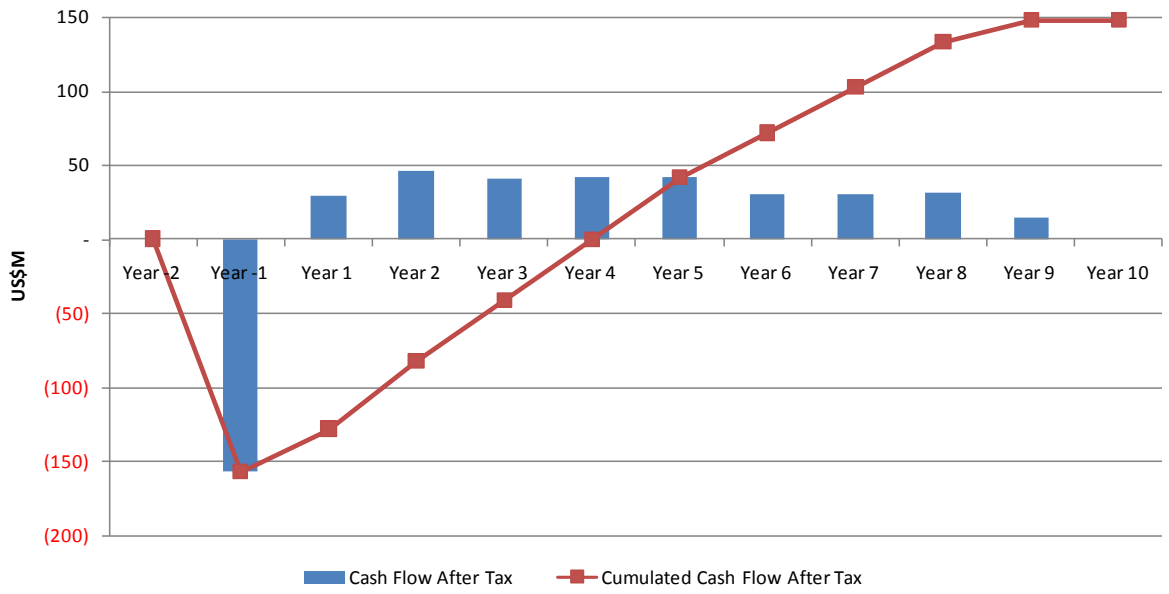


Table 9
Ollachea Gold Project
Project IRR and NPV with additional 2Mt at 4.0g/t Au

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

Table 10
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

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.

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 before an ore reserve can be reported and this will require measured or indicated resources.
  - It is recommended that a future study optimise the mining method selection with more detailed geotechnical input to make the most appropriate choice. This 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 performed as part of the PFS. Closure issues will need to be examined as part of a PFS. 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.
  - 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.